BALKANMINE 2009

3rd BALKAN MINING CONGRESS
3. BALKAN MADENCİLİK KONGRESİ

October 1-3, 2009
İzmir-TURKEY

This Congress is supported by TÜBİTAK (The Scientific and Technological Research Council of Turkey)

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UCTEA
The Chamber of Mining Engineers of Turkey

Published by Gurup Matbaacılık, Ankara, Turkey
Phone: +90 (312) 384 73 44 Fax: +90 (312) 384 73 46

Puplication No. : 160
ISBN : 978-9944-89-782-2

Adress: Selanik Cad. 19/3 06650 - Kızılay, Ankara - TURKEY
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SUNUŞ

Odamız, gerek bilimsel ve teknik bilginin paylaşılması, gerekse ulusal ve evrensel meslek ilkeleri ve sorumlulukları temelinde uluslararası dayanışma amacıyla, diğer ülkelerin meslek örgütleriyle ve üniversiteleriyle iletişim içerisinde bulunmayı önemsemektedir.

Bu çerçevede söz konusu örgütlenmeler ile iletişime geçilmesi hususunda çalışmalar başlatılmıştır. Bu doğrultuda birincisi 2005 yılında Sofya'da, ikincisi 2007 yılında Belgrat'ta düzenlenen Balkan Madencilik Kongresi'nin üçüncüsü Odamızın ev sahipliğiyle ülkemizde düzenlenmektedir.

Dünyada bilim ve teknoloji alanında çok hızlı bir gelişim ve değişim süreci yaşanmaktadır. Üretilen bilginin her 2-3 yılda ikiye katlandığı ileri sürülmektedir. Bilime ve teknolojiye hakim olan güçler dünyayı da egemenlikleri altına almaktadır. Bu nedenle gelişmiş ülkeler bütçelerinden mühendislik- bilim teknoloji ve eğitim alt yapısına ayırdıkları payı gün geçtikçe arttırmaktadır.

Madencilik sektöründe aramadan uç ürüne kadar her aşamada ileri teknoloji kullanılmalıdır. Üretim ve kaynak performansının iyileştirilmesine ve yeni ürünlerin elde edilmesine yönelik olarak yeni gelişen teknolojilerin kullanımı, bu sektörün ülke kalkınmasına katkısı bakımından son derece önemlidir. Bu nedenle sektörde yüksek teknoloji kullanımı ve üretilmesine yönelik araştırma-geliştirme çalışmalarına öncelik verilmelidir. İleri üretim teknolojilerinin geliştirilmesi ve kullanımı, daha temiz ve daha etkin madencilik süreç ve ürünlerinin temini bakımından önkoşuldur.

Bu Kongre'de sektördeki teknolojik gelişmeler paylaşılırken, ülkemizin madencilik sektörünün tanıtımı da yapılacaktır. Balkan ülkelerindeki maden mühendislerinin ve yerbilimcilerin bir araya geleceği toplantılar da deneyimlerin ve teknik bilginin paylaşımı amaçlanmıştır.

Kongre'nin gerçekleşmesine katkı koyan Yürütme Kurulu Başkanı Bahtiyar ÜNVER başta olmak üzere tüm Yürütme Kurulu üyelerine, Balkan Madencilik Kongresi Koordinasyon Kurulu Üyesi Tevfik GÜYAGÜLER'e ve emeği geçen herkese teşekkür ederiz.

Saygılarımızla

YÖNETİM KURULU

FOREWORD

The 3rd Balkan Mining Congress (BALKANMINE 2009) organized by Balkan Mining Association, BALKANMINE and The Chamber of Mining Engineers of Turkey is held between October 1-3 in İzmir, Turkey. The primary objective of the Congress is to promote operational, economical and scientific information pertaining to all aspects of mining technology, energy and sustainable development.

In conjunction with the Congress, 3rd Mining, Natural Resources and Technology Fair of Turkey, MINEX 2009 is organized at the same location for the exhibition of mining products together with companies offering machinery, equipment, instruments, software and services to mining, processing and energy industries.

The papers included in the proceedings volume have been grouped under ten specific themes including, Balkan Mining Industry; Mineral Resources and Mine Geology; Exploitation; Process Engineering; Rock Engineering and Design; Computer Applications in Mining and Processing; Management and Mining Economics; Ventilation and Safety; Mining and Environment; History and Mine Education. The 98 papers included in this volume have been prepared by authors from 14 countries. I am confident to state that papers included in this proceedings volume are testimonials to the vibrant role that mining technology plays in the identification and establishment of routes to sustainable resource development, environmental protection and globalization.

Every successful congress stems from a teamwork approach. We owe gratitude to the members of the Organizing Committee, Balkan Mining Association Coordination Committee, Executive Board of the Chamber of Mining Engineers of Turkey and Chairpersons of the technical sessions. There is no need to mention that this proceedings volume and the BALKANMINE 2009 would not come to reality without contributions of the speakers and authors. Our most genuine appreciation also goes to the delegates for their interest and contributions to the success of the Congress.

We acknowledge with gratitude the financial support provided by TÜBİTAK, The Scientific and Technological Research Council of Turkey. We also owe gratitude to İZFAŞ, İzmir Fair Services Culture and Art Affairs Trade Inc., for their professional effort in the preparation of the Congress venue.

Once again, I thank all of the participants of the BALKANMINE 2009 for their contributions which will become instrumental in the enhancement of our scientific and professional development. I am delighted to reiterate that it is a great pleasure for me to welcome all friends and colleagues to İzmir, to a congress that you will find technically stimulating and socially enjoyable.

Dr. Bahtiyar ÜNVER

for the Organizing Committee

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Border Area Deposits and Possibilities for Joint Projects

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ABSTRACT: There are a number of different deposits as per the Bulgarian Classification in accordance with the Bulgarian Law on Ore and Mineral Resources. Most of them have been explored and assessed. Some of the deposits have been mined by way of underground or open pit mining methods up to regulated distances up to the state borders. The industrial infrastructure rehabilitation and the potential of the acquired Bulgarian experience can be used for the future mining work along with the remaining parts of these deposits.

1 POSSIBILITIES

The countries on the Balkan Peninsula may establish a common mineral and ore base of raw materials.

The successful solution to this issue is determined by:

- the fact that mineral raw materials as well as these used for the purpose of energy and construction located in the subsoil layers within the border regions of Bulgaria, Turkey, Greece, Macedonia, Serbia and Romania, in their volume and quality have occupied a certain position in their national economies;
- the experience in the exploration of deposits, the achievements in the theory and practice related to the extraction and processing of mineral raw materials;
- the nature of the overwhelming globalization requiring the drafting of legislative and regulatory provision for a joint construction of industrial and domestic infrastructure and ecologically complied extraction and processing of raw materials;
- availability of facilities for primary processing and the related waste depots having clear and legitimated location and status;
- variety of deposits for raw materials of a different nature;
- the trend of extension and streamlining of the privatization forms in the countries from the former Eastern Block (Romania and Bulgaria);
- normal climatic and natural conditions.

The competitiveness of underground raw materials depends on natural factors (the existence of underground mineral resources and ores) and a complete and qualitative geological assessment of the subsoil resources.

The state borders are not consistent with the borders of the underground mining mineral resources and ores. During the Cold War period the countries on the Balkan Peninsula were influenced by three political, economic and social systems. The sources of geological provision in the exploration of the subsoil resources in the recent past were subjected to two schools. Nevertheless, the personal contacts between geologists, mining technologists, ore-dressing experts and mine surveyors maintained the level of professional and scientific ethics and exchange at the forums organized by the Balkan countries. The wide spectrum of geochemical, geophysical and distance methods as well as the exchange between representatives of the two schools recently, as regards the so-called closed regions (border zones) is a favorable factor in view of the qualitative research of the structures and the improvement of the deposits' assessment.

The methods and means for achievement of results by way of mathematical modeling and computer processing of the initial information, its generalization and interpretation at the contemporary stage, are considerably relieved in terms of hardware and software.

In the world practice, the approach for exploration of the subsoil resources starting from an assessment of a separate type of deposit and finishing with a complex assessment of the area as regards all the resources therein, is possible and appropriate as regards the production potential of the region. The growth of transport costs necessitates reducing the distance from the extraction location to the location where the processing infrastructure is built or where it would be appropriate for such to be built, i.e., a significant economic benefit from the situation of the unit "extraction - processing" in neighboring countries. In addition to the obtained benefit, if we add the favorable transport costs and legal and regulatory standards (criteria), then the profit for the concessionaire who has been granted the concession for extraction of mineral resources and ores will be considerable.

Hindering conditions in respect of the establishment of a Balkan inter-state raw material base may prove the following:

- the difference in the laws and regulations on obtaining a permit for exploration and concession for extraction of mineral resources (in the meaning of Bulgarian legislation);
- level and practice of the mining ecology and environment protection in the neighboring countries during the processes of geological exploration of the subsoil resources, extraction and initial processing of mineral raw materials and raw materials used in construction;
- communications and logistics between the neighboring countries, at this stage providing only tourism and communications exchange, but not transport of cargoes related to the development of deposits and processing of raw materials, i.e., a restriction of the unit "extraction - processing" if there is a necessity for it to be located on the territory of neighboring countries;
- differences in the motivation and professional attitude of the population on the two sides of the border as regards the development of mineral resource deposits;
- difference in the technological and economic assessment of the classification of the reserves and resources of minerals and ores.

To the presented list of negative conditions at this stage of development there may be added also the following:

- the crisis in the demand for products made from mineral raw materials and raw materials intended for the purposes of construction, ensuing from the worldwide financial and economic crisis;
- difference in the strategy of development of the national economies in the individual countries, conditioned by the peculiarities of the "World Order", and
- difference in the economic and political affiliation of the countries on the Balkan Peninsula.

The cooperating (associating) of the experience and efforts of experts in the field of discovery and research of mineral resources and ores, mining geology, mining technologies, computer technologies, and on the other hand, of businessmen oriented towards the development of deposits and the processing of raw materials is the way to improve the prospect for utilization of the deposits located in these regions. Of paramount importance is the establishment of single regulatory and methodical standards for protection of the environment (including the subsoil resources) on the territory of the neighboring countries.

The assessment of the subsoil resources in the borderline regions may be effected in the following directions:

- ➢ reassessment of known sites for extraction of mineral resources and ores situated in neighboring countries (given a different degree of research made and development of the deposit, but based on a marketing research proving a considerable benefit from the obtained raw material);
- ➢ reassessment of weakly explored deposit flanks, due to the restrictive measures of the political systems;
- ➢ forecast and exploration of new (most frequently deeply laying (metalogenic) deposits.

The activity under the first direction may be oriented towards:

- clarification of the spectrum of the elements that are admixtures of the main mineral resource, for instance, copper ore and lithium, gold, solver, platinum, etc., the demand for which is persisting even in times of crises and having broad application in nanotechnologies;
- development of computer models of the deposit based on the information from the thoroughly explored part, allowing for a flexible reassessment under variable economic conditions;
- elaboration of technologies for primary processing of raw materials ensuring the maximum extraction of the components being demanded on the market, the minimum environmental pollution, etc.

The second and third direction within the territory of the Republic of Bulgaria can be successfully developed in respect of several regions – Malko Tarnovo, Elhovo, Ermorechen, etc.

The efficiency of the work along these two directions is based on contemporary knowledge concerning the objective laws of natural accumulations (of ores, most frequently) on different scale levels within the earth's crust and the availability of technologies (axiomatic formalization of geological environment, computer modeling) in view of discovering hidden deposits.

2 GEOLOGICAL AND GEOGRAPHICAL DATA ABOUT THE DEPOSITS

The deposits of mineral resources and ores in the borderline regions on the territory of the Republic of Bulgaria are located in the mountainous massifs of Strandja, Sakar, Southern Rhodope, the Mount of Osogovo and the Balkan. In terms of variety they are metal, rock-facing, construction and energy-related. Their research in the regions of Malko Tarnovo, Ustrem, Zlatograd, Petrich – Gotse Delchev, Gyueshevo and Berkovitsa has reached a different degree. The geological mapping of the specified regions is different, and details as to its stages are specified in Figure 1.

Basically within the territory of the Republic of Bulgaria are developed borderline metal deposits and such for rock-facing and building materials (sand, gravels, limestone, etc.). The extraction from the various metalogenic deposits in terms of volume and interests is characterized by the following quantitative dimensions:

Strandja Region. During the period 1957-1996 its yield amounted to over 7 million tons of copper ores having copper contents of 0.96%, extracted from the deposits "Mladenovo" (1959-1984), "Propada" (1957-1996), "Bardtse" (1960-1996) and "Gramatikovo" (1960-1991). Under the resource category with the National balance are entered, accordingly, the following: – "Propada" - over 111,000 tons with a 1.0% copper content, "Bardtse"- 29 million tons having 0.32% copper content. The ores from Bardtse deposit are extracted via two classical methods. There is a stated interest on the part of Bardtse deposit in obtaining a research permit.

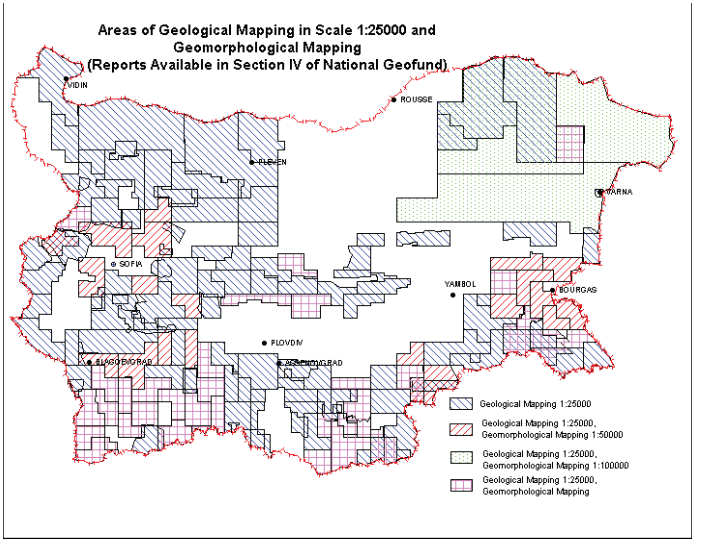

Figure 1. Map of the geological research coverage in the Republic of Bulgaria.

The region is rich in building and rock-facing materials having industrial significance. According to engineer L. Kovachev (eng. L. Kovachev – director of mining and dressing enterprise in the region of Malko Tarnovo) there is an experience and traditions in the region as regards the development mineral resource deposits. Along with the copper and gold ores there are possibilities for the extraction of black and pink marble, inert materials and limestone.

Elhovo Region – **Ustrem Mining Field.** The yield during the period 1952-1995 exceeded 8 million tons of ores with an average content of lead equaling 1.58 % and zinc - 0.46 %. The available resources identified in advance, as per the Classification of reserves/ resources in Bulgaria, exceed 1 million tons. To date no interests as regards the deposit have been stated.

According to engineer A. Mirchev (eng. A. Mirchev – former head of the mine "Lesovo") in the region around the deposit of Lesovo there are favorable conditions for the development of lead-zinc ores, and the existence of deposits of rock-facing and building materials condition business interests.

Madjarovo Mining Field. From the Madjarovo Mining Field are extracted about 10 million tons of lead-zinc ore during the period 1957-1996 with an average content of lead of 1.27 % and of zinc 0.66 %, accordingly. The deposit has been developed by employing the underground mining method. Along with the lead-zinc ore, out of the specified ore field there have been extracted over 300,000 tons of gold ore having average gold content of about 2 g./t and silver content of 19.7 g/t. Under the "resource" category in the national balance of reserves/resources, the volume of the lead-zinc ore extracted from the field exceeds 12 million tons with average content of lead, zinc and copper: 1.3 %, 1.5 % and 0.4 %, accordingly.

Gold containing ore, having content of gold in the amount of 3.25 gr./t. is about 60,000 tons. The mining and geological conditions in the deposits are suitable for underground mining extraction. There are declared interests as regards the obtaining of permission for research and concession on extraction.

Zlatograd Region. These are deposits in the farthest southern parts of Madan ore field and they are located in an immediate proximity of the Bulgarian-Greek border. From 1941 to 1996 their yield obtained through the underground mining method equals about 27 MT of ore. After the year 2002 three deposits are provided through concessions for extraction to Gorubso-Zlatograd AD – town of Zlatograd. Since 2003 about 1 million tons of ore has been extracted. In the deposits within the region the metalogenes fall within the category of resources. There is a current interest expressed in respect of three deposits in the neighboring Madan-Rudozem region.

Osogovo ore region. Two individual deposits are situated in the region – "Lebnitsa" and "Ruen", as the yield of lead and zinc ores from the two deposits is in the amount of about 5 MT. The natural conditions have predetermined the underground mining method for development. The volumes in the "reserves" category are about 1.8 MT, and in the "resources" category – about 25 MT. There is an interest as regards the obtaining of concession for extraction from "Ruen" deposit.

Berkovitsa ore region. In the region metalogenes originate from ferrous ores, lead ores, silver-containing ores and gold-containing ores. Till the year 1995 from the first and second type of ores there have been extracted about 5 million tons, and from the third – about 1 million tons of ore having an average gold content of 3.5 gr/t. The first and second types of ores in the region are entered in the National balance as having approximately 1 million tons of resources each.

3 TECHNOLOGICAL RESULTS FROM THE EXTRACTION AND PROCESSING

The extraction of ores from the deposits located within the borderline regions of Bulgaria, as it has already been mentioned, is effected by employing the underground mining method, and of the rock- facing and inert building materials – by employing the open-pit mining method.

The schemes for opening the deposits developed following the underground mining method have necessitated the excavation of vertical pits and adits, and as regards the opening of deposits by employing the open-pit mining method – most frequently by employing a central capital trench. With the closing of the individual technological units – mines, the opening developments are liquidated and the surrounding environment has been recultivated. An exception to said liquidation are the mines in the Erma river region.

In the underground mining method of development there have been used the systems involving shrinkage, caving in of the immediate roof, dry and hydro-filling, breaking out of sub-floor galleries, chamber, shield-like, etc. As explosive substances there have mainly been used Ammonite and Gelamon. The electrical schemes of exploding are applied within a wave and order system. The intra-block delivery of the broken of ore is effected through scrapers. The productiveness of the applied systems is as follows: shrinkage system – 10 m^3/h.cm., chamber system – 15 m^3/h.cm, caving in of layers – 3÷5 m^3/h.cm., systems with filling – 3÷5 m^3/h.cm. The used mining technology, mechanization and organization have ensured rhythmical extraction without violation of the regulatory standards for protection of subsoil resources, but with certain compromises concerning environment protection. The specified problem for the region where mining activities are being carried out, is eliminated.

The initial processing of ores extracted in the borderline regions is realized in 6 ore-dressing plants. The volume of ores processed in the latter throughout the period from 1955 to 1995 equals over 60 MT. The built as per Soviet designs in the beginning of the 1960s ore-dressing plants and technological schemes of dressing are closed to date and/or used to a limited extent. The infrastructure, roads and waste repositories of the plants are with a regulated status, as most of the latter are recultivated.

The obtained technological parameters of the individual ore-dressing plants are as follows:

Ore-dressing plant "Malko Tarnovo", designed for the processing of copper – pyrite ores with an average copper content of about 1 %, average extraction value of 90 %, average value of the residual copper – below 1 %. A magnet separation is applied in order to obtain ferrous concentrate with a ferrous content of 55 %. The ore-dressing plant is closed down, and the equipment has been dismantled.

Ore-dressing plant "Ustrem", designed for the processing of lead and zinc ores, for which cyanide has been used as reagent-depressor. Currently the plant processes feldspar raw material for the glass industry.

Ore-dressing plant "Erma river", designed for the processing of lead and zinc ore, currently operating, applying a selective-collective scheme of flotation. In the process of dressing is used sodium cyanide as a reagent-depressor. The obtained technological parameters of the plant for extraction of lead and zinc are 88 and 80 %, accordingly, the lead content in the lead concentrate is 68 % and of zinc in the zinc concentrate is 52 %, the content of lead in the waste is 0.2 % and of zinc: 0.4 %. Within the dressing technology, the problem created by the availability of quartz in the zinc concentrate is solved. Currently, the state of the equipment in the plant ensures a regular technological process.

Ore-dressing plant "Rudozem", designed for the processing of lead and zinc ores with technological schemes and reagent regime similar to these of the plant "Erma River". Currently the plant has an annual output of about 200,000 tons of ore, as the designed capacities of the plant are larger.

Ore-dressing plant "Chiprovtsi", designed for the processing of ferrous ores. Currently the plant processes fluorite raw material.

Ore-dressing plant "Gyueshevo", designed for the processing of lead and zinc ores. Currently it is processing metallurgy waste.

The concentrates from the said ore-dressing plants are a raw materials for the lead and zinc plants in the towns of Kardzhali and Asenovgrad and the metallurgy plant in the town of Zlatitsa.

4 RECOMMENDATIONS

The political mosaic of the Balkan Peninsula countries at this stage is characterized as follows:

- countries affiliated to economic unions and military alliances – Greece, Romania and Bulgaria;
- countries affiliated to military alliances only – Turkey, Greece, Romania and Bulgaria;
- politically unaffiliated and not participating jointly in military unions – Macedonia, Serbia and Bulgaria.

The industrial infrastructure within this mosaic in our opinion may be characterized by the following features as well:

- completed metallurgy base for extraction of non-ferrous metals in Romania and Bulgaria;
- completed modern ferrous metallurgy base – Turkey;
- lack of metallurgy industry – Greece, etc.

Given the above, it is logical to make the following recommendations:

1. The traditional contacts related to the exchange of scientific information on the research, development of the deposits and the initial processing of the raw materials extracted from these during the National conferences with international participation, symposia and scientific forums are to be kept and stimulated. The exchange of operational geological information provided through conventional and satellite means must be regulated between the respective neighboring countries. Such regulating is to be dictated by the sole purpose to achieve an even better geological research and qualitative ecologically complied development of the deposits within the borderline regions.
2. The joint development of the deposits by the neighboring countries and the

initial processing of the extracted raw materials necessitate the establishment of a regulatory and legal base satisfactory in terms of the legislations of the respective countries.

3. It is logical to improve to a substantial degree the extension of the experience and knowledge in the field of geological research and mining, and moreover, the conditions for that are existing, both based on the accumulated experience and the results achieved in the individual countries and based on the options provided by the contemporary means for processing, storage and delivery of information.

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3rd Balkan Mining Congress, BALKANMINE 2009, 1-3 October, İzmir-TURKEY, © 2009, ISBN 978-9944-89-782-2

Some Aspects of Development of Coal Industry in Bosnia and Herzegovina

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ABSTRACT For a long time, Bosnia and Herzegovina meet its energy needs (over 11 TWh in the 2008th) by the production of power from local power plants using its own coal (60%) and hydro potentials (40%). Available balance reserves of coal in Bosnia and Herzegovina (approximately 2.5 billion tons) can provide today's production (about 10 million tons) of coal over decades, but it can meet demand of new generation of thermal power plants.

It is expected that the coal industry in Bosnia and Herzegovina, as in many other countries, will be a guarantor of the general and energy security in the future, but in a new way and in the new conditions. In that order, many problems should be solved, reduced operating costs (primarily labor costs) and realize significant investments in the modernization of fixed funds.

Incorporation of Bosnian-Herzegovinian thermo energy in trans-European energy network will require not only restructuring of coal industry but also the intensive capital investment in the finalization of the coal with which Bosnia and Herzegovina does not dispose today. External sources of capital will certainly be necessary in the extent and intensity defined by thermal energy plans. This work is a vision of possibilities for solving the aforementioned Bosnian-Herzegovinian energy problems.

1 COAL RESOURCES IN BOSNIA AND HERZEGOVINA

Bosnia and Herzegovina, although a small country, has a significant production of coal. Coal sector is here, as in most countries of Southeast Europe, traditionally an important segment of the energy and economic structures. With regard to not proven reserves of oil and gas, coal represents 90% of the total energy potential of Bosnia and Herzegovina.

Coal reserves are located mostly in Tuzla, Central Bosnia, Ugljevik and Gacko pool, and get mainly on open pits (about 80%) and mostly burned in the local thermal power plants. Estimated geological reserves of coal in Bosnia and Herzegovina are approximately 5.6 billion tons, of which: 45% balance, 11% separately from the balance sheet and 44% of the potential indicating a quite low level of exploration, and the need for increased investment in geological research. Of the total estimated balance reserves on lignite waste 1.4, and on brown coal 1.1 billion tons.

Bosnian-Herzegovinian coal reserves belong to brown coals and more quality lignites quite similar to those in lignite deposits of some other countries in Southeast Europe. These reserves are not however, completely technical-technological, economic and ecological verified. It is especially the case for the operational reserves which are mostly in Bosnian-Herzegovinian coal mines undefined in accordance with applicable standards, given the present number of hydrogeological, geological and urban problems and conditions. It is necessary for that to invest in analysis of exploitation factors and situation in mines. Lower limit of exploitation reserves is continually changing, depending on the demand and price of coal. But regardless of the problems of quantitative and qualitative definition of coal reserves in Bosnia and Herzegovina may, however, state that it could be a solid basis for future energy development of this region. It should have in mind the expectation that the completion of research with total reserves will reach their increase, which will result in growth of coal potential in the coming period, not only underground, but surface exploitation.

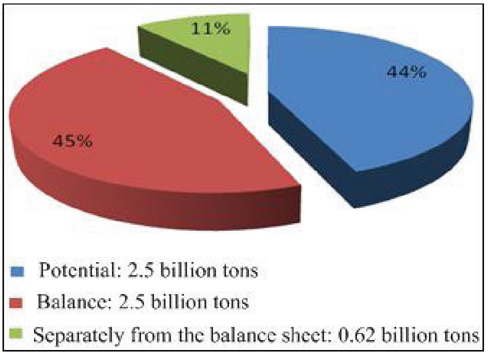


Figure 1. Estimated reserves of coal in Bosnia and Herzegovina (Anon, 2007).

The average calorific value of clean brown coal in Bosnia and Herzegovina is approximately 17,000 kJ/kg, and lignite of 7500-12500 kJ/kg. It is necessary however, note that the Bosnian coals usually have high content of moisture and ash and their burning in thermal power plants cause series of problems. To the long-term (since coal will probably remain primary energy source in the country for a long time) implies the need for continuous adjustment of the technological scheme of finalization of coal in thermal power plants to the quality of available coal. Bearing in mind this fact, and the geographical position of Bosnia and Herzegovina in the context of possibilities of supplying with qualitative coal, experts from USAID (1997) prognosticated taht Bosnia and Herzegovina 'might never become a market of cheap electricity' because according to them it requires reducing power in the total content of the production costs at the level of country. This forecast is, according to today's circumstances, but aslo the intensive energy demand proved to be totally ill-founded. Specifically, long-term prospects of coal as energy resources in Bosnia and Herzegovina, as well as in other countries, will depend primarily on development of technologies that would eliminate or drastically reduce air pollution from new power plants on coal.

2 GENERAL CHARACTERISTICS OF THE COAL MINES IN BOSNIA AND HERZEGOVINA

According to available statistical data in Bosnia and Herzegovina in 2007th it is produced nearly 10 million tons of brown coal and lignite which is about 53% of the pre-war production. The dynamics of growth of production of coal (brown and lignite) in Bosnia and Herzegovina and by the entities is quite heterogeneous and it was influenced by various production factors and unproductive.

According to the opinion of local experts „with certainty can be said that the available reserves of coal in Bosnia and Herzegovina are sufficient for ensuring the future needs for the production of electricity“ (Anon, 2006). Energy analysts of EIA believe that coal will remain significant energy resource in Bosnia and Herzegovina much longer and „ additional plans for new and reconstruction of existing thermal capacities in the East-European countries including Bosnia and Herzegovina, Bulgaria, Czech Republic, Macedonia, Slovakia and Yugoslavia are indicator that coal continues being significant source of energy in the region“ (EIA, 2005).

Based on information collected from the coal mines of Bosnia and Herzegovina, might be easy to identify quantitative facts about the total demand of coal in Bosnia and Herzegovina by the dynamic and structure. As in other countries, in Bosnia and Herzegovina the effective demand for thermal capacities is the regulator of manufacturing of coal sector given that Bosnian-Herzegovinian thermal power plants absorbe over 80% of coal production.

The second block of data is necessary in any analysis of Bosnian--Herzegovinian coal sector and includes specific elements of the structure of assets, which includes, first of all, the size of the basic resources engaged in coal production and the degree of their wear. If you will be analytical out of the data presented, it is easy to determine randman or effectiveness of total assets or randman of equipment and compare it in the dynamics of time or with similar companies.

Degree of wear is used for reasoning about the age, deterioration of basic resources or groups of assets (primarily equipment). Indicator of high deterioration of basic resources in the coal mines of Federation of Bosnia and Herzegovina (in Republic of Srpska is a similar situation) warns that is necessarry to access their renewal urgently.

Table 1. Coal production in Bosnia and Herzegovina for 1990 and the period of 2003-2007.

Elements	Coal Production by Years (000 tons)					
	1990	2003	2004	2005	2006	2007
Federation B&H	13,125	5,419	5,639	5,785	5,885	6,126
Brown coal	8,232	3,296	3,391	3,526	3,606	3,871
Lignite	4,893	2,123	2,248	2,229	2,279	2,255
Republic of Srpska	5,016	3,586	3,256	3,354	4,070	3,636
Brown coal	1,993	1,447	1,227	1,134	1,579	1,684
Lignite	3,023	2,139	2,029	2,220	2,491	1,952
Bosnia and Hercegovina	18,141	9,005	8,895	9,140	9,955	9,762
Brown coal	10,225	4,742	4,617	4,691	5,185	5,556
Lignite	7,916	4,263	4,278	4,449	4,770	4,207

Source: *Information Division of Statistics of FB&H and RS for each year*.

Table 2. Cumulative table for placing coal in Bosnia and Herzegovina.

(in 000 tons)

Description	2005			2006			Index		
	B&H	FB&H	RS	B&H	FB&H	RS	B&H	FB&H	RS
Total sales	9,159	5,789	3,370	9,995	5,924	4,071	109	102	121
From this: thermal power plants	7,721	4,579	3,142	8,482	4,787	3,695	110	105	118
(thermal power plants % participation in the overall placement)	84	79	93	85	81	91	-	-	-
Other consumers	1,086	934	152	1,381	1,006	375	127	108	247
- export	281	276	5	135	131	4	48	47	80

Source: *Data of professional services the coal mines of Bosnia and Herzegovina*

Table 3. The structure of fixed assets of coal capacities in 2006 in Federation of Bosnia and Herzegovina.

Name of assets	December 31, 2006		(in 000 tons)
	Cost	The current value	Level of write off
Intangible assets	5,334	191	96
Tangible assets	3,300,806	757,616	77
- Land	195,187	195,187	0
- Building	1,027,978	264,645	74
- Equipment	1,732,209	116,368	93
- Other	345,432	1,818,416	48
Long-term demarcation	11,690	11,331	3
TOTAL	3,317,830	769,138	77

Source: *Data of professional services the coal mines of Bosnia and Herzegovina*

The short analysis of employment, bearing in mind the specifics of coal sector, shows that it is necessary to process the primary qualification structure of employees, the structure of employees according to age and structure of employees according to work experience. In terms of the total employment by qualifications, the situation in the coal sector is given as follows:

Table 4. Qualification structure of employees in 2005 and 2006 in coal mines in Bosnia and Herzegovina.

Serial number	Actual qualifications	Employees on December 31, 2005			Employees on December 31, 2006		
		B&H	FB&H	RS	B&H	FB&H	RS
1.	non-eligible	2343	2,154	189	2,188	2,008	180
2.	semi-skilled	871	811	60	831	775	56
3.	skilled	6,645	5,732	913	6,552	5,616	936
4.	secondary education	3,304	2,731	573	3,406	2,803	603
5.	highly skilled	1,912	1,408	504	1,994	1,364	630
6.	higher qualification	272	183	89	275	179	96
7.	high qualification	782	605	177	811	633	178
8.	Master	48	37	11	46	37	9
9.	Doctor	8	7	1	9	7	2
	TOTAL	16,185	13,668	2,517	16,112	13,422	2,690

Source: *Data of professional services the coal mines of Bosnia and Herzegovina*

From the data collected can be found common characteristics more or less present in all Bosnian-Herzegovinian coal mines. Those are overabundance and high age structure for the conditions of mining activities which in itself indicates the necessary directions in the future time. In calculating the effects which provide engaged work force, the situation is quite heterogeneous by coal capacities regardless of whether the account of productivity of work by mines include all employees or only production workers. Factors that affect are, primarily, technical infrastructure, the number of workers with reduced working capacity, etc.

One of the central tasks of analysis of coal sector in Bosnia and Herzegovina is to assess and all those factors, conditions and circumstances that have worked on the formation of such cost structure in coal sector. With this aspect, the structure of costs of production of coal, regardless of whether the analysis of costs by the principal or treated through the participation of direct and indirect costs in total cost structure there is always a certain specificity in relation to other sectors. It is easy to perceive that in the case of today's Bosnian coal production:

The size, structure and dynamics of the costs are definitely the basic factors which must necessarily take into account when the cost price of coal produced by some Bosnian-Herzegovinian coal mines is determined. In addition, the factor of the volume of business the use of technical capacity, technical infrastructure, qualification structure of employees, productivity, etc. should be analysed. In comparison with global coal standards, productive work in the coal mines of Bosnia and Herzegovina is very low (surplus employees and technological backwardness) but among them there are significant differences.

Table 5. Actual cost of coal mines in Bosnia and Herzegovina in 2005 and 2006.

(in 000 KM)

Name of cost group	2005			2006			Index		
	B&H	FB&H	RS	B&H	FB&H	RS	B&H	FB&H	RS
Cost of materials	59,236	51,114	8,122	58,844	50,093	8,751	99	98	108
Energy costs	63,033	43,936	19,097	76,331	51,095	25,236	121	116	132
Amortization	57,635	38,610	19,025	62,248	38,972	23,276	108	101	122
Staff costs	195,932	169,002	26,930	227,287	192,634	34,653	116	114	129
Other costs	75,234	42,430	32,924	54,793	30,570	24,223	73	72	74
Costs earlier period	2,466	2,428	38	6,467	6,446	21	262	265	55
Other expenses	26,079	15,591	10,488	31,076	29,444	1,632	119	189	16
Internal expenses	7,584	7,584	0	8,443	8,443	0	111	111	-
TOTAL EXPENSES	487,319	370,695	116,624	525,489	407,697	117,792	108	110	101

3 THERMAL POWER CAPACITIES OF BOSNIA AND HERZEGOVINA IN THE ECONOMIC CONTEXT

Power has been one of the major factors of development of Bosnia and Herzegovina for a long time. Total installed power capacity is now about 4000 MW, of which approximately 50% waste on production of thermal energy. Relations between hydro and thermal production vary depending on hydro conditions during the year. Total production in the country exceeds consumption, so a significant portion of power is exported to neighboring countries.

In this time in Bosnia and Herzegovina exist three vertically integrated energy monopolies that produce and distribute power to different parts of the country.

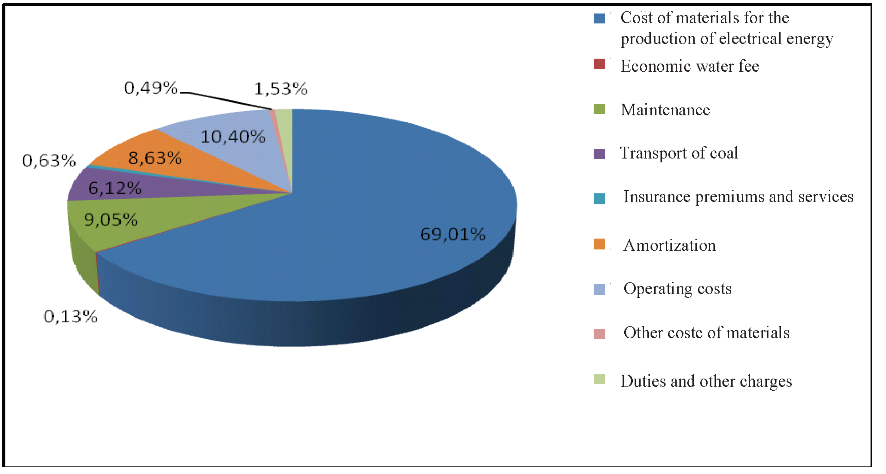
Table 6. Electro energy power of B&H companies.

Company	Installed power of thermal energy (MW)	Installed power of hydro energy (MW)
Elektroprivreda B&H	1357	492
Elektroprivreda HZHB	-	803
Elektroprivreda RS	600	769
TOTAL	1957	2064

The listed companies have their own production, consumer base, and the power of the thermal power is quite different.

Each of these companies has its own production, transmission and distribution, while the joint power coordination committee owned all three companies coordinate dispatching and ensure the integrity of the system within the country.

Amog these electric comapnies, only Elektroprivreda HZHB does not use coal in the production of power while the other two companies use coal input in this production significantly. Coal mines in Republic of Srpska (in Ugljevik and Gacko) are organizing incorporated in enterprises for the production of thermal energy (where the price of coal is internal category) what is a significant advantage in comparison to similar capacities in Fedearation of Bosnia and Herzegovina. For a log time in Fedearation of Bosnia and Herzegovina is felt a monopoly position of thermal power plants which is reflected, above all, through the unequal distribution of economic effects arising from the joint production of power in which cost price dominate the cost of production and transportation of coal. For example , in „Thermo power plant" (TE) Tuzla cost of materials for power production are 69% in the structure of total costs (of which 92% belong to cost of coal).



Source: *Report on the activities of the "Thermo power plant" (TE) Tuzla for 2008th, Pp.32*

Figure 2. Participation of certain costs in the total cost of power production in „Thermo power plant" (TE) Tuzla.

According to available data (Anon, 2007), Elektroprivreda B&H in 2008th realized profit of approximately 23 million EUR, while 7 Federal coal mines, in the same period, made a loss of about 25.5 million EUR. Only Coal mine „Banovici" realized profit of approximately 2.8 million EUR, while Coal mine „Gračanica" worked at the border of profitability. The salaries in the listed Federal mines are twice lower than those in thermal power plants, and the investment potential is minimal. The issue of price of coal for thermal power plants (today it is about 2.3 EUR/GJ) is still in progress. The latest activities of the Government of Fedeartion of Bosnia and Herzegovina are trying to change this situation by passing the Law on the allocation of funds to consolidate the coal mines and a decision about joining the coal mines to Elektroprivreda B&H on the level of the Fedeartion of Bosnia and Herzegovina. It is expected that the effects of these activities of the Government will give results in the near future.

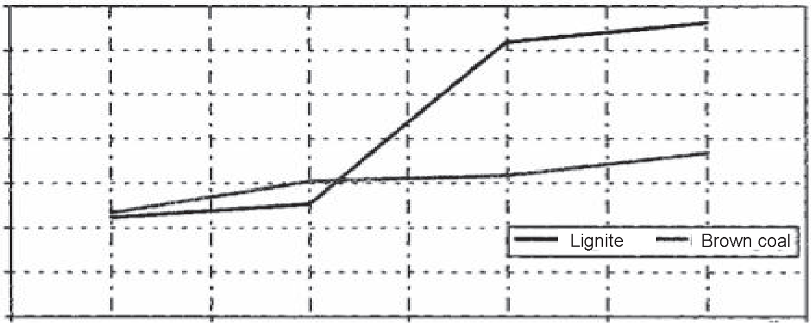
4 FRAMEWORK POSSIBILITIES FOR FUTURE DEVELOPMENT OF THERMAL ENERGY COMPLEX IN BOSNIA AND HERZEGOVINA

Research trends of development of coal industry in particular coal regions in the world points to certain laws in connection with the development logic of coal economy. Safe and sustainable energy offer takes the central place in the long-term projections of the development of many countries. Coal will play an important role in this, especially because of growth in energy demand in countries whose development in the expansion. Simply put, „coal will remain the most economic fuel in the years to come" (Anon, 2008a).

Today, it is difficult to predict how will Bosnian-Herzegovinian coal sector develop in the future. It will mostly depend on the dynamics of social and economic structure in Bosnia and Herzegovina. It is certain, however, that the presented current development of coal sector does not have a rational economic coverage because it does not show the elements of economic efficiency. It warns on difficult situation more than it provides a reliable economic rational forecast, or potential of future development.

It is certainly that the radical changes in energy-coal sector have to come. Specifically, it should be noted that the Bosnian-Herzegovinian energy system will be one of the sub-European energy system which can establish various forms of mutual dependencies. This is especially when it comes to the countries of Southeast Europe, because the energy plans of Bosnia and Herzegovina, as a member of Energy Community of South East Europe, should incorporate into the regional energy strategy of this area.

No matter what the country there is no energy development strategy, in Bosnia and Herzegovina is quite realistic to expect in coming decades in cooperation with foreign partners, launching and realization of several new energy projects such as: new thermo power plants „Bugojno“, „Kongora“, completion of the investment in in new thermo power plant „Stanari“ and the expansion of the capacity of current thermal power plants in Tuzla, Kakanj and Gacko. This will, as seen in the following figure, require twice as higher production of coal from today:



Source: World Bank, *The Energy sector study in Bosnia and Herzegovina* - module coal, 2007.

Figure 3. Approximate estimates of production of coal in Bosnia and Herzegovina for the period 2005-2020.

After the 2020th there are options (also with the arrangements with foreign partners) to continue investing in new thermal power capacities („Tuzla B“, „Bugojno 2“, first of all). Of course, before that it is necessary first, to fully execute the restructuring of Bosnian-Herzegovinian coal mines.

In the construction of future thermal plants it takes into account the choice of technologies depending on the composition of available coal, the location of future thermal power plants, etc. Such energy capacities will have to have to have embedded equipment for reducing emissions of harmful substances, which will increase investment and affect their efficiency. Respecting of European, Bosnian-Herzegovinian and regional recommendations, standards and laws for the protection of the environment will have its price of course. It withdraws the primary question of more efficient future production in coal mines in Bosnia and Herzegovina as a basic raw material for production of power.

5 CONCLUSION

Bosnia and Herzegovina, as the state created the disintegration of former Yugoslavia, now passes through the difficult phase of its social and economic development. This work is essentially reduced to the consideration of specific and actual problems in Bosnian-Herzegovinian coal industry. The basic issue is more efficient evaluation of the energy resources in the dimension of time.

Complexity of this issue and its importance for future economic and energy structure of Bosnia and Herzegovina requires increased involvement and coordination of economic and political actions in this plan. However, it will be no easy task because of given problems, but the current political and economic situation.

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Systems for Monitoring of Electrical Energy Consumption in Mining Industry of Bulgaria

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ABSTRACT The development and commissioning of systems for monitoring and control of electrical energy consumption at the mining companies have been reviewed. Structural schemes have been presented and the capabilities of the systems, based on different computer generations have been described. Methods of improving the systems for monitoring of electrical energy consumption have been formulated.

1 INTRODUCTION

The need of developing and commissioning of modern information technologies for monitoring and reading the energy consumption at the large mining industry enterprises is determined by several factors:

A/ Rational use of electrical energy is an issue of extreme importance worldwide, which is especially critical for industrial productions of high power consumption. The search of methods for reduction of energy consumption and making effective and time-relevant managerial decisions in the above direction is impossible without the availability of updated and precise information.

B/ Some of the large mining companies belong to the category of "Privileged consumers", for which electrical energy suppliers offer significantly lower cost under conditions of a contract with hourly fixed average consumption. The consumer, however, remains with the problem of planning and afterwards, monitoring and observing the consumption of negotiated power during each hour of the day. The decision of the above problem is related to the real time continuous monitoring and control of the power consumption, which is impossible without establishing the respective information measuring systems.

The processes of privatization and splitting of individual companies from large enterprises brought to establishment of many separate units with their own accounts. In many cases, the above brought to a more complicated scheme for pricing of the consumed electrical energy and financially-related arguments among the separate consumers, which is difficult to resolve by the old system for reading of the electrical energy consumption.

2 INITIAL STEPS

Computer-base systems for monitoring of electrical energy consumption have been commissioned at the mining companies, under the guidance of the author, since 1986. They have been developed based on different apparatuses and software programs. The regulative requirements, the limiting conditions, introduced by power supply companies and valid at the moment tariff rates for payment of electrical energy have been incorporated in the algorithms.

The first systems for monitoring have been developed based on 8-bite Bulgarian microcomputer with a standard periphery: video-display, floppy-disc device, printing device. Measurements of active and reactive power are performed by power converters with unified current output signal. All the other modules of the system, analogous-digital converter, real-time clock, current – voltage converter, block for controlling the printing device have been developed based on integral schemes. The software has been prepared on a module principle and all the modules have been programmed in Assembler for microprocessor 6502. The micro-processing system provides the following functions:

- continuous monitoring of consumed active and reactive electrical energy and the power factor for the entire factory;
- continuous monitoring of active power at each of the outlets of the sub-station;
- sound signal when the preset limits of power are exceeded;
- printing on an hour basis of the data of total consumed and over-limit energy in the peak zones as well as energy consumed by different consumers;
- printing of 24- hour record, including the consumed active and reactive electrical energy – totally and for the separate consumers; average value of the power factor; value of the electrical energy; amounts of sanctions or premiums in dependence of the value of the power factor.

Systems for monitoring and control of the consumption of electrical energy having the above structure have been commissioned at the mining companies – “Elatsite” Mineral Processing Plant and “Chelopech” Mine Enterprises (Stoilov *et al.*, 1986).

In 1987 a modernized version of the system has been developed and commissioned at the substation “Erma Reka” of the “GORUBSO”-mining and mineral processing enterprise. (Stoilov *et al.*, 1987). Energy-meters with pulse outlet for the active and reactive electrical energy have been applied. Pulses of each energy-meter go along a two-conductor line and enter into a block “Counters”. At an interval of one minute the micro-processing system scans all the counters and further to a processing takes the average values of power in the monitored points to the screen. The information is transmitted also to the energy department of the Processing Plant “GORUBSO” by a telephone line. Bulgarian modems of the type CM8107 of speed of data transfer 300 bps have been used.

3 EXAMPLES OF OPERATING SYSTEMS FOR MONITORING

With the development of electronics the apparatus of the systems for monitoring of electrical energy consumption has been continuously improving (Danailov *et al.*, 1990). Modern versions are based on the most recent achievements in the field of measuring micro-processing and communication technique.

The need of developing and commissioning modern information technologies for monitoring and reading the power consumption in the large industrial enterprises is becoming more and more topical. The introduction of free market of power consumption in Bulgaria laid down new requirements towards the functions of the systems for monitoring and operative control of energy consumption.

3.1 System for Monitoring at the “Elatsite-med” AD

A computer-based system for monitoring of energy consumption has been developed and put into operation at the “Elatsite–med” AD. The system measures the values, characterizing the electrical energy consumption in both sub-stations – “Elatsite” opencast mine 110/6 kV and the mineral processing plant – Mirkovo village 110/20/6 kV. The active and the reactive electric power are measured at the inlets 110 kV of the substations for all the outlets – 20 kV and 6 kV. Communication between the servers of both sub-stations is performed by the LAN network of the company (Stoilov *et al.*, 2003).

The apparatuses of the system are as follows: converters for active and reactive electric power, converter (I/U converter), microprocessor controller, which scans and identifies the signals, carrying the information and transfers them to a computer for processing in real time. The structure of the system at the mineral processing plant is shown in Figure 1.

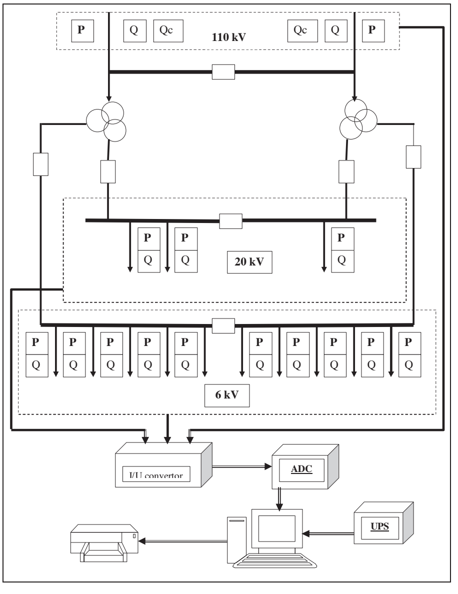

Figure 1. Structure of the system.

The software of the system has been composed hierarchically in the operational medium of Windows XP. At first level the primary converters for active and reactive electrical energy are scanned selectively and addressed to PC with licensed software products. At second level the incoming information is processed in real time following a specific algorithm and a program.

The system calculates:

- The active and reactive electrical energy and cos φ, mean values for each 5 s and 30 min

The mean values for 5s are indicated as momentum values. They are stored for 30 min. The mean values for 30 min are stored at least 5 years.

- The active and reactive electrical energy for a day, a month and an year
- Expenses for power consumption.

A module has been developed in order to control the electrical loads, which visualizes the information for energy consumption in real time, comparing it to previously contracted hour rates for the commercial consumer. Data are extracted for the hour zone, the applied quantity of electric energy in the weekly schedule, the consumed electric energy from the beginning of the current hour. The calculation of prediction values is applied to formulating recommendations for increase or reduction of electric energy with the objective of observing the contracted quantity of energy for respective hour. The actual management of electrical loads is performed by the electrical energy dispatcher.

The above described system has been working since 2003. The analysis of infor mation provides an option for in-time reaction and performance of effective energy management and reduction of cost for consumption of electrical energy in the company.

3.2 System for Monitoring at the "Maritsa East" Mines

The system has been designed to monitor the electrical energy consumed at the "Maritsa East" Mines. The system has been designed as a sub-system of the existing one for energy monitoring at the "Maritsa East" Mines. The system continuously monitors data, entering into the server from the energy-meters, installed in the zones for commercial measuring for payments to the electricity supplier, both on the side of 110kV, and the side medium voltage. A balance for the entire company is calculated based on the accepted time interval for calculating the mean values (5 min) and a recommendation is submitted for increase ore reduction of electrical load with the objective of observing the contracted quantity of electrical energy for the respective hour. In order to prepare the records of consumed electrical energy for the different mines, the system monitors the readings of energy-meters of transformers at the separate sub-stations. The above data are processed and saved for all measuring points and form the daily, monthly and quarterly records for the three mines, the Central Repair Workshop and the "Maritsa East" Mines EAD in general.

With the objective of more precise planning of contracted quantities of electrical energy for the week schedule of supplies the system has been provided with a program module for predicting, which operates on the basis of statistical processing of accumulated data. The system has been put into operation in 2004.

3.3 System for Monitoring at the "Stomana Industry" AD

A system for monitoring and remote reading of electrical energy consumption has been developed and it has a hierarchical structure, including server, local stations and energy-meters (Stoilov *et al.*, 2008).

Sources of primary information for the system are the electronic energy-meters with an option for remote transmission of information.

The local stations are computer systems, united in a network, performing recovery, accumulation, primary filtering and initial processing of the data from the group of energy-meters, associated with them. Depending on the structure of the company, the territorial disposition of energy-meters and customer's wishes the connection between the energy-meters and local stations may be performed by an interface RS 485, radio-modem or "currency circuit". The following is important for determining the maximum number of energy-meters, connected to a certain local station: the selected type of the connection, the capacity and admissible number of concentrators, connected to one local station and last but not least the desired speed of the actions.

The server maintains and controls a data base, processes information, synchronizes the work of local and consumer stations, routes and processes the queries in the entire system.

Each existing computer system, which is included in the local network, may be used as a consumer station. The only precondition is installation of respective software and provision of required access rights. The system for control of electric energy consumption has been commissioned at "Stomana Industry" AD. It comprises:

- 120 energy-meters, manufactured by the "Multi-processor systems" OOD company, distributed over the entire territory of the company;
- 12 local stations – the number and location of the local stations has been determined with regard three criteria: options for including to the existing inner optical network, minimizing the length of connections to the energy-meters and providing the required speed of the action, for that specific task it was 1 minute in order to refresh the data from all the energy-meters, included in the system.
- Server station, located in the Main Sub-station and computers of the consumer, defined as user stations.

Part of the structure of the system is presented in Figure 2.

The exchange of information between the energy-meters and the local stations is performed by means of an interface RS 485. An interim joint between the energy-meters and the local stations is the converter of interface I-7563. It allows the inclusion of up to 256 energy-meters in a topology of 3-ray star and submission of data through a standard USB port. The connection between separate energy-meters and the converter is performed by a screened cable of the type "coiled couple". The maximum distance between energy-meter and a local station does not exceed 1000 meters. The speed of data transmission is limited by the energy-meters and it is 9600 bps.

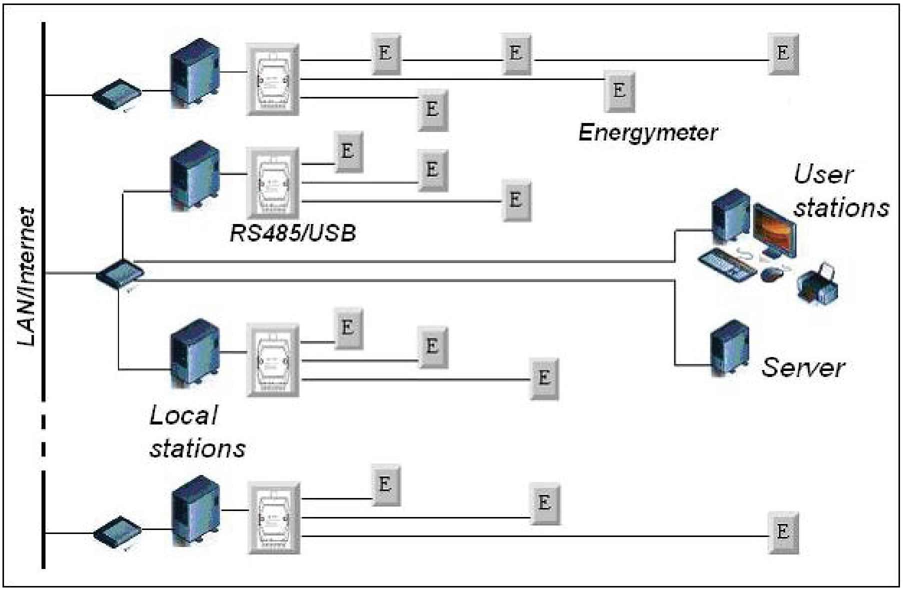

Figure 2.Part of the structural scheme of the system

The commissioning of the system at the "Stomana Industry" AD further developed the respective software, because the specific needs of the company required establishment of the following options:

- Defining several calculating schemes for pricing the electrical energy, because some of the consumers are supplied on the side of low voltage, and others on the side of medium voltage, some of them pay a fee for maintenance of the energy grid, others not etc.;
- Introduction of monitoring points, the data for which are formed by large number (in some cases more than 30) energy-meters;

Preparing monthly reports for paying the electrical energy, consumed by enterprises outside the "Stomana Industry" AD.

4 CONCLUSION – OPPORTUNITIES OF THE FUTURE

The measuring system for real time monitoring and operative control of electrical energy consumption is characterized by great flexibility and adaptation with regard specific features of the specific facility. At present new software modules are being developed, which provide the following additional opportunities:

- Remote access to the data for electrical energy consumption in real time;
- Access to all of the system modules by an internet connection if the server machine is set as accessible in the internet zone (real IP address);
- Modern fast data base, accessible by the internet;
- Option for including into the system of devices, which measure both energy and power;
- Selection of interface language;
- Operation of the system modules in a mode of service;
- Possibilities for preparing records, according to the customer's needs;
- Grouping of the energy-meters in so called control points;
- Extraction of prepared records for any period of time;
- Graphical representation of results from the records;

- Graphical monitoring of the consumption of electrical energy;
- Options for preparing many calculating schemes for pricing the electrical energy;
- Preparing of up to 5 different hour-schedules of electrical energy consumption;
- Option for selecting an hour-schedule of a total of 5 schedules for each control point of the system;
- Possibility for presetting of an interval for the hour schedule - 5, 10, 15, 20, 30, 60 minutes.
- Export and import filters in a standard (CSV) format and export to MS Excel;
- Possibility for preparing dynamic active block schemes of the working units of the company, according to the needs of the consumer;
- Possibility to use the data of other systems of the client;
- Possibility for establishing of consumers and limiting the use of some of the system modules;
- Possibility for remote maintenance of the data base;
- Maintenance of back-up servers in order to keep the data;
- Possibility for upgrades and extensions, according to the client's needs.

The commissioning of systems for monitoring of electrical energy consumption established an opportunity to observe the contracted hour-based quantities of electrical energy, which brings to reduction of cost for electrical energy consumption. The above improves the control of electrical energy consumption, establishes a reliable background for precise determining of quantities, required for individual productions and the company as a whole. It also supports the standardizing of consumption of electrical energy with regard technological processes and separate consumers.

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http://www.sigmadev.net Sigma Factor EMS

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3rd Balkan Mining Congress, BALKANMINE 2009, 1-3 October, İzmir-TURKEY, © 2009, ISBN 978-9944-89-782-2

Metalogenetic Characteristics of the Groot Fe-Ni Deposit Republic of Macedonia

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ABSTRACT In this paper will be present and analyzed the data for mineralogical composition of the weathering crust of Groot deposit. This deposit, in general, is combination of silicate nickel and redeposited nickel iron. There are several metals in the mineral association of the Groot deposit, and nickel is present in the mineral nepuite - garnierite. Except these minerals, in the Groot deposit, especially in the redeposited iron - nickel ores, there are sulphide nickel minerals, like millerite. Ore is with characteristic oolitic - pisolitic structure which is intensive deformed as a result of post ore tectonic movements. Ore bodies are lens like with different dimensions.

Nickel content in the deposit is 1.08%, Iron content is 25.13%, and Cobalt is 0.06%. According the genesis, the Fe - Ni deposit Groot belongs to the group of laterite - redeposited type.

1 INTRODUCTION

Groot Fe-Ni ore deposit is situated in the central part of the Republic of Macedonia, 4 km northeast of the town Veles and 1 km west of village Basino (Fig. 1). The main communication line is the railway Skopje - Veles - Gevgelija.

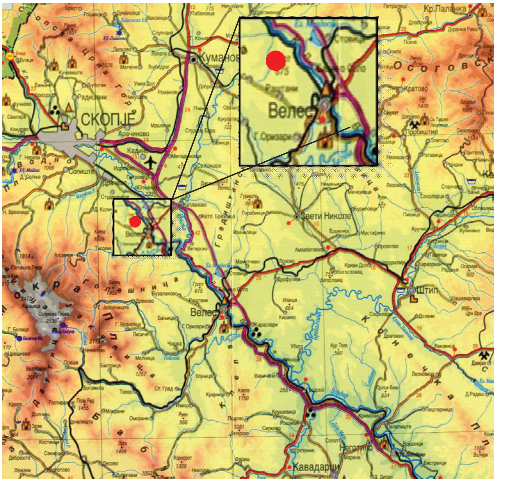


Figure 1. Orientation communication map with the location of the investigated area.

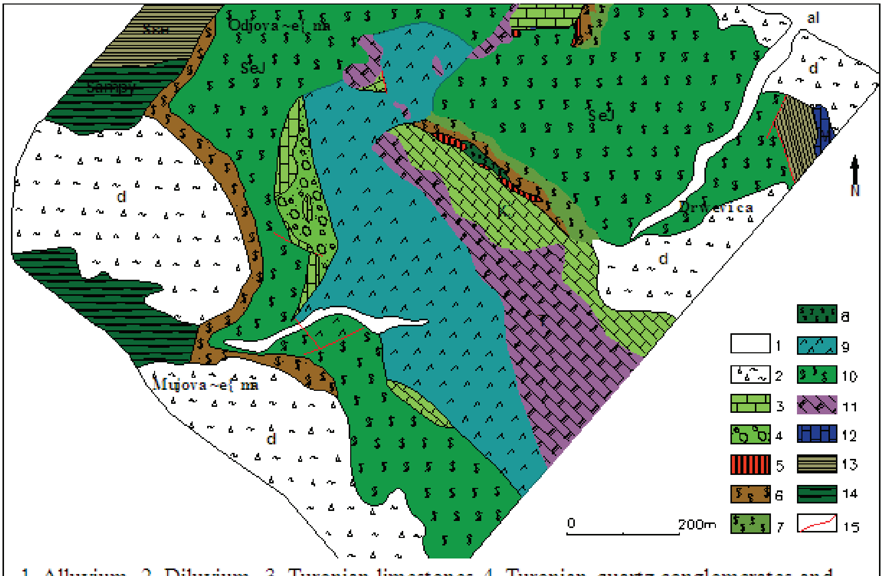
2 GEOLOGICAL CHARACTERISTICS

There are rocks with different age and different petrologic composition: sediments, metamorphic and igneous rocks from Paleozoic, Mesozoic and Quaternary age.

- Complex of Paleozoic metamorphic rocks,
- Complex of Mesozoic sedimentary and igneous rocks,
- Complex of Quaternary sediments.

The complex of Paleozoic metamorphic rocks represents the oldest rocks in this area and they are basis of all other rock masses. They are discovered at the west end of the terrain, between Mujova and Odjova cesma, at the northeast end near the location Drnjevica (Fig. 2). This complex of metamorphic rocks is known as "Veles series" and presents thick and complex creation characterized with very different lithological composition. Basically, they are presented with amphibole - piroxenite shists, amphibolites shists, amphiboles, quartz - micas and marbleized limestones which are folded and tectonically changed.

Mesozoic sedimentary and igneous rocks are the most spreading. They are presented with Triassic, Jurassic and Cretaceous sediments and magmatites.



1. Alluvium, 2. Diluvium, 3. Turonian limestones 4. Turonian quartz conglomerates and sandstones, 5. Iron nickel bearing pisolitic ore 6. Laterite crust, 7. Leached serpentinites, 8. Serpentinites with magnesite occurrences, 9. Diabases, 10. Serpentinites / peridotites, 11. Triassic limestones with cherts, 12. Carbonate - iron marbles, 13. Quartz - sericite shists, 14. Amphibolites, 15. Fault

Figure 2. Geological map of the Groot ore deposit (Boev, Jankovik, 1996).

Triassic sediments are spreaded in relative narrow belt, between localities Odjova Cesma and Drnjevica. They are presented with limestones and cherts (fig. 2).

Jurassic sediments are present in whole area, and in the central part are covered with younger formations. Tectonically and structural are connected with paridotite magmatism of the Vardar zone. Diabases are present in the central part of this terrain.

Upper Cretaceous is presented with quartz conglomerates, sandstones and Turonian clayey - marl limestones.

Quaternary sediments are presented with terrace along the river Vardar, and it is composed of gravel, sand, subsand and subclays. Diluvium is presented with fragments and blocks of different rocks, and dominated place have Triassic limestones and Paleozoic metamorphites. Alluvium is developed in Drenjacki Potok, and it is presented with gravel and sand.

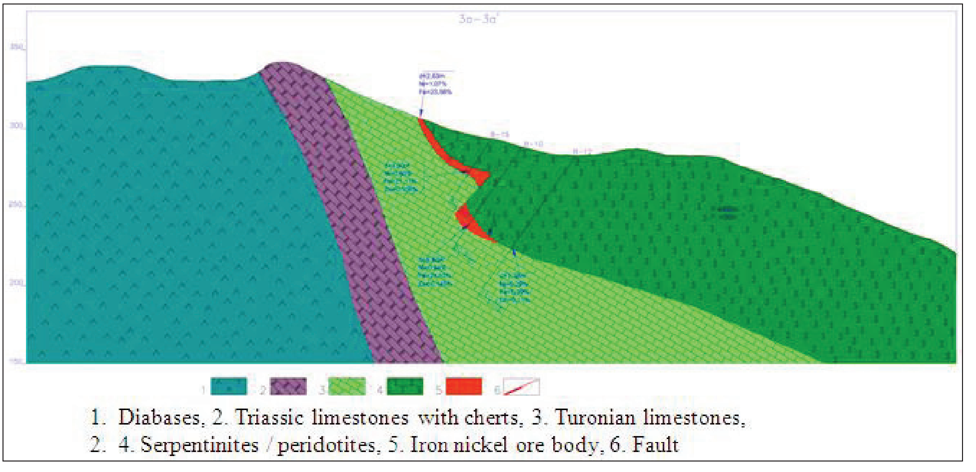


Figure 3. Cross section pat of the Groot ore deposit.

Ni-Fe ore is above the serpentinite, and it is covered with Turonian sediments - Turonian clayey - marl limestones, with relicts of fossil crust of disintegration of serpentinite (fig. 3).

Disintegration of the serpentinite and forming of the laterite crust was in lower Cretaceous, and with the transgression of the Turonian Sea, laterite crust was redeposited and oolite nickel bearing ore was formed. Forming of this ore was through the sedimentation of the disintegrated lower Cretaceous laterite crust, which was redeposited in the Turonian.

According the previous investigations, it can be considered that in the mineral association of the Groot ore deposit, several minerals are present: nepuite, garnererite, magnetite, maghemite, hematite, shamosite, chromium spinels. Nickel is present in the minerals nepuit - garnierite. Except these, in the mineral associationof the Groot deposit, especially in the redeposited iron - nickel bearing ores, sulphide minerals of the nickel, like millerite, are present. Other present minerals: talc, chlorite, ankerite, calcite, illite, quartz, opal and calcedony.

Nickel content in the deposit is 1,08 %, iron content is 25,13 %, and cobalt content is 0,06 %.

Ni-Fe ore has oolite - pisolitic structure. Oolite dominate, and together with pisolites are cemented with serpentinite detritus in which there are fine grains of magnetite, maghemite and rarely chromium spinels.

Oolites and pisolites are composed of magnetite, maghemite and fine grained detritus, i. e. they are composed of fine grained aggregates from the same minerals which are originated from the gel of which later were crystalised in in oolite - pisolitic structure. The space between the oolites and pisolites is made of grained aggregates of magnetite, maghemite, serpentinite detritus and separate grains from chromium spinels.

Ore minerals, as well as the sterile of the elementary mass, because of the later tectonic is fractured, and the cracks were filled out with iron hydroxides which originated from the iron oxides with the influence of the descend waters.

In this deposit, according to the recent investigations, it can be concluded that the disintegration crust of the ultarbasic rocks is the best preserved profile on the territory of the Republic of Macedonia.

In the upper parts of the profile, the upmost part is presented of upper Cretaceous limestones which transgressive lied on the redeposited nickel bearing iron deposites. The thickness of the nickel bearing iron is from 1 to 10 meters. Under this zone is situated the final product of the lateritisation process. This zone is the richest with iron and nickel. This area is directly above the zone of the reticulate laterite, and above it is the layer of the redeposited ores of nickel bearing iron. The content of the iron in this zone is 30 - 40 %, and nickel content is 1,5 - 3 %. The thickness of the zone is 0,5 - 2 m.

Under this zone is the reticulate laterite. The thickness of this zone is 10 - 30 m. There is some concentration of nickel and iron in this layer, and the magnesium is concentrated in magnesium silicates which are like veins or small lenses. Also, there is amorphous silica in the form of opal or chalcedony. Carbonates are presented with calcite.

Going down the profile of the laterite crust, the zone of washed - out serpentinites. The thickness of this zone is 5 - 20 m. this zone is presented with serpentinites and carbonates.

Under this zone is the layer of magnesites. It is presented with coarse net of magnesite in serpentinites. The thickness of the magnesite wires is up to 0.5 m.

Under the magnesite zone, there is zone of fresh serpentinites, which thickness is more than 500 m. nickel content is very low, about 0,3 %.

3 GENESIS OF THE NICKEL BEARING IRON ORES

Genetically, nickel bearing iron ores are directly connected with ultramafic rocks, concretely with serpentinized peridotites, and there is low concentration of silica, aluminium and alkalis, high concentration of magnesium, and always present are iron, nickel, chromium and cobalt.

Under the certain circumstances, during the disintegration, serpentinized peridotites give material for the nickel bearing iron ore structure. High concentration of Fe_2O_3, Cr_2O_3 and low content of Al_2O_3 in ore, as well as the concentration of the ore occurrences in the investigated terrain, lay down directly on the serpentinites. With the presence of the relicts of the fossil disintegration crust of the serpentinites, they confirm the correlation between them. To give material for the nickel bearing iron ore structure, serpentinites must undergo some physical - chemical transformations, i. e. should be exposed on laterite disintegration which lead to wash - out of the silica and concentration of iron in the surface of the disintegration crust.

Serpentinite laterisation was under certain circumstances, if long period serpentinite mass is exposed on continental regime with certain paleoclimatic characteristics. Laterisation could be done only in areas with tropical and subtropical clime, when serpentinite mass is exposed on wet and dry periods. With the influence of these climatic factors, porosity of the rocks and presence of thermal waters near the surface, intensive disintegration and decomposition of the silicates in the surface serpentinites is made.

In the process of laterisation, silicon was molted and partially moved with the solutions, and partially concentrated in the highest level of the laterite crust in form of opal and chalcedony.

Iron present in olivine and pyroxene is released and concentrated in form of oxides, and aluminium which is slightly present in serpentinites, stayed in the disintegration crust. Magnesium is highly present in serpentinite, up to 40 %, was took out of the disintegrated rock, and partially concentrated in form of magnesite. Nickel and cobalt from the olivine and pyroxene stayed as new products of the disintegration crust ore formed hydrated silica with high content of nickel, especially in the central parts of the laterite crust. Chromium present in serpentinite, stayed unchanged and connected with chromite or chromium spinel, and as grains without certain distribution in the disintegration crust.

With the laterization process, laterite crust of the serpentinite masses is formed. Today, this crust is partially preserved, as relict.

Laterite is spongy to reticulate, ochreous, partially magnetic because of the presence of magnetite grains.

Relicts presence in laterite crust shows that this area was in tropical region in which all conditions for intensive disintegration of the serpentinites were present. Serpentinites were subjected to the laterization process, and laterite crusts were formed with thickness of 10 - 30 m. these crusts were disintegrated and redeposited for a long period and with their redeposition in sea, sedimentary oolite nickel bearing iron ores were formed and they laid at the fossil relicts of the laterite crust or at serpentinites, in the basis of the upper Cretaceous (Turonian) sediments.

Process of laterization, or redeposited laterite - oolite nickel bearing iron ore was in the period between Jurrasic and Turonian stage and maybe partially in Cenomanian stage, because upper boundary of laterization is presented by the Turonian sediments.

4 CONCLUSIONS

According the presented data, it can be concluded:

Within the wider area, the following geological formations can be distinguished: complex of Paleozoic metamorphic rocks, complex of Mesozoic sedimentary and igneous rocks, and complex of Quaternary sediments.

Mineral association in Groot ore deposit is presented with: napuite, garnierite, magnetite, maghemite, chromium spinels, talc, ankerite, calcite, quartz, hematite, shamosite, chlorite, illite, opal and chalcedony. Beside these minerals, in the redeposited Fe - Ni ores, sulphide nickel minerals, like milerite, are present.

Tectonically, investigated terrain has complex tectonic structure, expressed with disjunctive structural forms - faults. In the process of their formation, in the first phase, vertical movement of rock masses was done, and because of this, fault structures have character of peels.

According the conditions of the origination, Fe - Ni deposit Groot belongs to laterite - redeposited type. It is with 350m length and small thickness. Ore body has sincline shape which locally is tectonically disintegrated.

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3rd Balkan Mining Congress, BALKANMINE 2009, 1-3 October, İzmir-TURKEY, © 2009, ISBN 978-9944-89-782-2

Nickel, World Production and Demand

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ABSTRACT During the last years, nickel prices have reached historic record heights, while production shows an increment and the metallic nickel stock reserves are dropping sharply down.

The greatest part of the nickel produced comes from ten companies. The rest of the production comes from smaller producers some of which are located at Balkan Penninsula.

The greatest consumer globally is Asia, followed by Europe. During the last decade there was an augmentation of 17% of nickel consumption and is expected on a long-term basis that the nickel consumption will be increased by 2 - 3% annually. In 2008, due to the worldwide recession there has been a falling in nickel consumption and production.

Because of the high demand of the product, nickel prices reached in 2007 historic high peak records. The worldwide recession of 2008 has resulted to a price falling, but the forecasts are positive and is expected an increment of nickel price.

1 INTRODUCTION

Even though nickel is known for over 4000 years for manufacturing of metallic objects, it wasn't recognized as independent element until 18th century. Nickel has gained economic importance in the mid of the 19th century when English chemist Faraday developed a production method of stainless steel based on nickeling.

After Second World War, nickel and cobalt found new applications which led to great demand, leading to the exploration of new deposits and the operation of new nickel production plants.

Globally, 80% of the produced nickel comes from ten producers, while the rest of the world production comes from smaller producers. Amongst them in Balkans are LARCO in Greece, FENI in FYROM and recently became operational Ferronikeli in Kosovo.

Apart from the nickel producers, in the wider area of Balkans there are several companies in Albania which extract nickel ores and export it to the metallurgical plant of FENI in FYROM. In Turkey, nickel ores are extracted and exported to the metallurgical plants of LARCO and FENI. Semi-industrial scale of hydrometallurgical treatment of lateritic nickel ores is also being tested.

The biggest nickel consumer globally is Asia, followed by Europe. The steel industry global recovery, showed China as the greatest nickel user. But in 2008 because of the global economic recession, there has been a global dropping down of the nickel consumption and production.

Nickel usage is constantly increasing over the years and will lead to the exploration and development of new nickel deposits in Balkan area and to the utilisation of new lower cost metallurgical processes.

2 USES OF NICKEL

Austenitic stainless steel comprises 2/3 of total stainless steel production (Fig.1). Series 200 and 300 of austenitic stainless steel are the most used, having very high resistance to stress and corrosion.

Series 200 contains 1-6% Ni, while series 300 contains 8-14% Ni. The most commonly applied stainless steel is series 304, containing 8-10.5% nickel and 18-20% chrome. The 316 series is the stainless steel with the highest resistance to corrosion containing 10-14% nickel, 16-18% chrome and 2% molybdenum. A new category of stainless steel is the hyper–austenitic one, which has higher nickel content and special applications.

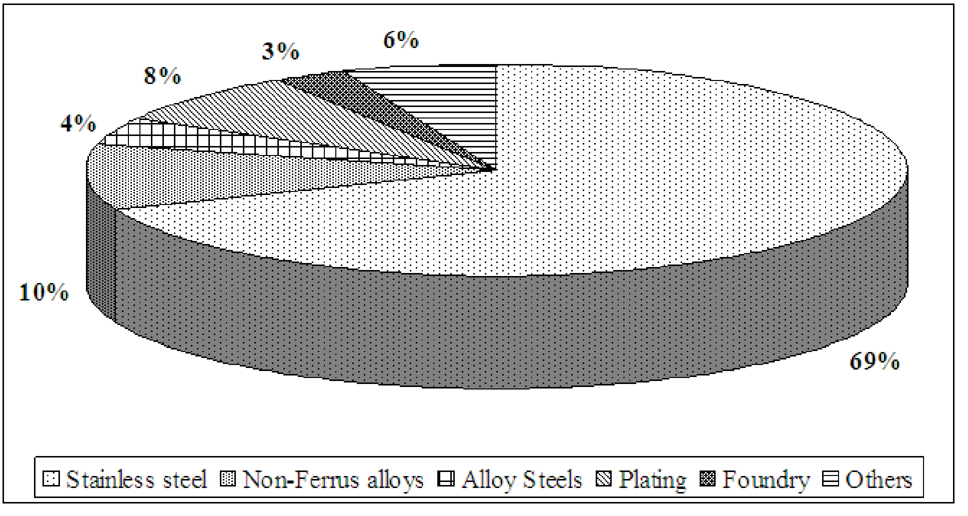

Figure 1. Uses of Nickel.

Other uses include iron and special nickel alloys which are applicable to specialized uses and nickeling. Also, there are many applications in many areas of electronics, transport, communications, re-chargeable batteries and chemical industry.

In the years to come, many technological challenges are expected towards the utilization of nickel, which is to find usage in many breakthrough applications.

3 GLOBAL MINERAL RESOURCES

Total nickel ore Mineral Resources globally are estimated to be 23 billion tones with average nickel content 0.97%. These resources consist of sulphide ores (45%) with average nickel content 0.58% and lateritic ores (55%) with average nickel content 1.32%. Based on the inclusive nickel content according to Dalvi et. al (2004) is estimated that in the lateritic ores are included 72% of the global Mineral Resources while in the sulphide ores are included 28% of the global Mineral Resources (Figure 2).

Even though lateritic ores contain the highest nickel content, only 42% of world production comes from lateritic ores, while 58% originates from sulphide ores. Until 2015 is scheduled to be initialized new units in Brazil, Cuba, New Caledonia, Australia, Philipines and New Guinea with resources of more than 2,000,000,000 tones and capacity 500,000 tones of nickel per year.

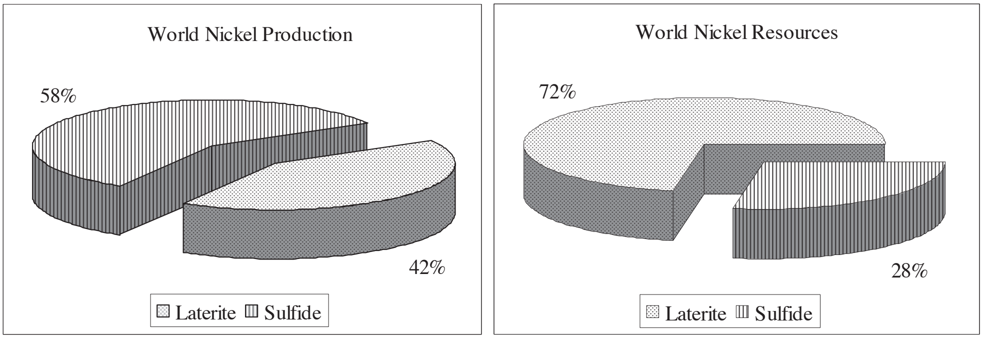

Figure 2. Global Mineral Resources – Nickel production in relation to ore type.

4 NICKEL PRODUCTION

4.1 World Production

After Second World War, nickel and cobalt found new applications which led to great demand, leading to the exploration of new deposits and the operation of new nickel production plants.

World nickel production in 2008 has reached 1,388,500 tones presenting an increasing tendency during the last years (Table 1, Figure 3). The 80% of world production comes from ten Companies. Some of them are the following: CVRD, Xstrata, Norilsk, BHP Billiton, Jinchuan,

Table 1. World nickel production in thousand Tons, (International Nickel Study Group 2009).

	2000	2001	2002	2003	2004	2005	2006	2007	2008
Africa	50.2	52.9	55.0	57.2	54.8	55.5	54.5	49.1	42.1
America	253.0	274.1	290.5	277.3	312.3	307.6	324.5	330.7	312.9
Asia	221.8	213.5	220.3	238.9	249.4	270.6	303.5	379.4	377.3
Europe	403.1	445.5	433.7	457.0	468.3	462.9	511.6	513.7	514.3
Oceania	154.5	174.0	180.7	166.5	166.3	177.5	162.6	156.2	141.9
Total	**1082.5**	**1160.0**	**1180.2**	**1196.9**	**1251.1**	**1274.1**	**1356.7**	**1429.2**	**1388.5**

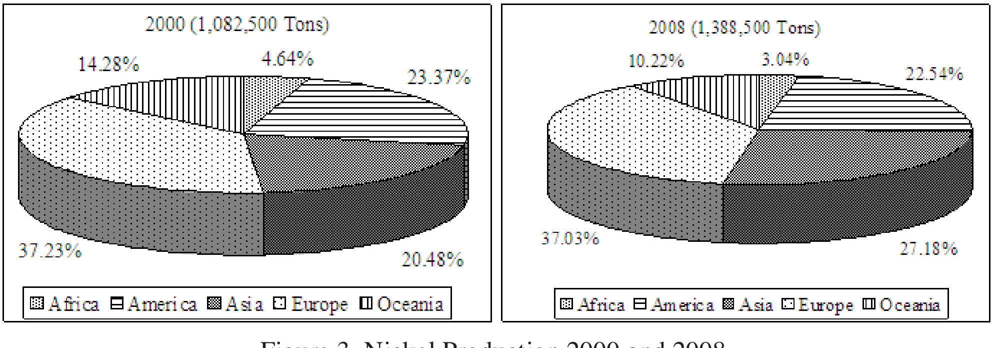

Figure 3. Nickel Production 2000 and 2008.

Eramet and Sumitomo. The rest of the production comes from smaller scale producers and amongst them in Balkans are LARCO, FENI and Ferronikeli.

The main nickel producers of African countries are South Africa and Botswana, while in America, Brazil, Canada, Columbia, Dominican Republic, Cuba and Venezuela. In Asia the nickel producers are China, Indonesia and Philipines.

The European countries producers of nickel are Greece, Finland, Ukraine, Russia, FYROM, Kosovo, Spain and Norway. Finally in Oceania, Australia and New Caledonia are the nickel producing countries.

The great nickel demand has led China, the greater nickel demander globally, to seek out alternative solutions. The result was the replacement of conventional nickel with a new type called “Pig Iron”, a lower cost and quality product. The nickel content “Pig Iron” is between 1.5-8% and it is being used only in China for producing stainless steel series 200 and the total consumption in 2007 was 80,000 tones.

World production in 2008 is distributed as follows: 3.04% in Africa, 22.54% in America, 27.18% in Asia, 37.03% in Europe including Russia and 10.22% in Oceania. Comparing to the production of the previous decade there has been a 35.6% increment but in comparison to 2007, there is a 2.8% reduction.

The economic recession of 2008 has led to production reduction and cancellation of investments in nickel industry. The effects of the recession are expected to last a few more quarters of an year. Future prospects of nickel and all metals in general, are connected to the success of the efforts to support the global economy in order to sustain a high level of demand.

4.2 Nickel Production in Balkans

Nickel production in Balkans is concentrated in Greece, FYROM and Kosovo (Table 2). In Greece nickel production is performed entirely by LARCO and is based exclusively on the mining of Greek deposits located in Evia, Viotia and Kastoria. In the past, there has been an import of small amounts of ore from Albania and Turkey. LARCO’s production covers 1.7% of the global nickel production and takes the 12th place among the companies that produce nickel. LARCO’s nickel production covers 4% of Europe’s demand in nickel.

The second company that operates in Balkans is Cunico which possesses metallurgical plants in FYROM (FENI) and Kosovo (Ferronikeli) and processes the nickel ores from these countries, along with imported ores from Asia.

Apart from nickel production, in the wider area of Balkans operate companies that extract nickel ores without any processing and sent these ores to the metallurgical plants in the area. In Albania, there are several small companies that extract ores, which are sent to the metallurgical plant of FENI. In Turkey the nickel ores are mined and in the past exported to the metallurgical plants of LARCO and FENI. Hydrometallurgical nickel ore process is also being tested at semi-industrial scale in Caldag at Western Turkey.

Table 2. Nickel production in Balkans (.000 Tons).

	2003	2004	2005	2006	2007	2008
LARCO	21.4	21.7	23	21.7	21.2	18.6
FENI	5.6	4.9	7.8	10.9	15	15.1
Ferronikeli	0	0	0	0	0.8	7.1

5 NICKEL CONSUMPTION

The biggest nickel consumer globally is Asia, followed by Europe. The steel industry global recovery, showed China as the greatest nickel user, with nickel consumption 360,000 tones in 2008 (Figure 4).

In 2008 Asia consumed 53.93% of world production, Europe 31.58%, America 12.13%, Africa 2.13% and Oceania 0.2%. During the last decade, there has been a 17% increment of the demand and is estimated that on a long-term basis, the nickel consumption will be increased 2 – 3% yearly. Of course in 2008 there was a 4.1% reduction in demand compared to 2007, but in 2009 the situation will be reversed and the demand is estimated to become positive again.

In the Table 3, the increasing demand for nickel from 2000 to 2007 and the 2008 reduction in demand, due to the global recession are shown clearly.

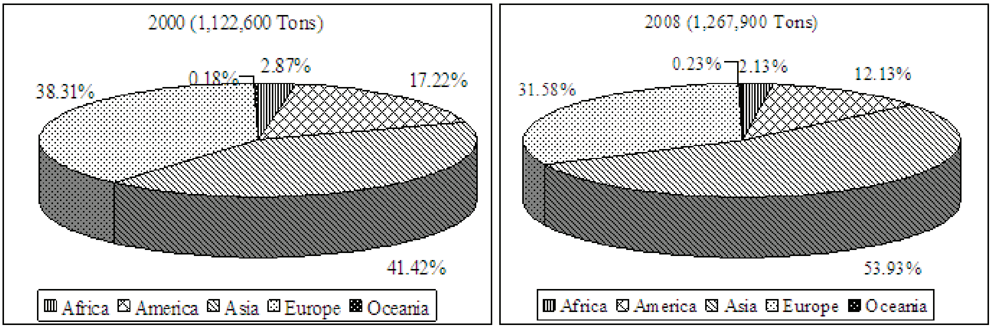

Figure 4. Nickel Consumption in 2000 and 2008.

Table 3. World Nickel Consumption in thousand Tons, International Nickel Study Group 2009.

	2000	2001	2002	2003	2004	2005	2006	2007	2008
Africa	32.2	31.2	35.9	45.5	45.5	32.0	42.0	33.6	27.0
America	193.3	170.7	159.7	159.0	164.6	174.3	180.4	171.4	153.8
Asia	465.0	441.5	501.2	550.4	579.6	592.2	683.7	690.9	683.8
Europe	430.1	458.4	476.0	461.0	454.3	447.2	492.1	423.9	400.4
Oceania	2.0	2.0	2.0	2.7	2.0	2.8	2.9	2.9	2.9
Total	**1122.6**	**1103.8**	**1174.8**	**1218.6**	**1245.9**	**1248.5**	**1401.1**	**1322.7**	**1267.9**

6 NICKEL PRICE DEVELOPMENT

Because of high demand, nickel price has reached historic highest peak in 2007, with about 800% increment compared to the nickel price ten years ago. The year's average price was 37,800 $ per tone, with higher price 52,000 $ in May. In 2008 though, due to the global economic recession, the year's average price was 21,300 $. To get a clear picture of the price development, one can see the nickel price development from 1980 until today. In 1980 the nickel price was 2.18 $ per pound, the historic lowest peak was in 1986 1.76 $ per pound, while in 2007 the same price was 17.15 $.

Nickel prices are settled according to stock market rules, from **L**ondon **M**etal **E**xchange. The nickel price per tone during the decade 1998 – 2008 can be seen in Figure 5.

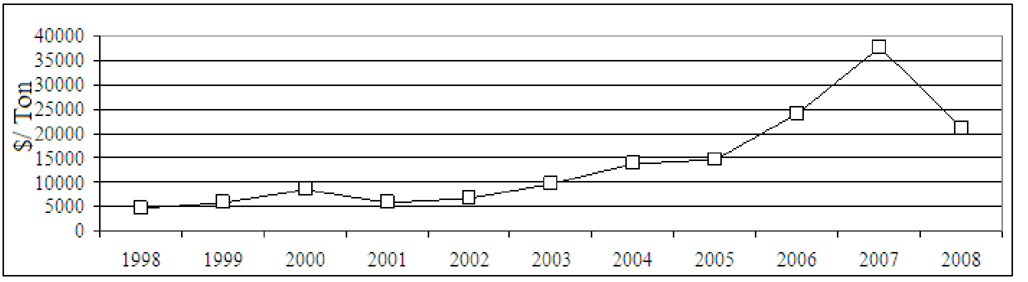

Figure 5. Nickel price development during 1998 – 2008.

7 CONCLUSIONS

Total nickel Mineral Resources globally are estimated to be 23 billion tones with average nickel content 0.97%. According to nickel content, is estimated that in the lateritic ores is contained 72% of the world nickel Mineral Resources, while accordingly in the sulphide ores is contained 28%.

Most of the nickel produced is being used in stainless steel industry. Other uses include iron and special nickel alloys and nickeling. In the years to come, many technological challenges are expected to utilize nickel, which is to find usage in many breakthrough applications.

World nickel production during last years is increasing steadily. Nickel production in Balkans is located in Greece, FYROM and Kosovo. A part from the nickel producers, in the wider area of Balkans there are several companies which extract nickel ores and export it to Balkan metallurgical plants.

The biggest nickel consumer globally is Asia, followed by Europe. The steel industry global recovery, showed China as the greatest nickel user. During the last decade, there has been a 17% increment of the demand and is estimated on a long-term basis that the nickel consumption will be increaed 2 – 3% yearly. Of course in 2008 there was a 4.1% reduction in demand compared to 2007, but in 2009 the situation will be reversed and the demand is estimated to become positive again.

Because of high demand, nickel price has reached historic highest peak in 2007, about 800% increment compared to the nickel price ten years ago. In 2008 though, because of the global economic recession, there was a falling down of the nickel prices, which continues until today with a slight increasing in the last months of the 2009 year.

The economic recession of 2008 has led to production reduction and cancellation of investments in nickel industry. The effects of the recession are expected to last a few more quarters of the year. Future prospects of nickel and all metals in general, are connected to the success of the efforts to support the global economy in order to sustain a high level of demand.

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Gypsum in Albania and its Near Future Development Perspective

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ABSTRACT Simultaneously with the great increment of traditional minerals extraction in Albania as, chrome, iron-nickel, copper, decorative stones, etc., on the way of development is running also the industrial minerals mining and processing industry, particularly based on the high demand of building industry, mainly to fulfill the domestic needs and further to export.

In mineral mining and processing industry are operating by licences about 1200 private companies, for building materials, chrome ores, copper, clays, nickel, and only in the gypsum industry are operating about 15 companies one third of which is foreign.

The gypsum potential is relatively huge and the greatest advantage is that it is propagated in different areas of Albania.

Albanian government programme of mineral mining and processing industry intends to the incitement and support of private investments of domestic and foreign companies, for development of mineral extraction and particularly processing.

The Albanian gypsum mining and processing industry seems to be developed in a right way in the conditions of free market economy.

1 INTRODUCTION

Presently the mining industry is completely privatized. Particular positive is the fact of entering of domestic business in the mining industry which seeks not small investments.

Distribution of mining activity not only in the field of traditional minerals or to the building materials, but also to the all industrial minerals, most part of which presently are imported, constitutes one the priorities in the development of this industry, especially for the fact they could increase the export hugely

The fact that the mining industry has been linked with the rural zones, seeks a particular attention, taking into consideration the social aspect of the state as a possibility for new employment places, their development and integration possibilities of these zones with overall development of Albanian society.

The development of mineral mining and processing industry intends the consolidation of the reforms carried out in this field, incitement and supporting of foreign and domestic private investments and partially those public indispensable for the safe and harmonic development of the mining objects in operation and also determining the directions of mineral mining&processing industry through the policies of development of the economy as whole country and regions.

Gypsum is a mineral composed of calcium, sulfur, and oxygen. Its chemistry is $CaSO_4$-2(H_2O), Hydrated Calcium Sulfate, of Class of Sulfates and uses as plaster, wall board, some cements, fertilizer, paint filler, ornamental stone, etc.

Gypsum is one of the more common minerals in sedimentary environments. It is a major rock forming mineral that produces massive beds, usually from precipitation out of highly saline waters. Since it forms easily from saline water, gypsum can have many inclusions of other minerals and even trapped bubbles of air and water.

Plaster of Paris, a fine white powder, is produced by heating gypsum to expel the water. Its major use is in the manufacture of gypsum lath and wall board, and for casts and molds.

Uncalcined gypsum is added to Portland cement as a retarder.

Crystals of gypsum can be extremely colorless and transparent, making a strong contrast to the most common usage in drywall. The crystals can also be quite large. Gypsum is a natural insulator, feeling warm to the touch when compared to a more ordinary rock or quartz crystal.

2 PHYSICAL CHARACTERISTICS

Color is usually white, colorless or gray, but can also be shades of red, brown and yellow.

Luster is vitreous to pearly especially on cleavage surfaces.

Transparency crystals are transparent to translucent.

Crystal system is monoclinic; Crystal Habits include the tabular, bladed or blocky crystals with a slanted parallelogram outline.

The pinacoid faces dominate with jutting prism faces on the edges of the tabular crystals. Long thin crystals show bends and some specimens bend into spirals called "Ram's Horn Selenite".

Two types of twinning are common and one produces a "spear head twin" or "swallowtail twin" while the other type produces a "fishtail twin". Also massive, crusty, granular, earthy and fiberous.

Cleavage is good in one direction and distinct in two others.

Fracture is uneven but rarely seen. Hardness is 2 and can be scratched by a fingernail.

Specific Gravity is approximately 2.3+ (light). Streak is white.

Associated Minerals are halite, calcite, sulfur, pyrite, borax and many others.

Other Characteristics: thin crystals are flexible but not elastic, meaning they can be bent but will not bend back on their own. Also some samples are fluorescent.

Gypsum has a very low thermal conductivity (hence it's use in *drywall* as an insulating filler). A crystal of Gypsum will feel noticeably warmer than a like crystal of quartz.

Notable Occurrences include Naica, Mexico; Greece, Bulgaria, Romania, Italy, USA; and many other localities throughout the world.

3 GYPSUM DEPOSITS OF ALBANIA

In Albania there are distributed a lot of different sorts of the gypsums dhe anhydrites, and of huge reserves. They are either of old ages, Early Permian (Korabi and Ionian tectonic zones), nor of younger ages of Miocene (Littoral of Adriatic Sea). Sporadically the gypsums are located even in other regions in different geological conditions, but without any practical importance.

3.1. Gypsums of Korabi (North–East Albania)

In the region of Korabi (North-East Region of Albania), in the Diber District, the gypsums and anhydrites are outcropped in the ground surface in the form of two irregular outcrops in the East and North-East of the town of Peshkopia.

The outcrops have a thickness of 1000-1200 m and a length of up to 14 km, with generally step dipping, of white up to light grey colour.

In the great mass of gypsum, layers is observed the presence of the layers and lenses of anhydrite that are distinguished from the strengths and saccaroidal structure.

Among the gypsums and anhydrites of the Korabi Zone are distinguished also the alabaster layers, a sort of micrograined-saccaroidal gypsum used as decorative stone.

3.2 Gypsums of Dumrea (Central Albania)

In the region of Dumrea-Central Albania, which is included in the Elbasan District, there is an outcrop in the ground surface of 290 km² and representing a great diapir structure, there are observed of the gypsums and anhydrites, forming entire hills as that of Gradishta, etc.

The gypsums have white-grey colours with crystalline texture and knitting with anhydrites. In the more deep levels have been found, through the drillings (for oil exploration), also the rock salt layers.

Many of gypsum deposits and occurrences there are in the Adriatic-Sea littoral depression as in Durrës, Shijak, Kavaja and up to Vlora, located in the form also of lenses among the argillous-aleurolites of Upper Miocene age.

The gypsum in these places is of big-sized crystals and rarely micro-grained. Among the layers are observed intercalations of filamentous gypsum that is called selenite and multi-grained gypsum called alabaster.

In the region Golem-Kavaja, in the Saranda, the gypsums are accompanied with rock salt deposits in the form of layers and also of diapirs.

Table 1. Data on of Albanian gypsum deposits.

No.	Description	Peshkopi-Diber	Mengaj - Kavaja	Bistrice - Saranda
1.	Geol. Reserves, million tons	**2.7**	**2.25**	**1.4**
2.	Volumetric Weight, ton/m³	**2.2 - 2.92**	**2.23**	**2.1-2.27**
3.	Content of $CaSO_4\ 2H_2O$, %	**>92**	**90-98**	**91**
4.	Compression Strength, kg/cm²	**200-880**	**140-240**	**160-220**
5.	Moisture, %	**6.21-0.49**		
Explored Gypsum Reserves 85 million tons				
Potential Gypsum-Anhydrites Reserves about 50 billion tons				

The gypsum mineral is met also in the region of Librazhd (Central-South), in the Kolonja district, in the place of Tri Urat, etc.

Total geological reserves of gypsum in Albania are about 300 million tons with the content of 88-98% $CaSO_4$-2(H_2O).

Known from carried out geological workings are about 82 million tons of above mentioned quality.

4 MINING AND PROCESSING OF GYPSUM IN ALBANIA

Presently in Albania are operating about 20 companies obtaining the licenses of prospection & exploration and mining rights.

Table 2. Companies of gypsum mining and exploration.

	Licenses according to Districts						Σ
	Kavaja	Elbasan	Vlora	Saranda	Diber	Durres	
Mining	4	3	2	1	1	1	12
Prosp.-Explor.	2	1	1		2		6

They are concentrated and producing small quantities of gypsum mainly in the region of Kavaja, Elbasan –Belsh, supplying the present cementeries in Elbasan and Fushe-Kruja.

Actually in Albania the gypsum processing industry is not developed and only some small primitive plants that are manufacturing the plaster of Paris

Table 3. Production of gypsum mining companies.

	Year,			
	2005	2006	2007	2008
Export	-	3,000	1,200	11,953
Production	14,770	19,693	46,200	77,633

5 USAGE OF GYPSUM

Some of gypsum deposits are mined to use the mineral as raw material for the cementeries and also for production of plaster of Paris.

In some cases it is used as decorative stone cut and polished.

Nowadays, in the existing cement plants of Fushe-Kruja and Elbasan, is consumed an amount of not more than 30-40 thousand tons gypsum per year.

Albanian building industry is supplied presently from the import with the gypsum materials as dry systems, which include various using fields as walls, ceilings and floors.

Within these fields there are a lot of different systems, conditionned from the physical-construction requirements as, fire-proof, noise-proof, noise-absorption, thermal-isolation.

Total import volume purchased in Albania is about 1.5 million square meters, and most part of it is transported from Macedonia.

6 THE PERSPECTIVE OF MINING AND PROCESSING OF GYPSUM

Article 7 of Albanian Mining Law that: "The State incites the development and modernization of mining sector in Albania, in the conditions of market economy, based in the competition and free initiative".

According to the investments presently initiated in the country could be said that are under costruction three cementeries by foreign companies.

They are concentrated in the Central-North part of Albania, utilizing the huge reserves deposits of high characteristics of limestone, clays and additives like gypsum.

Taking into consideration the demand of other construction works for gypsum ready-made products it is forecasted that the total production for domestic needs of Albania could reach the level of about 1.0 million tons per year, besides the demand of export in neighbor countries and in Europe.

Table 4. Cementeries of Albania.

Seament - **Elbasan** Existing& capacity	Fushe Kruja Cement Existing & Reconstruct.	Antea Cement Greece Under Constructio n	Colacem (FaSa) Italy Under Constructio n	Cemento Aguillas-Spain Under Constructio n
Forecasted Production Capacity, million tons/year				
0.2 & 0.6	0.3 & 1.3	1.3	1.3	1.2
Forecasted Gypsum supply, thousand tons/year				
16 - 48	24 - 104	104	104	96
Total Gypsum Needs, 40 – 456 thousand tons/year				

Another issue is the environment rehabilitation which is an obligatory task of the mining&processing company. It will increase the costs of mining and consequently will decrease the interest for investors in the Albanian mining industry.

By the intention to increase this interest, the only way to incite the investments could be the facilitating of present fiscal package.

This principle is one of main standards of EU: "Strong rules for protection and rehabilitation of environment (and as compensation for it), facilitating of fiscal package (obligations-duties to the state) to the administrators of mineral wealths".

7 CONCLUSIONS

Albania is rich on the gypsum&anhydrites minerals deposits throughout its territory as, North-East, Central-South, South-West.

Its geologic reserves explored up to now by Geologic Survey of Albania workings are about 85 million tons.

By very few and simple geologic workings the mineral reserves could be explored, up to not very deep levels and only by open-pit mining, about 300 million tons gypsum ore, while the potential of gypsym-anhydrites of Albania is evaluated to be about 50 billion tons

Increasing demand of domestic industry on gypsum products will create a very favorable climate of effective investments in the geology, mining and processing of gypsum-anhydrites in the country.

Further on, due to the cheaper manpower and other facilitated fiscal packages, the road network improvements, the investments, particularly from well-known foreign companies, will be attractive and intensified to build and install gypsum production plants to supply not only the domestic needs but also a part of the European countries.

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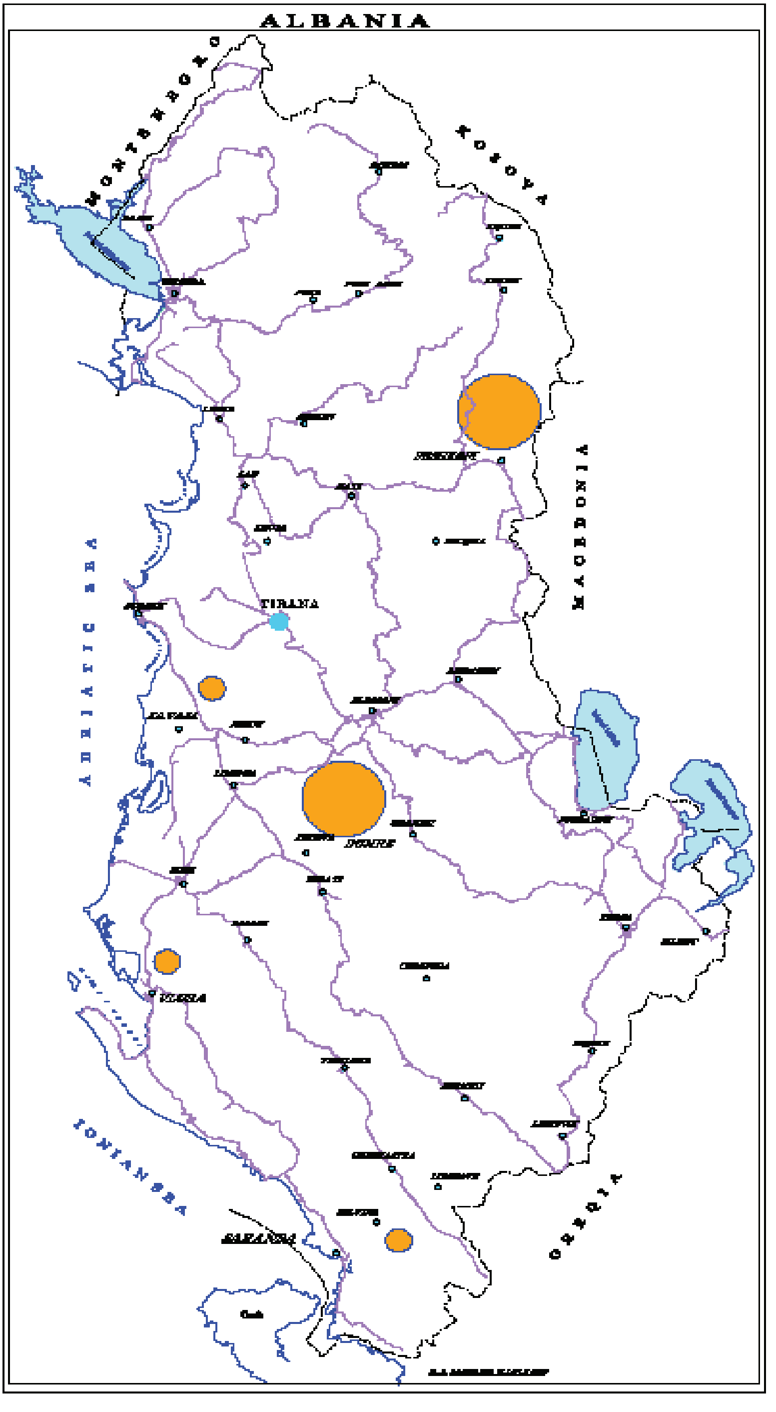
ALBANIA
MONTENEGRO
KOSOVA
MACEDONIA
ADRIATIC SEA
IONIAN SEA
GREQIA
TIRANA

Gemstone Mining History and its Importance for Environmental Economics of the Şaphane Mountain Fire Opal Deposit (Yeni Karamanca-Şaphane-Kütahya)

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ABSTRACT Fire opals (quartz group) being used in gemstone trade, which are the second one of the most important industrial rough material of Turkey are found in the Şaphane Mountains where is situated between Şaphane and Simav Districts of the Kütahya Province, and nearly in the North of the Yeni Karamanca Village. At the same time, this deposit is in north block of the Simav Graben, and is also adjacent with Şaphane alunite ore deposit in its near the east.

In the view of the geological formation, the fire opals were formed as a result of the precipitation of the silicic acid rich hydrothermal solutions raising thoroughly a secondary fault zone with N-S striking crosscutting the great Simav Fault, under relatively lower pressure and temperature conditions, in the gas cavity and vacancies of the rhyolite and consolidated tuffs.

In the case of geological material, this deposit including the fire opal gem roughs with being in demand for their size and color saturation is the unique in Turkey, and at the same time, is the second one after Mexico according to their mineral reserve.

The existence of the fire opals in this region, even though they have been known since the Lydia Period, was in perceptible by German miners during the extraction of the alunite ore in the near vicinity at the 19th century in the meaning of modern. Especially, between 1914 and 1919, the fire opal deposit was exploited by German miners together with Turkish villagers, and the gem roughs in a huge amount were exported to Germany in those times. Environmental economics was gained with large working moneys of the Turkish miner villagers during exploiting of both fire opal gem material and alunite ore. Then, the deposit was abandoned until the end of the World War II. In the various time portions from the War I to the present day, because of being unknown their gemological rough material value in Turkey, fire opal gem roughs in many amounts have been illegally exported to mainly Germany and other countries. The fire opals, which were discovered their beauty and rarity by the mineral collectors and the mineral museums, were expensively sold in especially Munih (Germany), Basel (Switzerland), and Tucson (USA) mineral fairs.

Now, the deposit held as if it is a borax ore field by Etibank (the official mining company) have been not already exploited. However, if the Şaphane Mountain fire opal deposit is exploited by somebody, its economics would be an important source of income for both near villagers and Turkey. However, the deposit must be exploited with a 5th Group mining licence according to the current Turkish mining law.

It is determined that the average tenor is 30-40 gr/m^3 in only surface width of 5.000.000 m^2 of whole deposit and in depth of 10 m (specific gravity value is about 2.1 gr/cm^3 for opal). According to this, visible reserve of the Şaphane Dağı fire opal deposit is about 1.750 tons for gem rough.

In the world gemstone market, rough price for fire opals is the ranging 5.000-10.000 kg per USDollars. If it is considered the average price, total value of the deposit is accounted as about

13 billion US Dollars, so, its addition to the environment economics of this value is pretended not to see if it is exploited.

1 INTRODUCTION

Opal ($SiO_2.nH_2O$) is an amorphous (opal-A) or poorly crystalline (opal-CT and opal-C) hydrated silica. Both volcanic-igneous (opal-CT and opal-C) and sedimentary origin (opal-A) opal formations are very widespread in nature. Therefore, natural hydrous silica phases are subdivided into three well-defined structural groups; opal-C (disordered α-cristobalite), opal-CT (disordered stacks of cristobalite and tridymite like layers) and opal-A (highly disordered, amorphous), even if opals -CT and -C form is a continuous series. These silica phases are main opalline quartz building phases. Opal is not a pure form of silica as it contains water as a component, and some impurities and trace elements can also enter its structure. (Elzea and Rice, 1996; Gaillou et al., 2008; Jones and Segnit, 1971).

Although various types of opals are found formed in volcanic-igneous rocks at low and moderate temperatures, the formation of fire opals is rare in nature. There are only a few locations in the world reported to have fire opals; Mexico, Brazil, Hungry, Kazakhstan, Ethiopia, Slovakia, and the USA (Erel et al., 2003; Fritsch et al., 1999, 2002 and 2006; Gaillou et al., 2008; Gübelin, 1986; Koivula et al., 1983; Ostrooumov et al., 2007; Ramirez, 1995; Rondeau et al., 2004). The fire opal deposit in the Kütahya region of Turkey is the unique fire opal deposit in the Anatolian land (Fig. 1), which has been mined for a long time to make gem objects. However, there are only three studies reported on the fire opals from Turkey (Andaç et al., 1976; Esenli et al., 2001, Hatipoğlu, 2009).

In this study, we mainly aim the announcement of the existence, mining history and economics of the currently unexploited Anatolian fire opals to be precious industrial gem rough material, in addition, contributing their mineralogical aspects and also geological formation.

2 LOCATION

Anatolian fire opals deposit are found in the Şaphane Mountains where is situated between Şaphane and Simav Districts of the Kütahya Province, and nearly in the North of the Yeni Karamanca Village. In addition, this deposit is in north block of the Simav Graben, and is also adjacent with Şaphane alunite ore deposit in its near the east.

3 MINERALOGICAL ASPECTS

Anatolian fire opals have various colorations; brownish-red, deep or light orange, yellow and colorless.

They contain mainly moganite and quartz inclusions (Hatipoğlu, 2009). Inner structure of the fire opal investigated by a scanning electron microscopy consists of nano-sized (10-60 nm in size) silica opal building components (Hatipoğlu, 2009).

The refractive indices of the fire opals are measured to be between N=1.408-1.414 for the transparent samples and N=1.428-1.436 for the translucent samples having massive zoned inclusions. However, the refractive index of the transparent colorless sample is found to be N= 1.392.

The specific gravity values of the fire opals are found to be between SG=1.98-2.08 for the transparent samples and SG=2.12-2.17 for the translucent samples depending on massive zoned inclusion contents.

The fire opals do not give out any fluorescent colors when exposed to long (366 nm) and short (254 nm) wave ultraviolet light. However, the transparent white variety of the fire opals, which has good play-of-fire in sunshine, indicates strong whitish-green fluorescence in short wave ultraviolet light and weak whitish-green fluorescence in long wave ultraviolet light.

The dimensions of the fire opals in this deposit vary from the size of a lentil to a walnut (2 mm to 40 mm in diameter). They are generally found as chick-pea shaped nodules.

In the region, there are also a number of alunite ore deposits and mines not far from the fire opal deposit. Thus, the effect of the alunite mineralization ($KAl_3(SO_4)_2OH_6$) is also reflected in the chemical contents of the Anatolian fire opals, with higher ratio of Al_2O_3 and Fe_2O_3 (Andaç et al., 1976).

The thermogravimetric (TGA) tests show that total mass loss percentage for the fire opal was found to be less than 8%, in contrast to those of other common and sedimentary opals (up to 21%), in addition, that care should be taken not to exceed temperatures above nearly 342 °C for fire opals during the gem cutting and polishing processes, because the value is the starting points of decomposition.

4 MINING HISTORY

The most famous deposits of fire opal which was explored around 1860's are in the state of Queretaro in Mexico. These deposits also are the most commercially viable and produce what has become known throughout the world as Mexican opal (Koivula et al., 1983, Gübelin 1986, Spencer et al., 1992, Ramirez, 1995).

It is officially known that the fire opals in Turkey have been mined as gemstone for over 100 years in this region, even though the existence and/or mining of them have been known since the Lydian Period.

Legends holds that the Lydian mined opal in Turkey in ancient times, and it was worked by the Genoese 550 years ago—there is still one mine called the Genoese mine in the region. Commercial mining of alunite was commenced in the 1850s. However, during this period some mining companies, especially those based in Germany realized the gemological importance of the fire opal deposits nearby. Both reports from local people and historical documents establish that the Germans intensively mined fire opal from the galleries and fracture zones that were opened during World War I between 1914 and 1919. Mining continued intermittently in the 1940s, 1950s and 1980s (Andaç et al. 1976, Esenli et al. 2001). Millions of tons of rhyolitic waste can be seen today as evidence of past episodes of intense mining activity in search of fire opal. Local free-lance miners still recover it from the dumps and sell it abroad, where it is unfortunately still marketed as Mexican opal, although some collectors are aware of the true locality (Fischer, 2007).

5 GEOLOGICAL FORMATION

Turkey, on the Minor Asia land, quite possibly has the greatest resources of natural gemstone material in the whole of Europe, the Middle East, and North Africa. This directly results from the fact that the Anatolian plate sits at the intersection of the four other continental plates. As a consequence, magmatic and volcanic activity directly related to the active tectonic zones near the plate boundaries have been ongoing, virtually continuously, since the Mesozoic Era.

The Anatolian fire opal samples are found in partially altered rhyolitic lavas and siliceous solidified tuffs (Andaç et al., 1976; Esenli et al., 2001, Hatipoğlu, 2009). It is likely that hydrothermal fluids caused the alteration and accumulation in the rhyolite and tuff host rocks in the formation of the nodule shaped Anatolian fire opals. Rock forming minerals such as quartz, feldspars, amphiboles and micas experienced hydrothermal alteration, releasing silicic acid (H_4SiO_4) into the solution during the deposition of the opals. Thus, the dissolved silicon ions were transported through the cracked zone. These fluids then combined with ground waters. The co-action of silicic acid (H_4SiO_4) must have also been present during the formation of the fire opals, which coagulated as colloids into the vesicles and pores of both the altered rhyolite and siliceous solidified tuff, as nodules from the size of a lentil to a walnut in a relatively low pressure and temperature environment with sufficient pH conditions.

In the area there is a large tectonic belt trending in a west-northwest/east-southeast direction, which is about 100 to 120 kilometers long and 40 to 50 kilometers wide.

The Simav, Gediz, and Emet Faults lie within this belt and these faults form the boundaries of the grabens in the same region (Işık, 2004). It was observed during the field studies that the Anatolian fire opal occurrences are related to the small N-S extending transfer fault which cuts across the NW-SE running Simav Fault.

6 ECONOMICS

Although the geological history of Anatolia has not produced deposits of precious industrial rough materials such as diamond, ruby, sapphire, emerald, alexandrite, or topaz, it has produced many occurrences of relatively semiprecious industrial rough materials as gemstones that have been traded worldwide. Consequently, fire opals (quartz group) being used in gemstone trade, which are the second one of the most important industrial rough material of Turkey.

According to the detailed geological field investigations made in the region, visible reserve of the Şaphane Dağı fire opal deposit is about 1.750 tons for gem rough. It is determined that the average tenor is 30-40 gr/m^3 in only surface width of 5.000.000 m^2 of whole deposit and in depth of 10 m (specific gravity value is about 2.1 gr/cm^3 for opal). Thus, it can be stated that Turkey is the second important fire opal producer in the world after Mexico.

In the world gemstone market, rough price for fire opals is the ranging 5.000-10.000 kg per US Dollars. If it is considered the average price, total value of the deposit is accounted as about 13 billion US Dollars.

7 CONCLUSIONS

Hydrothermally deposited Anatolian fire opals are found as nodules within the shrinkage and dehydration cracks of rhyolitic lavas and siliceous solidified tuffs, and their occurrences are related to the small N-S extending transfer fault which cuts across the NW-SE running Simav Fault.

Visible reserve of the Şaphane Dağı fire opal deposit is over 1.750 tons for gem rough, and total value of the deposit is accounted as over 13 billion US Dollars. According to this fire opal reserve, Turkey is the second important fire opal producer in the world after Mexico.

During the extraction with mining methods of the opals, any dangerous residues are not left to the environment.

The deposit must be exploited with a 5th Group mining licence according to the current Turkish mining law.

Now, the deposit held as if it is a borax ore field by Etibank have been not already exploited. However, its economics would be an important source of income for both near villagers and Turkey, if the Şaphane Mountains fire opal deposit is exploited by somebody.

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Economic Productivity in Turkish Boron Mining

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ABSTRACT Within this paper, it is aimed to propose the production productivity of the effective parameters in boron mining industry in Turkey. Firstly, the changes in the parameters in terms of production costs, production quantity, labor wages, selling prices etc. in Turkish boron mining industry are investigated between the years 1980 and 1998. Secondly, the productivity measurement in boron mining industry is evaluated by calculating Schmookler index, based on Cobb-Douglas production function as in indexes and it is statistically analysed for determination of the productivity effecting factors. In the statistical analyses, the linear regression-correlation and variance analyses work out between unit production factor indexes and production productivity indexes. Consequently, effective parameters of the production productivity in Turkish boron mining sector are determined in terms of invested capital expenditure, labor wages and labor productivity.

1 INTRODUCTION

Turkey is one of the few countries that possesses a vast and diverse mineral resource base. The availability of this wide range of minerals contributes to Turkey's industrial development through the supply of raw materials for its manifacturing industry. Turkey contains the largest rate of the world's known boron mines. Turkey, being first in the world production, has the largest and highest quality reserves. Turkey is the second in production capacity, following US. The reserve base contains all the economically mineable boron grades. Three major minerals of boron ore production in Turkey are colemanite ($Ca_2B_6O_{11}.5H_2O$), ulexite-probertite ($NaCaB_5O_9.8H_2O$-$NaCaB_5O_9.5H_2O$) and tincal ($Na_2B_4O_7.10H_2O$) ores. The largest tincal deposits are located in Eskisehir, Kırka region. Colemanite ore is obtained in Emet region of Kütahya, Bigadic-Balıkesir and Mustafa Kemalpasa-Bursa. Ulexite is produced in Bigadic, region of Balıkesir. Total production capacity of Bigadic, Emet, Kestelek and Kırka is more than 1,5 million tons of borate ore. All borate plants in Turkey are implemented by the state-owned company, Eti Holding. Eti Holding owns nearly 60%-70% of the world's known boron reserves on B_2O_3 content base.

Boron is an important material for the industry and it has a wide utility areas extending from glass to soap and detergents, fertilizer and agricultural chemicals to heat and flame resistant materials and to nuclear applications. Boron products also have wide range of utility areas and therefore they are significantly important for technology. (Eti Mine, 2003).

Additionaly, except from the known boron reserves and boron ores which are under operation, by the exploration program conducted in 2002 and 2003, the drilling activities have determined 1 billion m.tons of boron reserve, which means that, Turkey has 72% of the total boron reserves on earth. (Eti Mine, 2003)

Eti Holding operates mineral processing plants at Bandırma and Kırka. Eti Holding also produces concentrates of all the minerals in several grades, and refined products of sodium borates (borax decahydrate, Etibor-48 borax penta-hydrate, sodium perborate, Etibor-65 anhydrous borax and borax acid.) The refined products are produced at the plant which is also a shipping point for export to the US and Europe, in Bandırma. (www.maden.org.tr, March 2005)

Productivity is defined as the relation between the produced amount of products in a production period and the amount of resources consumed for the production or generation of these products. Generally, it is aimed to increase the productivity rate as much as possible. All the actions related to this issue have characteristics to increase the economic efficiency. Namely, it is necessary not to use unneeded data in input and it is also necessary to avoid input loss. In output, it is necessary not to produce unneeded products and service production. Only the required products and, the products and services needed by the community should be produced.

Newcomb (1968) states the economical productivity development of iron ore in the U.S.A., between the years 1925-1962. In this study, the usage of the model developed by using the Schmookler productivity index as a basis for measuring the economical productivity in mining is proposed. Stiroh (2001), employed a Cobb-Douglas specification to the extension of the neoclassical model allowing for a better understanding of the growth process. Chezum and Garen (1998), put their application to coal mines by using Cobb-Douglas production function to test the hypothesis, hat a spurious correlation between unions and productivity may emerge, in case the unions tend to organize the exogenously 'more productive' firms in industry. Shadbegian and Gray (2005), studied about pollution abatement expenditures and plant-level productivity by estimating a Cobb Douglas production function to measure the contribution of capital, labor, and materials input to output. . Pina and Aubyn (2005), while comparing the macroeconomic returns on human and public capital, estimated a Cobb-Douglas production function.

In this paper, the efficiency of usage of the effecting factors in production of boron mines is evaluated. Firstly, changes in the production factors have been investigated in Turkish Boron Mining between the years, 1980-1998. Secondly, the productivity measurement has been carried out by using Schmookler index, based on Cobb-Douglas production function base which is often used in the literature, and thirdly the statistical analyses of the relation between the production factors and the output obtained by these factors, namely the statistical analysis of the production functions is studied.

2 CHANGES IN THE PRODUCTION FACTORS

2.1 Production Quantities and Selling Prices

Several Boron minerals produced in boron mines in Turkey are firstly enriched and refined. Next, refined and enriched boron is exported to foreign countries or it is purchased by domestic markets. Total quantity of boron products sold between the years 1980 and 1998 is shown in Figure 1. As seen in Figure 1, boron content with average grade of 40.4% B_2O_3 and 34.85% B_2O_3 was sold 804,816 tons and 1,439,184 tons between the years 1980 and 1998, respectively.

The selling price of boron products was 115.1 USD/ton in 1980. However, the selling price of boron products did not have remarkable high increases up to year of 1989. Even though selling price was sharply increased up to 217.1 USD/ton, the price started to decrease after 1990. The selling price reduced to 153.5 USD/ton in 1998 as seen in Figure 2.

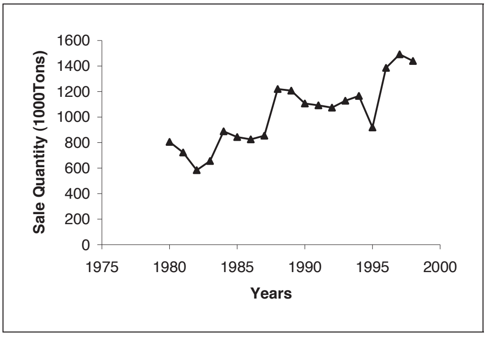

Figure 1. Changes in the sale quantity, 1980-98 (TURKSTAT, 2001).

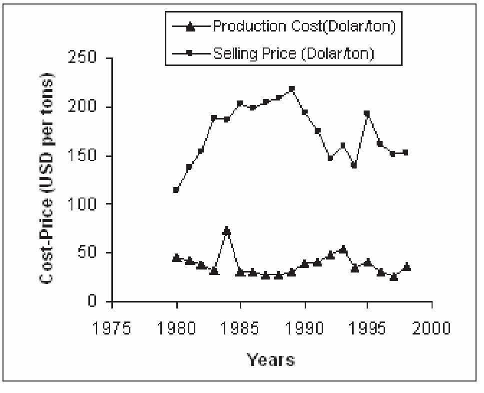

Figure 2. Changes in the production cost (USD per tons) and the selling price (USD per tons) (TURKSTAT, 2001).

2.2 Labor Parameters

Production methods used in the boron mines of Turkey are generally operated as surface mining. In recent years, underground mining methods used for boron has been abandoned. Although the number of the labor working in boron mines was 3829 people in 1980, -as shown in Figure 3, the number of labor decreased to 2488 people by 1998. Besides, considerable investment to open pit mine machines caused the reduce in the number of labor and increased the selling quantity and caused a regular raise in the labor productivity as shown in Figure 4. The labor productivity increased from 0.08 ton per hour to 0.483 ton per hour in 1980 and 1998, respectively.

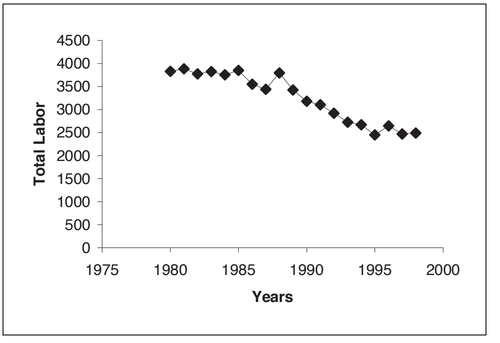

Figure 3. The number of the labor working in boron mines, 1980-98 (TURKSTAT, 2001).

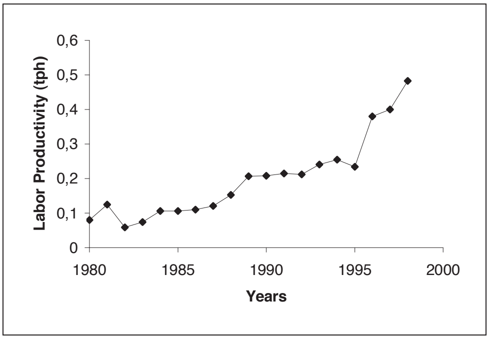

Figure 4. Labor productivity in Turkish boron mining, 1980-98.

When labor wage was about 2 USD per hour between years of 1980 and 1987, it started to raise after 1989, and became 5 USD per hour in the 1990's as shown in Figure 5. Due to the increase in the labor wages, the proportion of labor cost in total cost had been tried to minimize after 1988 by reducing the number of labor and implementing mechanized production methods, as shown in Figure 5. The consequences of these arrangements were first obtained in 1993. Whereas unit cost of labor was 30.6 USD per ton in 1993, it decreased to 14.9 USD per ton in 1998.

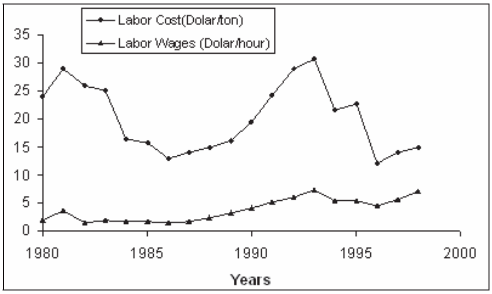

Figure 5. Changes in the labor cost and the labor wages, 1980-98 (TURKSTAT, 2001).

2.3 Production Cost

Unit-cost of production in Turkish boron mining sector is given in Figure 6. The biggest cost in the production cost is labor wages and other costs sequentially follow as material-energy expenses and the investment cost. The second biggest cost was material-energy expenses and third was the investment cost. Whereas total unit-cost of production was 46.3 USD per ton in 1980, it became 36.4 USD per ton in 1998 as shown in Figure 6. However, 47.2 USD per ton was spent for investment in 1984. Total production cost was also increased to 73 USD per ton in 1984. Boron plant built in Kırka-Eskisehir in Turkey, abnormally caused the increase of investment cost.

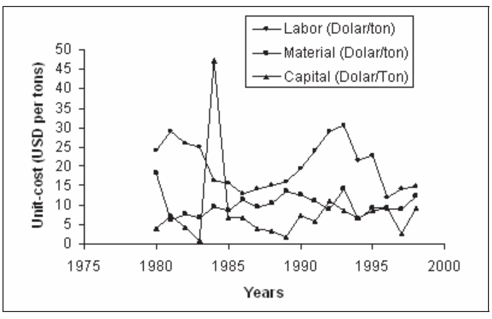

Figure 6. Unit-cost of production in Turkish boron mining, 1980-98 (TURKSTAT, 2001).

3 PRODUCTIVITY MEASUREMENT

Production functions are defined as the functions showing the relation betweeen production factors and the output obtained by these factors. Out of the many studies carried out about the improvement of the production factors, one of the most widely used production function for empirical estimation is the Cobb- Douglas production function (Ramanathan, 1995; Intriligator et al., 1996). In economy references, it is possible to find various functions developed on the basis of Cobb-Douglas production function as well as many studies which are carried out by using these functions.

Schmookler who studied the productivity measurement on the base of production function, formulates the productivity index as given below (Newcomb, 1968):

$$P_t = Q_t / \left(L_t^{W_l} . C_t^{W_c} \right) \quad (1)$$

When the aim is to analyse the total productivity, concerning that all input factors are included in the evaluation; the formula used by TURKSTAT is gained, via adding the material & energy costs to the Schmookler index. Consequently, all inputs and the ratios of all the inputs in total cost is included in the calculation.

In the studies using the data published by TurkStat as a base, productivity index on the base of production function is described as given below. This productivity index is used in the studies.

$$P_t = Q_t / \left(L_t^{W_l} . M_t^{W_m} . C_t^{W_c} \right) \quad (2)$$

Where;

P_t : Productivity index,

Q_t : Production quantity index (with constant prices) in the year t.

L_t : The index of the number of laborers in the year t.

M_t : The index of material-energy costs in the year t.

C_t : The index of the investment costs in the year t.

W_m: The ratio of material-energy costs in total cost.

W_l : The ratio of labour costs in total cost.

W_c : The ratio of investment costs in total cost.

The productivity index explained above, by altering the both sides of the equation to logarithmical form, can be defined as follow,

$$\mathrm{Ln}\,P_t = \mathrm{Ln}\,Q_t - W_l.\mathrm{Ln}\,L_t - W_m.\mathrm{Ln}\,M_t - W_c.\mathrm{Ln}\,C_t \quad (3)$$

to easy up the calculation.

The effects of total productivity factors on the development of production productivity can be determined by calculation of P_t for each t year.

Labor productivity index can be calculated as follows;

$$LPR_t = Q_t / L_t \quad (4)$$

4 PRODUCTIVITY MEASUREMENT IN TURKISH BORON MINING

The calculation of the index of the production factors between the years 1980-1998 with the constant prices of the year 1980 (Cubukcu, 2002) is given in Figure 7. Figure 8 shows the ratio of the production factors value in total costs. The productivity calculation is worked out by using Equation (3-4) and the total results are given in Figure 9. It shows the variation of the productivity index (P_t) and labor productivity index (LPR_t) depending on the years.

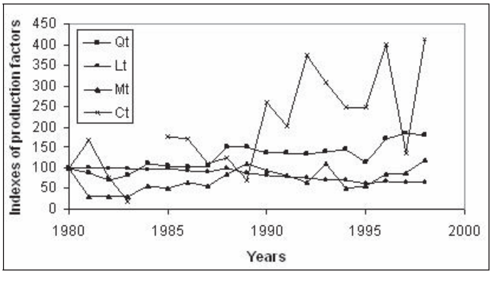

Figure 7. The indexes of production factors in Turkish Boron Mining, 1980-98. (C_{1984}=1311,8)

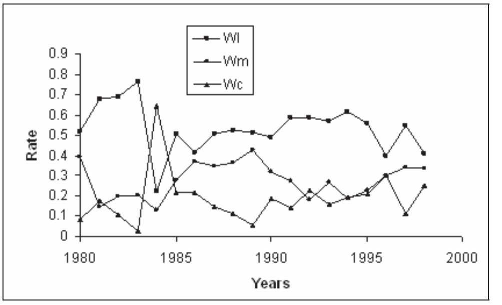

Figure 8. The rate of the production factors in total cost.

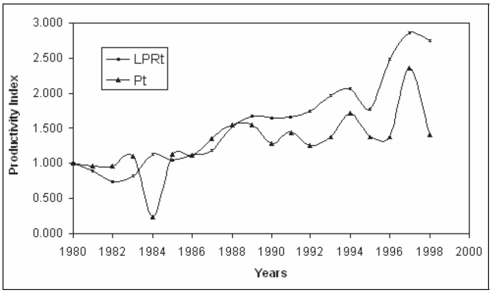

Figure 9. Production productivity index (Pt) and labor productivity index (LPRt) in Turkish Boron Mining.

Manufacturing Trade Materials Index is used to convert nominal values to real inflation-adjusted values.

As shown in Figure 9, the lowest value of production productivity is 0.23 in 1984, and the highest value is 2.36 in 1997. Generally viewed, it is seen that the production productivity increases. Although there is not a very clear increase in production productivity beginning from the year 1984, it is not lower than 1.00 and the graph generally shows that there is an affirmative development in production productivity.

The increase in labour productivity is more significant since it is not lower than 1.00 beginning from the year 1984, It even reaches higher values every year. The highest value is 2.868 in 1998.

5 STATISTICAL ANALYSE OF PRODUCTION PRODUCTIVITY

The statistical analyze for the functions of the relation between production factors and the output obtained by these factors is evaluated in two main groups as production productivity and labour productivity ordinary least squares technique is used in linear regression estimation.

5.1 The Effects of Unit Production Factors on Production Productivity

The results of the linear regression-correlation and variance analyses worked out between unit production factor indexes and production productivity indexes (Table 1) are given below in Table 2.

Table 1. Unit Production Factors Index Values in Turkish Boron Mining Sector (TURKSTAT, 2001).

Years	SP	LE	LP	LW	CE	LO	Q_t	L_t	Pt	LPR_t
1980	1.000	1.000	1.000	1.000	1.000	1.000	100,0	100,0	1.00	1.000
1981	1.200	1.203	0.636	1.891	1.887	1.563	89,7	101,5	0.96	0.884
1982	1.336	1.083	1.356	0.798	1.052	0.738	72,4	98,6	0.96	0.735
1983	1.636	1.042	1.077	0.967	0.254	0.925	81,5	99,9	1.10	0.816
1984	1.617	0.682	0.752	0.907	11.877	1.325	110,4	98,0	0.23	1.127
1985	1.759	0.660	0.750	0.879	1.686	1.325	104,7	100,5	1.12	1.042
1986	1.719	0.536	0.725	0.738	1.671	1.375	102,5	92,7	1.11	1.106
1987	1.772	0.588	0.658	0.894	1.036	1.513	106,1	89,9	1.35	1.180
1988	1.819	0.624	0.521	1.198	0.831	1.913	151,6	99,1	1.56	1.530
1989	1.887	0.674	0.386	1.745	0.469	2.588	150,1	89,5	1.55	1.677
1990	1.686	0.808	0.383	2.108	1.879	2.600	137,5	83,1	1.28	1.654
1991	1.524	1.004	0.371	2.709	1.489	2.688	135,6	81,1	1.45	1.672
1992	1.284	1.208	0.377	3.205	2.799	2.650	133,4	76,3	1.25	1.749
1993	1.389	1.277	0.331	3.852	2.194	3.013	140,1	71,1	1.38	1.970
1994	1.210	0.897	0.312	2.872	1.704	3.188	144,8	69,8	1.72	2.074
1995	1.671	0.949	0.341	2.783	2.168	2.925	114,2	64,0	1.37	1.784
1996	1.400	0.505	0.210	2.410	2.322	4.750	172,1	69,1	1.37	2.491
1997	1.320	0.588	0.199	2.952	0.732	5.000	185,3	64,6	2.36	2.868
1998	1.334	0.621	0.165	3.762	2.304	6.038	178,8	65,0	1.41	2.751

P_t : Production productivity index
LPR_t : Labor productivity index
SP : The index of unit ore selling price
LE : The index of unit labour expenses
LP : The index of unit labour period (working duration per tone (h/ton))
LW : The index of unit labour wage
CE : The index of unit capital expenses
LO : The index of unit labour output

Table 2. The results of the linear regression-correlation and variance analyses between values of unit production factor indexes and production productivity indexes.

Variables		Regression coefficients		Correlation coefficient	Variance analysis	
Dependent Y	Independent X	Constant a	Slope b	r	Calculated F	Relation between variables
P_t	SP	1.376	-0.056	-0.034	0.020	Insignificant
	LE	1.576	-0.340	-0.207	0.764	Insignificant
	LP	1.714	-0.761	-0.602	9.667	Significant
	LW	0.896	0.199	0.516	6.157	Significant
	CE	1.510	-0.106	-0.635	11.504	Significant
	LO	0.860	0.174	0.620	10.592	Significant

In the variance analyses the theoretical F_k value in 95% level, the theoretical Ft = 4,45, the relation is accepted to be significant when $F_h > F_k$, and insignificant when $F_h < F_k$.

$$P_t = 1.714 - 0.761LP$$
$$P_t = 0.896 + 0.199LW$$
$$P_t = 1.510 - 0.106CE$$
$$P_t = 0.860 + 0.174LO$$

According to the correlation coefficients, the parameters affecting the change of production productivity are unit capital expenses (CE), unit labour output (LO), unit labour period (LP) and unit labour wage (LW). It is seen that unit ore selling price (SP) and unit labour expenses (LE) do not have an effect on unit labour indexes. When unit labour wage (LW) and unit labour output (LO) increases, production productivity decreases. When unit capital expenses (CE) and unit labour period (LP) decreases, production productivity increases.

5.2 Effects of Unit Productivity Factors on Labour Productivity

The regression-correlation analysis and variance analysis results between the values of unit production factor indexes and labour productivity indexes are given in Tables 1 and 3.

Table 3. The results of the linear regression-correlation analyse and variance analyse between values of unit production factor indexes and labour productivity index.

Variables		Regression coefficents		Correlation coefficent	Variance analyse	
Dependent Y	Independent X	Constant a	Slope b	r	Calcuated F	Relation between variables
LPR_t	SP	2.311	-0.483	-0.188	0.625	Insignificant
	LE	2.239	-0.780	-0.307	1.768	Insignificant
	LP	2.531	-1.705	-0.870	52.810	Significant
	LW	0.628	0.483	0.805	31.333	Significant
	CE	1.622	-0.018	-0.071	0.085	Insignificant
	LO	0.536	0.423	0.972	294.941	Significant

$$LPR_t = 2.531 - 1.705LP$$
$$LPR_t = 0.628 + 0.483LW$$
$$LPR_t = 0.536 + 0.423LO$$

In terms of the correlation coefficients, the parameters affecting the alteration of labour productivity are unit labour output (LO), unit labour period (LP) and unit labour wage (LW). It is seen that unit ore selling price (SP), unit labour expenses (LE) and unit capital expenses (CE), do not have an effect on labour productivity indexes. When unit labour wage (LW) and unit labour output (LO) are in increase, the labour productivity index (LPR_t) is also in increase. When unit labour period (LP) is in decrease, production productivity is in increase.

6 CONCLUSIONS

When production productivity in Turkish Boron Mining is analyzed, it is seen that despite the decrease of production productivity in some years, there is a tendency to increase between the years 1980-1998. It is figured out that the common parameters which are effective on production productivity and labour productivity are labour effectivity, labour period and labour charges. In the case of the decrease of labour period, and increase of labour effectivity and labour charge; it is indicated that the productivity in production and labor will reach higher values. Additionally, the increase in production productivity is also possible with the decrease in investment decreases. In the productivity increase, the increase in effectivity related to the specialization of labour has also been effectual. The increase in unit labour charges related to the specialization and qualification has also created an increasing effect on productivity raise. Production and labour productivity have harmonic developments with each other after the year 1990 and the increase of labor is more than the increase of production productivity.

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3rd Balkan Mining Congress, BALKANMINE 2009, 1-3 October, İzmir-TURKEY, © 2009, ISBN 978-9944-89-782-2

Sedimentation Problems at Main Measure Regulation Stations

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ABSTRACT During gas pipeline system functioning there are different problems related to damages and defects of gas installation and sedimentataion appearing of certain elements. Reduction of gas installation crossing section which is resulting by its reducted runing power is caused by sedimentation. This problem is solved by temporary gas installation cleaning with so-called kracer. Gas installations which are out of function can be exposed to vastly sedimentation and even more to total block of rate gasification of flow. This paper is considered to similar case on gas installation main measure regulation stations (MMRS) Banatski Karlovac.

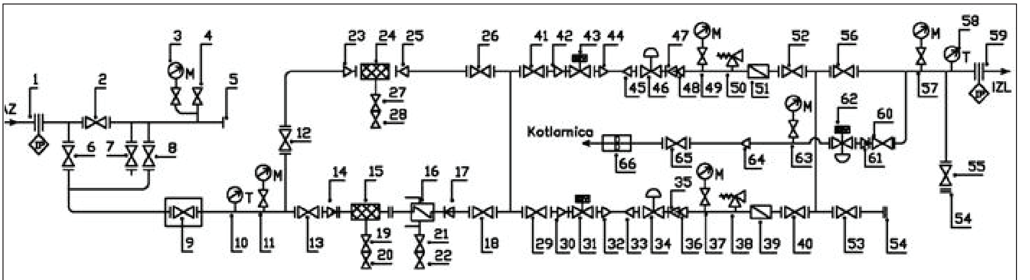
1 INTRODUCTION

Main measure regulation station (MMRS) Banatski Karlovac works over 20 years. MMRS is consisted of two lines, line I which enables gas flow without measurement and line II which enables gas flow with measurement. In Figure 1 is shown detail schema of gas installation MMRS Banatski Karlovac. MMRS started to work without installation of gas flow measurer and in that way worked line I. Necessity of gas flow measurement on MMRS demanded installation of gas flow measurer. After its installation (on line II) and gas hang out it is determined that there is no gas flow. In the first moment it looked like gas flow blockade caused by defect of valve. Gas is rerouted to line I, line II is opened so that gas flow measurer is uninstalling to check dynamic of valve. Valve was right, but firm sedimentation was created under this valve (Danilovic, 2008; Srbijagas 2008).

On installation MMRS Banatski Karlovac appeared sedimentation which totally reducted gas installation crossing section and blocked gas flow, Figure 2. Since gas installation line II has not been used during years, more than 20 years, for all that time sedimentation was created under this valve, V1 in service pipe of gas line I. After connecting gas line II with gas line I and its releasing to function it is determined that there was no gas flow. Uninstalling of gas line II resulted with creation of firm tube in sedimentation of unknown composition situated on indicated place. From that reason firstly was done the analysis of created sedimentation with the purpose to define its cause.

2 ANALYSIS OF DEPOSIT

Analysed material had dark brown colour, delicate oil smell and clammy brightness (Fig. 3). Small amount of fluid phase occurs on the surface of material. Washing of this material lead to remove of dark brown film of organic components (hydrocarbons) brining achromatic, completely transparent crystals obvious (Fig. 4).



1. Isolation joint, 4"
2. Valve, 4"
3. Manometer, 1/2", 0-40 bar
4. Valve 1", ANSI 300
5. Cliner box, 6"
6. Valve, 4"
7. Valve, 1"
8. Valve, 2"
9. Valve, 4"
10. Thermometer
11. Manometer, 1/2", 0-40 bar
12. Valve, 4"
13. Valve, 4"
14. Reductor, 31/2"/3"
15. Filter 31/2"
16. Heater, 31/2"
17. Reductor,
18. Valve, 4"
19. Valve, 6/4"
20. Valve6/4"
21. Valve
22. Valve
23. Reductor 31/2"/3"
24. Filter 31/2"
25. Reductor 31/2"/3"
26. Valve, 4"
27. Valve, 6/4"
28. Valve, 6/4"
29. Valve, 4"
30. Reductor, 4/2"
31. Flow gauge
32. Reductor, 4/2"
33. Reductor, 4/2"
34. Regulator
35. Reductor, 4/2"
36. Reductor, 6/4"
37. Manometer, 1/2", 0-10 bar
38. Reliable valve
39. Safe valve
40. Valve, 6"
41. Valve, 4"
42. Reductor, 4/2"
43. Flow gauge
44. Reductor, 4/2"
45. Reductor 4/2"
46. Regulator
47. Reductor 4/2"
48. Reductor 6/4/
49. Manometer, 1/2", 0-10 bar
50. Reliable valve
51. Safe valve
52. Valve, 6"
53. Valve, 6"
54. End joint
55. Valve, 6"
56. Valve, 6"
57. Manometer, 1/2", 0-10 bar
58. Thermometer
59. Isolation joint, 2"
60. Valve, 3/4"
61. Reductor 1"/3/4"
62. Regulator
63. Manometer, 1/2"Valve, 6"
64. Reductor 1"/3/4"
65. Valve, 3/4"
66. Flow gauge

Figure1. Schema of MMRS Banatski Karlovac.

Figure 2. Sedimentation place of origine on installation MMRS Banatski Karlovac.

Figure 3. Deposit from gas instalation.

Figure 4. Transparent cristals–solid base of deposit.

According to X-ray powder diffraction analysis (Philips PW-1710 diffractometer) mineral halite is determinated (Table 1).

Table 1. Results of X-ray diffraction analysis.

$d_{cal.}$	I	$d_{ref.}$ (ICSD 028948; 75-0306)	hkl
3.251		3.256	111
2.818	100	2.820	200
1.989	60	1.994	220
1.703	4	1.700	311
1.617	18	1.628	222
1.413	7	1.410	400
1.259	18	1.261	420

3 A CAUSE OF DEPOSIT ORIGIN

The cause of the deposit origin has been suggested on the basis of available technical documentation, direct observations and performed analysis of deposit (Danilovic, 2008).

The oil gas usually contents drops of salty water. These small amounts of salty water were precipitated on the place of lower rate gasification of flow inside installations. Owing to temperature changes of gas and pipes the concentration in this solution has been changed, also. At the applicable conditions which consider supersaturated solution, halite starts to crystallize.

The second important reason for sedimentation tube creation is position of gas inatallation, ie. service- pipe of gas line I in which was created sedimentation tube. Since service pipe of gas line I had vertical position, it was ideal place for sedimentation creation (sedimentation effect). Also, since gas installation line II has not been used during years, more than 20 years, for all that time firm sedimentation was created as tube which blocked gas pipeline flow (Danilovic, 2008).

Such sedimentation can not be created in functioning gas installations as to existing gas flow, what is very important for distributor and practicly confirmed. Also, sedimentation can not be created in gas installation horizontal part and that is confirmed by this case. There was no sedimentation in horizontal part of gas line II, which was not in function, but strictly in vertical part of gas line I.

Confirmed reasons of caused sedimentation suggest if it is possible to put horizontaly gas installation which do not function for longer period of time how to prevent sedimentation in pipe vertical part where is very suitably for sedimentation as tube. If pipe is put in horizontal position, created sedimentation can not form a tube and block gas flow.

In this mentioned case it was sufficinetly to substitute working and measurement line position how to put working line (line I) lower, and measurement line (line II) upside. In that way condensat collection under valve was disabled because it is situated under working line (Danilovic, 2008).

4 CONCLUSION

The case of firm sedimentation in gas installation is very rare in pratice. Because of that example of sedimentation created on installation MMRS Banatski Karlovac is particulary interesting. On this installation appeared sedimentation which totally reducted gas installation crossing section and blocked gas flow on line II which has not been used for more than 20 years. For all that time sedimentation was created under this valve, V1 in service pipe of gas with analysis of created sedimentation it is determined that mineral halite, which was caused by small amounts of salty water in gas and as part of gas installation position.

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Industrial Mineral Resources in Bulgaria

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"EUROTEST-CONTROL" AD

ABSTRACT: The industrial minerals and the production of construction and rock-lining materials occupy 45.3% of the mining production in Bulgaria and in 2008 they reached 48.14 million tons. According to the balance of minerals for 2008, the reserves of industrial minerals and construction materials are 22,404 million tons and guarantee further production for more than 4.6 centuries.

Bulgaria has approximately 250 deposits of industrial minerals, construction and rock-lining materials. More important of them are: kaolin raw materials, limestone for the chemical industry, bentonite clays, quartz sands for glass, limestone for the forage industry, limestone and marl for cement, limestone, dolomite and marl for crushed stone, sands and gravels for concrete, marble, gneiss, granite, schists for lining, and etc.
About 150 types of products are offered in the home market as well as for export.

1 INTRODUCTION

The production of industrial minerals in Bulgaria began in 1900. The first mine operated in Pleven region for opencast extraction of clays for bricks and tiles production. After the Liberation of Bulgaria in the period of industrialization, extensive geological surveys were carried out. Approximately 200 deposits of industrial minerals and inert construction materials are developped. The market offers about 100 types of products and materials which meet about 90% of the needs.

In the early 20th century the extracted industrial minerals were mainly related to construction, namely: building stone, roof tiles, gravel and sand, clays for bricks, tiles and pottery, limestone for lime production, mill stones, refractory earth, fuller's earth, ochre, etc. The construction of the Black Sea salterns and salt-works for rock-salt waters began. The production of table salt also commenced.

The initial extraction of gypsum was found in 1912-1965. The industrial production of kaolin raw material began in 1924 in the regions of Kaolinovo and Vyatovo. Kaolin is extracted from the kaolin raw material and is used for refractories, ceramics, faience, chamotte, fillers, and etc. The extracted sands are used for production of glass, faience, in the foundry, and etc.

Until the nationalization of mining industry (1948), new mineral deposits had been revealed and the extraction by reserved parameters and concessions of gypsum had began - in the town of Radnevo, of talc – in the village of Zhivkovo, of kieselguhr – in the town of Gotse Delchev, of barite - in the village of Kashana, of mica – in Pastra village, of fluorite – in Palat village, of chalk - in Nevsha village, and etc.

After the nationalization of mining industry in 1948, the geological and exploratory works significantly expanded by opening deposits of asbestos, limestone, dolomite, quartzite, feldspar, fluorite, flint, cement and other materials. After further geological and prospecting works the reserves increased and new mineral resources were found such as: magnesite, bauxite, fire clay, gypsum, rock salt, bentonite, limestone for flux, disthene, perlite, zeolite, etc. The production of industrial minerals, construction materials and rock lining materials increased hundreds of times.

After the political changes in 1990, early restructuring of the mining industry started together with the elimination of inefficient capacities, privatization of existing mines, flotation plants and the infrastructure around mineral deposits, transition to market economy principles.

The data (Table 1) for development of the mining production of industrial minerals and construction materials indicate three stages:

Stage 1: 1990 – 1995. Restructuring of mining production. Updating mining legislation, creating a new regulatory framework. Preparation of new conditions and formation of a new mineral balance.

Stage 2: 1995 - 2000. Adapting mining production to market demands, privatization of mining facilities, new rules for prospecting, mining discoveries, ecology of mining production, recultivation of lands damaged by mining activities.

Stage 3: after the year 2000. Conditions for sustainable development of mining production in a market economy are established. Development and implementation of high technologies. Quality management of mining raw materials and industrial waste. Adaptation to the European mining legislation and regulatory framework, integration with European mining organizations. New markets for industrial minerals and rock-lining materials. Changes in the education of mining experts – implementation of Bachelor's, Master's and Doctor's degrees.

Table 1. The mining production of industrial minerals and construction materials in the transitional period in Bulgaria, million tons.

No	Raw materials	Beginning of the transition 1990	Restructuring 1995	End of transition 2000	Sustainable development 2005	Production 2008
1.	Industrial mineral resources	7.2	4.3	3.7	8.9	8.84
2.	Construction materials	15.0	16.2	19.4	27.1	38.90
3.	Rock-lining materials	0.21	0.24	0.31	0.28	0.40
	Total %	22.41	20.74	23.41	36.28	48.14
	Production of all mining industry	73.5	72.1	75.2	87.5	106.2
	Participation in the mining industry of Bulgaria, %	30.5	28.8	31.1	41.5	45.3

2 MINING PRODUCTION

The mining production of industrial minerals in 2008 (Table 2) is represented by 22 kinds of industrial products: kaolin raw materials, bentonite clays, quartz sands for glass, rock salt, quartzites for refractory materials, limestone for chemical industry, limestone for forage industry, etc.

Bulgaria disposes of industrial mineral resources (Table 3) that are studied in detail. There are established reserves, which are not exploited in expectation of relevant investors interested in economically viable market production.

The mining production of building materials in 2008 (Table 4) reached 38.9 million tons, thus fully satisfying the needs of the construction industry. The existing established reserves of 5005 million tons guarantee the security of production.

The mining production of rock-lining materials in 2008 (Table 5) reached 187.8 thousand m^3, mainly limestone, marble, granite and granodiorites. The production of rock-lining materials, despite their high natural qualities, variety of colours and durability, operates under market competitive conditions. The products: marble floor, mosaics, tiles, curbs, cobbles and other stone items comply with the European and world standards and find a good market and implementation in construction and household.

Table 2. Mining production of industrial minerals for 2008.

№	Raw Material	Reserves, (1000 tons)	Production, (1000 tons)
1.	Kaolin	129 121.0	1 529.8
2.	Fire clays	6 166.2	9.9
3.	Stone clays	4 358.8	0.5
4.	Bentonite clays	26 269.3	178.7
5.	Refractory earth	3 233.2	0.3
6.	Rock salt	15.6 billion tons	2 100.0
7.	Limestones for chemical industry	501 334.2	3 447.0
8.	Quartzite - refractory	7 297.8	21.7
9.	Talc schists	38.1	0.6
10.	Quartz sands for glass	668.6	296.1
11.	Quartz, feldspar	1 361.4	4.5
12.	Quartz sands	82 224.0	734.5
13.	Dolomite for xylolite	7 953.6	21.5
14.	Gypsum	84 985.3	21.2
15.	Perlite	1 954.2	7.4
16.	Chalk white lead	6 667.5	0.4
17.	Limestone for forage	25 844.6	181.7
18.	Oil schists	706.9	1.0
19.	Flint concretions	285.7	1.1
20.	Oligomicts	2 067.3	3.9
21.	Sandstones for feldspar	24 732.3	0.9
22.	Potassium pegmatite	7 452.6	6.6

Table 3. Mineral deposits with industrial materials, which are not exploited.

№	Raw Material	Reserves, (1000 tons)
1.	Fluorite	950.7
2.	Baryte	216 476.8
3.	Limestone for flux	53 778.6
4.	Dolomite for refractories	15 559.4
5.	Quartz for quartz glass	73.2
6.	Sands for stoneware	166.9
7.	Vermiculite raw material	7 265.1
8.	Quartz sands for filters	1 946.2
9.	Bentonite for wine production	243.4
10.	Chalcedony cillicites	167.0
11.	Quartzites for metallurgy	574.0
12.	Asbestos raw material	32.8
13.	Smectite-illite aleurolites	4 903.1
14.	Quartz sand for filler	12 014.6
15.	Vein quartz for ferro-alloys	155.9
16.	Marble for chemical industry	17 263.1
17.	Bentonite for cosmetics	34.2
18.	Kaolinic clays	160.7
19.	Dolomite for fillers	2 079.4
20.	Vitrophyre for glass	1 689.8
21.	Bentonite for ceramics	165.0
22.	Zeolite	46 063.7
23.	Dolomite for metallurgy	33 319.8

Table. 5. Mining production of rock-lining materials for 2008.

№	Name of material	Reserves, thousand m^3	Production thousand m^3
1.	Limestone for facing	71 597.1	161.5
2.	Marbles for facing	70 266.9	17.9
3.	Granites and granodiorites	10 957.1	7.0
4.	Rhyolites for lining	3 666.0	0.7
5.	Gneiss for flooring	6 133.9	3.1
6.	Schists for lining	820.9	4.2
7.	Gneiss-schists	5 245.8	3.4

Bulgaria has large deposits of construction and rock-lining materials (Table 6) which are not exploited, awaiting their concessionaires.

Table 4. Mining production of construction materials for 2008.

№	Name of material, measure	Deposits	Production
1.	Limestone for cement, thousand tons	436 410.9	3 699.1
2.	Marls for cement, thousand tons	507 225.4	1 518.2
3.	Clays for cement, thousand tons	18 625.1	4.6
4.	Quartz sands, thousand tons	6 306.4	101.4
5.	Trass for cement, thousand tons	14 929.8	353.2
6.	Limestone for lime, thousand m^3	60 733.8	266.2
7.	Marble for lime, thousand m^3	21 713.6	243.9
8.	Clays for tiles, thousand m^3	16 022.1	15.7
9.	Clays for bricks, thousand m^3	89 480.0	124.6
10.	Marls for bricks and tiles, thousand m^3	14 049.6	271.2
11.	Marls for bricks, thousand m^3	48 366.4	237.4
12.	Loess clays, thousand m^3	4 363.0	3.0
13.	Clays for façade tiles, thousand m^3	8 410.3	2.0
14.	Clays for faience, thousand tons	2 489.0	4.8
15.	Marbles for mosaic and flour, thousand m^3	12 162.4	19.6
16.	Marbles for crushed stone, thousand m^3	142 147.2	1 197.5
17.	Limestone and dolomite for crushed stone, thousand m^3	774 232.7	6 739.3
18.	Rhyolites for crushed stone, thousand m^3	5 160.5	279.2
19.	Andesites for crushed stone, thousand m^3	346 440.7	1 647.5
20.	Sandstone for crushed stone, thousand m^3	2 275.2	56.8
21.	Sand and gravel for concrete, thousand m^3	396 887.4	5 516.6
22.	Limestone for road construction, thousand m^3	9 215.5	19.7
23.	Black schists for mosaic, thousand m^3	229.1	2.7
24	Clay schists, thousand m^3	10 787.3	23.4

Table 6. Deposits of construction and rock-lining materials which are not exploited.

№	Name of material	Reserves, x1000 m^3
	A. Construction materials	
1.	Sands for bricks	777.2
2.	Sands for calcareous sand bricks	5 869.9
3.	Marls for crushed stone	1 602.9
4.	Granites for crushed stone	20 720.6
5.	Metamorphites for crushed stone	2 409.0
6.	Tuffs for cement	24 565.3
7.	Sandstone for crushed stone	2 275.2
	B. Rock-lining materials	
1.	Breccia-conglomerates and marble breccia	16 744.1
2.	Quartz-monzonites for lining	10 272.4
3.	Gabbro, andesite, basalt	5 062.2
4.	Tuffs for lining	2 284.1
5.	Travertine for lining	52.7
6.	Travertine	41.3
7.	Sandstone for lining and articles	14 204.0
8.	Amphibolites for lining	2 694.3
9.	Gneiss and gneiss-schists for flooring	663.3

3 INFORMATION

Mining and geological information and technologies for exploitation of mineral deposits for production of industrial minerals, construction and rock-lining materials (exemplary).

Kaolin. The kaolinic deposits (Vetovo, Kaolinovo, Dulovo) are located in northeastern Bulgaria across an area of about 3 000 km^2. They are exploited according to the opencast method. The kaolin is obtained by flotation (washing) of kaolinic sands, containing 12 - 15% kaolin. The mineral content of the kaolin is: kaolinite 75 - 86%, muscovite 4 - 7%, quartz 7 - 13%, orthoclase 2 - 4%, albite 1 - 4%. It is used for the manufacture of porcelain, faience, ceramics, paper with 70% - 85% whiteness and yellowness up to 19%. The quartz sands, products of kaolin processing, are used in construction, metallurgy and refractory industry.

The kaolin raw reserves are about 95 million tons with 19 – 20 % concentration at annual production of about 1.6 million tons, which is guaranteed for 60 years.

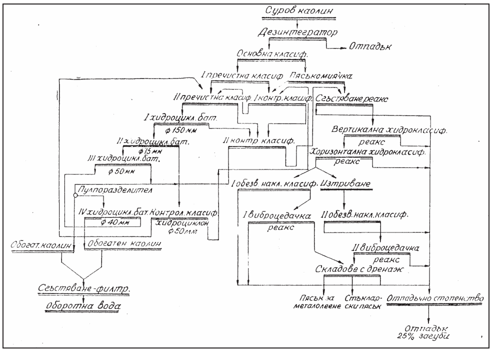


Figure 1. Scheme for ore-dressing of the kaolin raw material in "Kaolinovo" deposit.

Rock salt. The salt mine "Mirovo" has a diapiric salt structure in the form of incorrect truncated cone. It has been developed since 1956 through fluid mining from output boreholes for salt mass. Nearly 90 million tons of salt is obtained. Fresh water is used for solution of rock salt. The exploitation of the deposit is carried out at three horizons to a depth of 1000 m, 1250 m and 1750 m and 42 output boreholes (Fig. 2). The boring productivity is about 50 m^2/h salt mass, out of which 15.0 - 15.5 t / h salt is obtained. The boreholes have a diameter of 295 - 550 mm and are driven in a network at intervals of 200 m. The extraction chambers are 100 m in diameter and from 300 to 800 m in height.

One problem is the complicated geomechanical situation, caused by the applied mining technology. The state of the massif, the excavated chambers, the strained state of the rock-salt mass, the induced seismicity in the region, the sudden local earthquake shocks – all this has necessitated to work out a technology for safe and faultless operation.

The rock-salt reserves amount to 15.6 billion tons at annual output of 2 million tons. The production is guaranteed for 70 centuries.

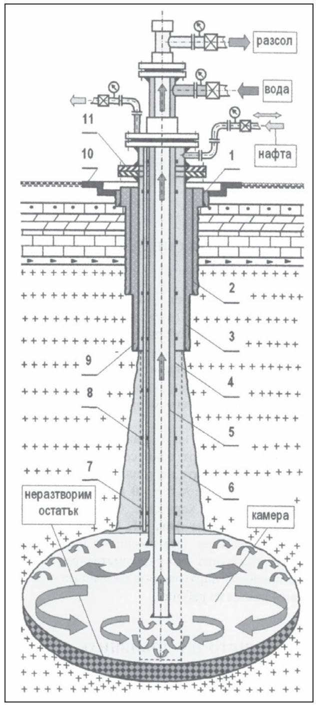


Figure 2. Salt mining borehole in "Mirovo" deposit.

The extraction and production of sea salt is carried out by salterns near the Black Sea towns of Bourgas and Pomorie by evaporation of sea water.

The average annual production of sea salt since 1986 is about 57 000 tons, and the highest yield was in 1994 - 74 000 tons. The marine salt extraction provides about 60 000 – 80 000 tons of lye for the roads as antifreeze and in medicine for production of toothpaste.

Due to the seasonal character of the works (around 6 months of the year), extremely hard work (everything is performed manually), and the technology depending on the rainfalls and evaporations, the production of Black Sea salt is ineffective in market conditions. The estimated (foreseen) capacity of the evaporative basins for Black Sea salt is 80 000 tons/year and depends on the needs, the import and export of crystal sea salt and possibilities for rock salt processing both for import and export.

Gypsum. "Koshava" gypsum deposit is located at a depth of about 300 m and represents a 20 m thick layer. The deposit is developed by chamber-and-pillar system. The excavated chambers are filled with sand, obtained from the Danube by dredge, stored on the mine yard and delivered to the chambers by hydraulic devices (Fig. 3). Chamber-and-pillar system of mining in "Koshava" gypsum deposit by filling of the chambers. 1 – the river Danube; 2 - main sands storage with capacity of 350 000 tons; 3 – dump-trucks transport at 800 m distance; 4 - temporary storage with capacity of 20 000 m3 sands; 5 - receiving hopper; 6 – rubber conveyor belt; 7 – shaker; 8 - mixing vessel; 9 - bunker for oversieve product; 10 – pulp-conduit; 11 - vertical shaft 300 m deep; 12 - horizontal workings up to 2 500 m long; 13 - panel gallery; 14 - chamber under filling; 15 – mining water-collector with a pumping station; 16 – water-conduit for leading mining waters away to the surface; 17 - surface reservoir with a pumping station; 18 - pipeline for passing water into the mixing vessel.

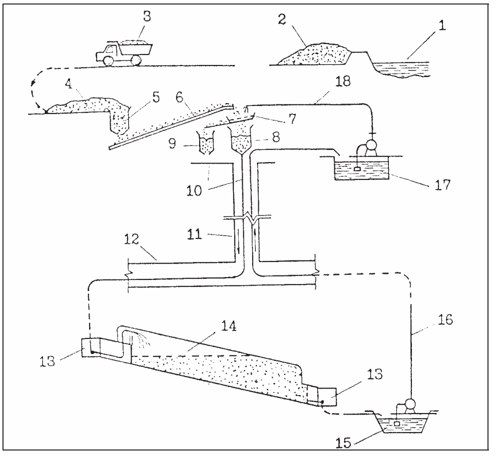


Figure 3. Chamber-and-pillar system of mining in "Koshava" gypsum deposit by filling of the chambers.

The parameters of the chamber-and-pillar system for the various mining sections are:

- thickness of the layer: 18 – 20 m;
- mining depth: 260 – 330 m;
- chambers width: 5 – 8 m;
- chambers height: 11 – 15 m;
- width of the solid blocks: 16 – 22 m;
- height of the solid blocks: 11 – 15 m;
- chambers length: 100 – 260 m.

The annual production of gypsum is 150 000- 200 000 tons, with achieved output over 400 000 tons (1983 - 1988). The reserves are about 90 million tons. About 70% of crude gypsum is used in cement production.

The geomechanical situation requires the development of a rational technology for underground mining, providing an effective extraction of gypsum and ensuring stability of the geomechanical complex.

Zeolite. The deposits are mainly of clinoptilolite type - for example, "Beli Plast" deposit near the town of Kurdzhali. Opencast method of mining is applied. Through modification of the Bulgarian natural zeolites, adsorbents for purification of industrial wastewaters are obtained. In agriculture, the zeolite is used for forage additives, fertilizers and feeding of poor soil materials as well as for recultivation of contaminated soils. It is used in industry as filler, in cosmetics and medicine, in the construction of ground coats, chinking and insulation materials.

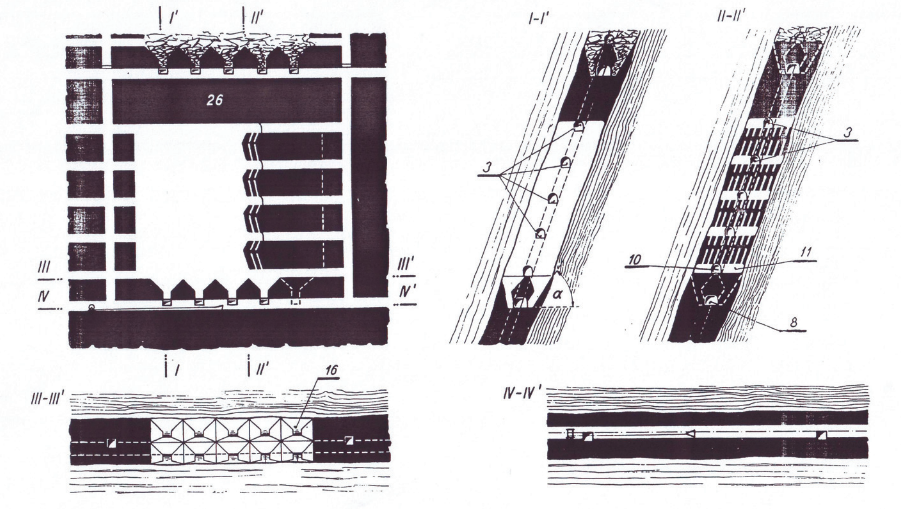
The zeolite reserves are over 37 - 40 million tons, and the resources - more than 1 billion tons (mainly clinoptilolite).

An analysis is made on the opportunities for the use of modified Bulgarian clinoptilolites in the cleansing of anions and adsorption of cations from industrial wastewaters.

Fluorite. The fluorite deposit "Lukina Padina" is located in the Chiprovtzi orefield. The ore bearing reserves (540 000 tons) are located in 4 ore bearing blocks with dimensions: length of 34m to 120m and thickness of 7m to 20m; content: CaF_2 of 28% to 48%, SiO_2 of 30% to 56 %, CO_3 of 4% to 18%. The deposit will be mined by a chamber system with sublevel galleries and a block system - by dispensing of the excavated ore (Fig. 4). Fluorite concentrate containing CaF_2 of 80% to 90% will be obtained from 80

- 90% extraction of the flourite raw material. It is applied in metallurgy (85%), chemical industry (98%), ceramics and glass-making (95%).

Perlite. The opencast mining method was applied on the perlite deposit "Schupenata Planina" to the south of Kurdzhali town since 1960. 17 perlite deposits with about 2 billion tons of reserves are located in the East Rhodopes region. According to its properties, the perlite raw material is: light, heavy or universal. The light perlites ("Schupenata Planina") reach compactness at expanding up to 300 kg/m^3 and compressive strength up to 5 MPa and can serve as thermo- and audio-insulation materials, filters, adsorbents, fillers for plastics, and etc.



3 - quarrying galleries, 8 - scraper horizon, 10 - gallery 11 - driving, 16 - draw-out funnels, 26 - subgallery pillar.

Figure 4. System of quarrying with sublevel galleries.

Decorative rocks for rock-lining materials. More than 70 deposits are developed, producing mainly limestone, marble, granite, granodiorites, rhyolites, gneisses, gneiss-schists, monzodiorites, breccias, gabbro, diabase, tuffs, travertine, kieselguhr, and etc. – all of them with very high decorative effect (Fig. 5). There are rock-lining deposits with more than 220 million m^3 of reserves, which satisfy the needs of slabs, curbs, cobbles, mosaics, mineral flour and domestic items in Bulgaria and for export.

Metamorphic rocks are located mainly in South Bulgaria, sedimentary rocks – in the Pre-Balkan region, the Moesian plate and the Rhodopes, magmatic rocks – in Srednogorie and Strandzha - Sakar mountains.

More than 120 types of decorative rocks are explored, including 50 types of marble, 30 types of limestone, 15 types of granite, gabbros, rhyolites, and etc.

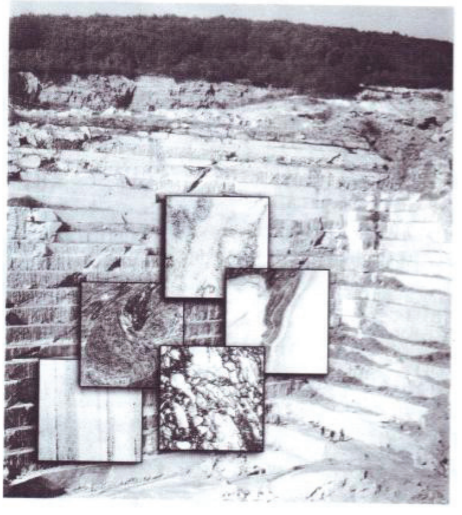


Figure 5. Quarry for extraction of rock-lining blocks.

4 CONCLUSION

The industrial minerals, construction and rock-lining materials are national wealth of Bulgaria. The scientific and industrial experience, gained in prospecting, exploration, design, construction, mining and processing of these materials, is a guarantee for the sustainable development of this production in a market economy. After the transition (beginning - 1990) from central planning economy to market economy (completion - 2000) in compliance with the world and European standards, the production of industrial minerals and construction materials is doubled (sustainable production - 2008), which guarantees the right course of development.

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Petrography and Geochemistry of Lignite, Kostolac Basin, Serbia: Importance to Mining

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ABSTRACT The Neogene Kostolac Basin in Serbia, one of biggest lignite basins in SE Europe, is an important energetic resours of Serbia. It is a base for operation two electric power stations (EPS) (Kostolac A and B). The basin is detailly geologically explored and is expoited only in its eastern parts and the major (next year the only one) lignite producer is the deposit Drmno, mining by the oppen pit (production 6 Mt/year 2008, 12 Mt/year 2012). The exploitable coal resources belongs to coal seam III (oldest of three ones in the basin), as thick as 18 m in average. Moister and ash contents and calorific values (CV) of lignite very significantly in the seam: in average moister around 40 %, and (on dry basis) 62 % organic matter and 38 % mineral matter («ash»). The firing technology in EPS is designed based on average values of these features.

1 INTRODUCTION

Coals are geological row materials which may vary significantly in its geochemical features as may contain some major and numerous trace elements harmful to the environment, as sulfur, arsenic, lead, nickel, etc. (Dangić, 1992, 1995, 2001, Dangić & Dangić 2007, Dangić & Dimitrijević 2001, Dangić & Putnik 1998, Mardon & Hower, 2004, Vassilev & Vassileva 1996, Ward, 2002, Zubovic et all. 1960). Lignites as low rank coals commonly appear to be massive mining and firing in electric power stations (EPSs). Due to enormous large masses of lignite firing in Eps, coal geochemical features of lignite may be very imporant for its utilisation in EPS.

The Neogene Kostolac Basin in Serbia (600 km^2), one of biggest lignite basins in Europe, is one of most signigicant energetic resourses of Serbia and is the base for operation two electric power stations (EPSs). At the present, major lignite resources in the basin are situated in the eastern part of the basin, in the Drmno deposit. The Drmno deposit is now the only one lignite producer

in the basin. Lignite is mining by an open pit wich is planed to be in operation up to 2042.

Lignite firing in EPSs produces large mases of waste solid and gasous products. The solid waste products are depositing into geo-media and appear to be sources of hazardous pollution of the surrounding environment (soil, ground and surface waters, air).

The previous study of lignite in the Kolubara Basin in Serbia (Dangić et al., 2007) indicated that variations in petrographic and geochemical faetures of coal may have very important impacts to economy and design of coal mining.

The complex petrographic and geochemical studies of coal seams in the Kostolac basin idicate that there are signicant variations of petrpgraphic and mineralogical composition and geochemical features (contents of heavy metals/harmful trace elements) of coal.

Due to these, a more racional coal exploitation is necessary based on more complete investigations of coal quality variations. It may enable better assessments of coal firing and its impacts to the environment and designs of related selective exploitation of coal.

2 GEOLOGY AND MINING FEATURES

The Kostolac Basin is located around 80 km at east of Belgrad, the capital of Serbia, and is situated at the right bank of the Danube River (at south of the Danube), between the Morava River, at west, and the Pek River, at east. It is belived that it is a souther part of a larger Plicene basin Kostolac-Kovin wich is intersected by the Danube River into the Kostalac Basin (at south) and the Kostolac Basin (at the north).

The Kostolac Pliocene Basin is as large as 600 km^2 and is morphologicaly divided into two units, the eastern and the western parts. The Pliocene sedimentary series of the basin contains three lignite seams, named as the coal seams I, II and III. The most widespread of them is the coal seam III, the oldest and deepest one.

The geological exploration and mining exploitation of lignite in the Kostolac Basin started more than hundred years ago. However, the exploration have been mostly concentrated to the eastern part of the basin and coal was and is mining in this part of the basin only. Most of the coal resources in the estern part of the basin belongs to the Drmno coal deposit.

The present and future operations of EPSs in the basin (up to 2042) are based on the resources of the coal seam III in the Drmno deposit and their mining by the open pit (OP) Drmno.

In the deposit Drmno, the coal seam III is as thick as 16-20 m and has a small slope toward NW and slowly decline toward the north (probably it exists bellow the the Danube River course). A part of coal resorces of the Drmno deposit could not be exploitated by the open pit as it is situated bellow the famous archeological site Viminacium, from the Roman time.

In the OP Drmno, from 1987, when started, up to the end of 2007, it was excaveted in total 83x10^6 t coal, and the present usable coal resorces are as much as over 400x10^6 t coal.

The avearge coal quality in the deposit is as follows: moister 39.22 %, ash 17.70 %, comustable matter 43.14 %, sulfur 1.18 % (comustable 0.56 %, in ash 0.60 %), UTE 11.49 MJ/kg and LTE 10.02 MJ/kg.

The complex gechemical, petrological and physicochemical studies of the coal seam III in the Drmno deposit (Fig. 1) have been carried out (Dangić, 2008), at the open pit profiles and several bore hole profiles.

3 MATERIAL AND METHODS

Both organic and inorganic matter of the coal seam III have been studied. Bulk lignite samples, represented large intervals of the vertikal profile of the coal seam as well as several individual samples and subsamples have been studied.

Field and lab studies involving macroscopic, microscopic and several physicochemical investigations have been carried out. In petrographic and mineralogical studies of coal and mineral matter the optical (binocular, polished and thin section microscopy) and X-ray diffraction (XRD) have been applied.

In geochemical studies, major and trace elements have been analysed by X-ray fluorescence (XRF), atomic apsorption spectroscopy (AAS), and clasical chemical analyses have been applied. The following heavy metals and other trace elements have been analyzed: As, B, Ba, Be, Cl, Co, Cr, Cu, F, Hg, Mn, Ni, Pb, Se, Sr, Th, U, V, W, Zn and Zr.

4 PETROGRAPHIC AND MINERALOGICAL FEATURES OF COAL

Lignite of the coal seam III of the Drmno deposit significantly varies in its macro-petrographic features in the seam profiles, in both vertical and lateral directions.

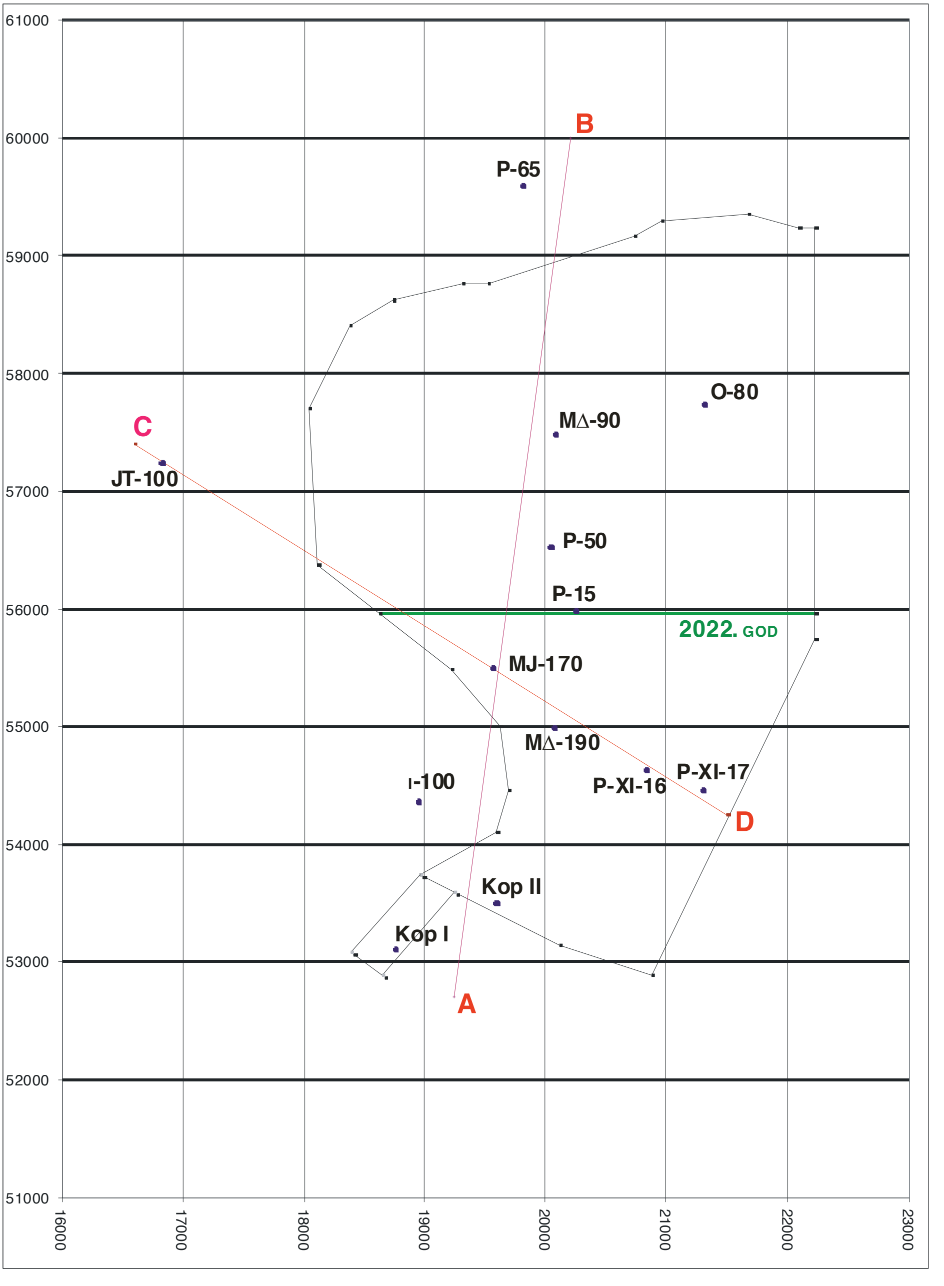

Figure 1. Map of the Drmno deposit: contours of the PK «Drmno» (poligon), with position of the mining front in 2022, locations of investigated objects and profiles accross the deposit (A-B and C-D).

Coal consists of the following coal lithotypes: pit, textite (xylithe) and doplerite coal. Among them, the pit, earthy, coal (rich in coal matter) dominates. It is followed by the xylithic coal, which appears to form interbeds, lenses or thin beds as thick as nx10 cm, which alternates with the pit coal. Doplerite (gelified plant tissue without relics of the plant structure) appears to form thin layer (as thick as to 20 cm).

The microscopic petrographic studies found thet there are also wide variations of contents of coal micro lithotypes: detrite, gelite, fuzite and resins, as well as mineral admixures of pyrite and clayey matter, in both vertical and lateral profile of the cola seam across the deposit.

Most typically coal consists of gelled xylithic ground mass which contains coal macerals (gelinite, cutinite, microspores), impregnations of clayey matter and dispersed pyrite (Fig. 2, upper).

Pyrite appears to form isolated grains (in size mostly up to 0.1 mm) and mineral agregates (Fig. 2, upper and middle).

Upper: The gelled xylithic ground mass containing coal macerals (gelinite, cutinite,microspores), impregnations of clayey matter and dispersed pyrite (bright grains).Size of picture 2x1.5 mm.

Middle: Pyrite grains in coal (datail from theprevios picture). Size of picture 0.8x0.4 mm.

Lower: The contact and transition of xylithe (dark gray with bright bands, the cell structure well expressed) into gelite mass (gray, homogenious). Size of picture 5x3.8 mm.

The process of transformation of xylithe into gelite matter is sporadically well expressed (Fig. 2, lower).

The coal petrographic composition of the Drmno deposit, with range of variations an average contents of lithotypes, is presented in Table 1.

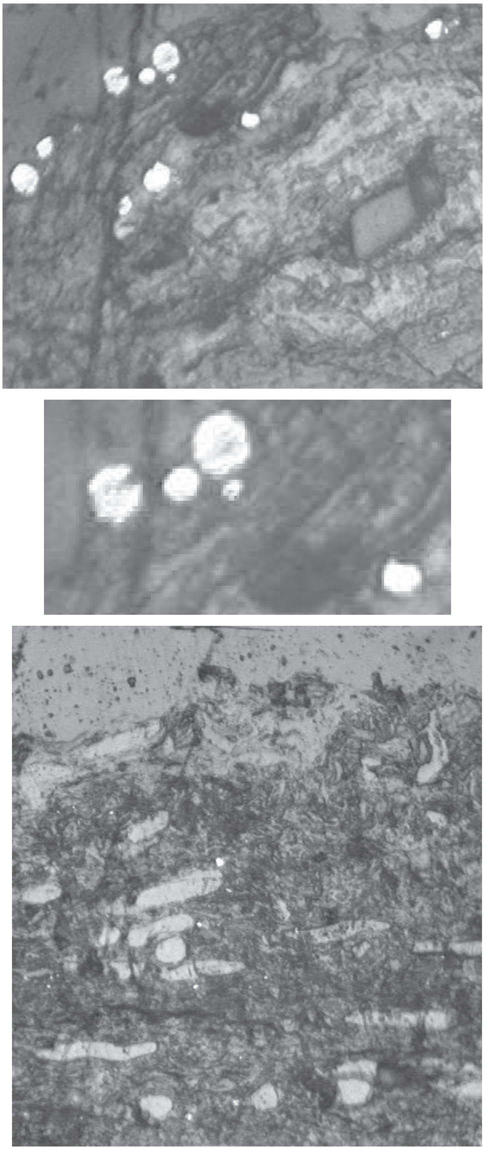

Figure 2. Microphotographs of coal.

Table 1. Petrographic composition of 39 samples of coal in the Drmno deposit.

Constituent	Range (%)	Average (%)
Detrite	11.92 - 83.00	36.91±17.53
Textite	8.98 - 82.54	37.07±17.31
Gelite	0 - 43.48	13.90±10.70
Fusite	0 - 21.26	5.51±4.86
Resin	0 - 3.50	0.29±0.86
Pyrite	0 - 8.09	1.13±2.40
Clay	0 - 45.28	4.97±8.77

The Figure 3 illustrate variations of coal petrographic composition in the vertrical profile of the coal seam.

The corellation coeficients between contents of detrite, textite, gelite and fusite in coal are presented in Table 2.

Table 2. Correlations between detrite, textite, gelite and fusite contents in coal (39 samples).

	Textite	Detrite	Gelite	Fusite
Textite	1			
Detrite	-0.620	1		
Gelite	-0.315	-0.355	1	
Fusite	-0.214	-0.128	0.202	1

The best correlation appears between textite and detrite - it is negative and as high as -0.620. Less expresed negative correlatons appear between gelite and textite and gelite and fusite.

5 GEOCHEMICAL FEATURES OF COAL

The lignite of the Drmno deposit is characterized by significant variations of contents of moister, ash and combustible meter and calorific values. Based on hundreds analyses (Table 3), average contents of moister, ash and combustible matter are 38.15 %, 17.21 % and 44.63 %, respectively. Average gross calorific value (GCV) is 11823 kJ/kg and net (or low) calorific value (NCV) is 10399 kJ/kg.

Table 3. Technical features of coal in the Drmno deposit.

Parameter	n*	Range	Mean**
Moister (%)	1217	14.39-52.60	38.15
Ash (%)	1217	0.28-53.48	17.21
Combust.(%)	815	20.16-69.27	44.63
GCV (kJ/kg)	1086	4195-19557	11823
NCV (kJ/kg)	1188	1112-18221	10399

* Number of analysis ** Arithmetic mean
GCV=gross calorific value; NCV=net calorific value.

The 15 coal samples representing coal seam profiles from diverse parts of the deposit have been selected for proximate, ultimate, chemical and trace element analyses.

In Table 4, main parameters of proximate and ultimate analyses are presented: contents of ash, combustible matter sulfur (total, organic, inorganic), carbon, hydrogen, nitrogen, and oxygen. All of them vary significantly, but their average values are typical for lignites.

Table 4. Proximate and ultimate analyses of 15 samples of coal (in wt. % of dry matter).

	Range	Average*
Ash	12.36-53.77	32.12
Combustible m.	46.23-87.64	67.88
S_{total}	1.35-2.61	1.96
$S_{organic}$	1.01-2.06	1.43
$S_{inorganic}$	0.34-0.72	0.53
C	24.23-57.48	42.02
H	3.04-5.09	4.09
N	0.53-0.96	0.71
O	15.25-25.58	20.53

* Arithmetic mean.

Sulfur (total) appears in a range of 1.35-2.61 %, in average 1.43 %. Organic-bounded sulfur is more abundant then inorganic sulfur (ratio is around 3:1).

The average contents of carbon, hydrogen, nitrogen, and oxygen are (in wt. % of dry matter): 42.02; 4.09; 0.71; and 20.53, respectively.

Chemical composition of coal (in wt. % of dry matter) is presented in Table 5. It is characteriyed by Allmost all of refered components vary significantly It is characterized by significant variations of contents of almost all components. Silica (SiO_2) is most abundant, with a range 11.11-28.78 %, in average 14.91 %. Second abundant and more variable is alumina, with a range 0.38-12.49 %, in average 6.05 %. Iron expressed as Fe_2O_3 is in a range 0.73-3.69 %, in average 1.90 %. CaO and MgO appear in contents 1.73-3.69 % (in average 2.71 %) and 0.13-1.42 % (in average 0.66 %), respectively. Contents of Na_2O and K_2O are up to 0.20 % and 0.46 %, respectively.

Table 5. Chemical composition of 15 amples of coal (wt. % of dry matter).

	Range	Average*
SiO_2	11.11 – 28.78	14.91
TiO_2	0.04 – 0.30	0.19
Al_2O_3	0.38 – 12.49	6.05
Fe_2O_3	0.73 – 3.69	1.90
MgO	0.13 – 1.42	0.66
CaO	1.73 – 3.69	2.71
Na_2O	0.01 – 0.20	0.09
K_2O	0.02 – 0.46	0.23
SO_3	3.37 – 6.52	4.88
P_2O_5	0.02 – 0.04	0.03

* Arithmetic mean

Sulfur expressed as SO_3 vary in a range 3.37-6.52 %, in average 4.88 %.

TiO_2 and P_2O_5 are in ranges 0.04-0.30 % and 0.02-0.04 %, respectively.

Abundance of heavy metals and other trace elements is presented in Table 6. Among analyzed elements, the following have been found in all samples: As, B, Ba, Cl, Co, Cr, Cu, F, Hg, Mn, Ni, Pb, Sr, V, Zn and Zr. The following elements have been in all samples bellow analythical detection limits: Be, Se, Th, U and W.

B and Cl are the most abundent trace elements: boron appears in a range 390–1130 (in average 727) mg/kg, chlorine 270–590 (in average 368) mg/kg.

Mn, Ba, Sr and Zr are rather abundant: manganese 80–430 (in average 190) mg/kg, barium 280–320 (in average 190) mg/kg, strontium 80–230 (in average 170) mg/kg, and zirconium 9–140 (in average 97) mg/kg.

Table 6. Trace elements abundance in 15 samples of coal (mg/kg of dry matter).

	Range	Average*
As	16 – 70	41
B	390 – 1130	727
Ba	280 – 320	297
Be	<1	<1
Cl	270 – 590	368
Co	2 – 6	4
Cr	50 – 60	53
Cu	10 – 20	11
F	10 – 40	26
Hg	0,11 – 0,41	0,27
Mn	80 – 430	190
Ni	10 – 40	19
Pb	3 – 32	10
Se	<1	<1
Sr	80 – 230	107
Th	<1	<1
U	<10	<10
V	20 – 50	37
W	<10	<10
Zn	10 – 30	12
Zr	9 – 140	97

* Arithmetic mean

As, Cr, and V are as abundant as up to 50-70 mg/kg: arsenic 16–70 (in average 41) mg/kg, chromium 50–60 (in average 53) mg/kg, and vanadium 20–50 (in average 37) mg/kg.

Ni, Zn, Pb, Cu, and F are as abundant as up to 20-40 mg/kg: nickel 10–40 (in average 19) mg/kg, zinc 10–30 (in average 12) mg/kg, lead 3–32 (in average 10) mg/kg, copper 10–20 (in average 11) mg/kg), fluorine 10–40 (in average 26) mg/kg.

Cobalt (Co) appears in contents 2-6 (in average 4) mg/kg.

Mercury (Hg) is as low abundant as 0,11-0.41 (in average 0.27) mg/kg.

6 GEOCHEMICAL AND MINERALOGICAL FEATURES OF COAL MINERAL (INORGANIC) MATTER

Based on macroscopic, microscopic and geochemical studies, the following three types of mineral matter or coal matter admixtures in the coal seam are determined:

(1) Clayey admixtures

(2) Marly-sandy admixtures
(3) Epigenetic mineralizations

The clayey admixtures consist of clay with variable contents of thin particles of coal matter. By XRD analyses, clay minerals of kaolinite and smectite types have been identified.

The marly-sandy coal admixtures are represented by two samples of marly and three of sandy (clastic) materials.

The marly materials are determined as slightly compacted marlstones. They consists of 45-54 % pelitic, 34-46 % carbonate and 9-12 % organic, i.e. coal matter (Table 7).

The clastic materials consists of 12-30 % sandy, 42-74 % aleuritic and 15-29 % clayey fractions, have average grain diameters 0.027-0.039 mm and are mostly poorly sorted (Table 8). They are determined as sandy-clayey and clayey-sandy aleurite.

Table 7. Contents and composition of carbonates in clastite and marly admixtures in coal (in wt. %).

Material	Pelitic	Carbonate	Org.matt
1 marl	54.0	34.0	12.0
2 marl	45.0	46.0	9.0
3 clastite	2.76	0.79	3.03
4 clastite	1.80	0.40	1.85
5 clastite	0.55	0.38	0.87

Table 8. Granulometric parameters of clastic coal inorganic admixtures.

	Sand %	Aleu-rite, %	Clay %	A.g.d mm	Sorted
1	16.9	57.6	25.5	0.030	poorly
2	11.83	73.63	14.53	0.027	moderate
3	29.50	41.94	28.56	0.039	poorly

A.g.d.= average grain diameter.

The carbonate in marls and aleurites is calcite (Table 9).

The fraction <0.063 mm of aleurites consists of: 3-50 % coal matter, 6-15 % rock fragments, 32-72 % quartz, 4-8 % micas, 2-7 % carbonates and traces of heavy minerals (Table 10).

Table 9. Contents and composition of carbonates in marly and clastite admixtures in lignite (in wt. %).

material	CaO	MgO	CO_2	$CaCO_3$
1 marl	18.04	0.84	15.08	33.96
2 marl	25.30	0.60	20.50	46.40
3 clastite	2.76	0.79	3.03	6.58
4 clastite	1.80	0.40	1.85	2.69
5 clastite	0.55	0.38	0.87	1.82

Table 10. Mineralogical composition of fraction <0.063 mm of aleurites (in %).

Sample	1	2	3
Coal matter	50	10	3
Rock fragments	6	15	12
Quartz	32	61	72
Feldspars	4	8	8
Micas	1	3	3
Carbonates	7	3	2
Heavy minerals	trace	trace	trace

There are the following three types of epigenetic mineralizations in the coal seam:
(1) Calcite crystal/aggregates
(2) Gypsum crystal/aggregates
(3) Silicification

Calcite appears sporadically and form thin individual crystals, as large as nx0,1-nx1 mm, and thin granular crystal aggregates, yellowish and brown-yellowish in color, on coal surface in fissures and cavities.

Gypsum is less abundant than calcite. It appears in the form of thin individual crystals and thin crystal aggregates on coal surface in fissures and cavities, mostly in the pit coal litho-type. The gypsum crystals are hear-like or fibrous, as long as nx0.1-nx1 mm, white or partly translucent.

Silicification of coal appears very rarely and is represented by thin veinlets (mm in size) of opal.

7 DISCUSSION AND CONCLUSSION

The usage of lignite for production elecric energy in EPS comrpises in general management of the following three areas:
(1) massive coal mining,
(2) coal firing in EPS

(3) handling of gaseous and solid wastes of ignite firing in EPS and protection of the environment.

The complex geochemical, petrological and mineralogical studies of lignite in the Drmno deposit found that both organic and inorganic matter of the coal seam are of complex and of variable mineralogical and geochemical features.

The petrographic composition of organic matter as well as content of inorganic admixtures in coal (i.e. ash) very in the vertical and lateral profiles of the coal seam across the deposit. Variations in petrographic composition and in contents of moister, comustible matter and ash in vertical profiles of the coal seam across the deposit are illustrated in Figure 3.

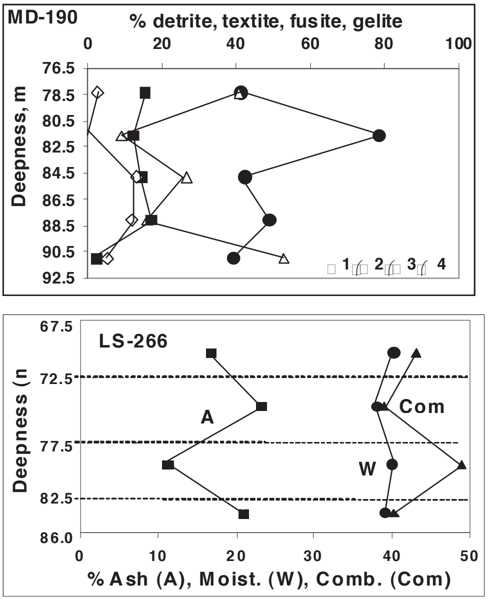

Figure 3. Diagrams showing variations of petrographic composition and technical features of lignite in vertical profiles of coal seam. *Upper:* 1-detrite, 2-textite, 3-fusite, 4-gelite. *Lower:* Moist=moister, Com= combustible.

The mineralogical and chemical composition of inorganic admixtures in coal seam (i.e. ash) also vary significantly in the vertical and lateral profiles of the coal seam across the deposit. Variations of some major elements in lateral profiles accross the deposit is illustrated in Figure 4.

Our study found that coal also contains numerous trace elements, which almost appear in rather variable contents accross the deposit (Fig. 4). Some of these elements (As, Cl, F, Hg, Ni, Pb, etc.) appear to be dangerous to the environment as enter in gaseous and/or solid vaste products of coal firing in EPS.

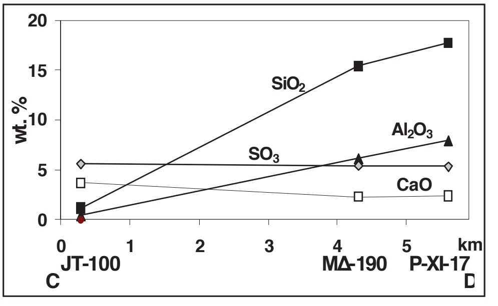

Figure 4. Diagrams showing variations of some major chemical elements in lignite in lateral profiles of the coal seam across the deposit.

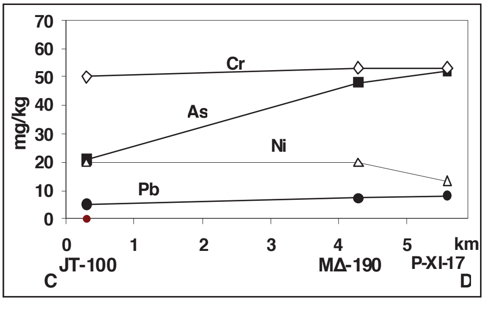

Figure 5. Diagrams showing variations of arsenic and some heavy metals in lignite in lateral profiles of the coal seam across the deposit.

The calorific value of coal is one of most important indicators of coal value for its usage in EPS. It primarily depend on content/ratio of combustible matter and ash in coal but also may depend on petrographic composition of organic matter and in some extent on mineralogical and chemical composition of inorganic matter in coal.

Variations of appeariance and contents of diverse lithotypes and microlithotypes of organic (coal) matter as well as mineral and chemical composition of inorganic matter (clayey, sandy and marly components, ratio pyrite/gypsum) may significantly impact the coal firing process in EPS. Due to these as well as variations in heavy metal and other trace element contents in coal, the impacts of coal firing in EPS to the environment may significantly vary.

Lignite of the Kostolac basin is no dubt a valuable energetic row material which will be massive used in next tens of years for firing in EPS. Coal firirng in EPS produces electric power but also waste and gasous waste products dangerous to the environment.

Our study found that lignite in the deposit is characterized by several variable geochemical features wich appear to be important to its usage in EPS. Accordingly, to a more rational mining of the deposit, it is necessary to make rather detail geochemical investigations of coal in the deposit. Based on that, it will be possibile to govern quality of the coal mined and transported into EPS, which is important for prediction of both firing process and protection of the envitonment.

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Valorisation of Raw Material Resources of the Western Part of Kostolac Mining Basin

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ABSTRACT Kostolac Mining Basin is a significant energy resource and an integral part of Electric Power Industry of Serbia represents its support for the achievement of current and future energy plans. The entire electricity generation obtained from Kostolac Mining Basin is founded on coal mining from its eastern part.

Considering the assumption that investigation results may contribute to further strategic decisions of Electric Power Industry of Serbia, i.e. CE TPPs & OCMs Kostolac, as the strategic decisions of the Republic of Serbia, this part of Kostolac Mining Basin is undoubtedly deserves a wider geological-economic analysis. This paper contains the following: (i) Preliminary analysis of raw material resources of the western part of Kostolac Mining Basin from the aspect of coal balance reserves and accompanying raw materials, with observation of already constructed infrastructural network in this area and (ii) Social – economic feasibility analysis and valorisation of these raw material resources.

1 INTRODUCTION

Kostolac Mining Basin is located in Eastern Serbia on 90 km south-east from Belgrade, covering the middle part of River Danube region. The basin is limited from the west by River Velika Morava, from the north by River Danube, while the eastern boundary is made of coal series thin-out border. The southern boundary of this basin is the town of Pozarevac. The position of Kostolac Mining Basin is shown on Figure 1.

Kostolac Mining Basin is divided into three parts: eastern, central and western. Active open cast mine 'Drmno' is situated in the eastern part, covering the area of about 50 km^2. In the central part there are two open cast mines 'Klenovnik' and 'Cirikovac', with the area of 10 km^2. Both of these mines are gradually decommissioned, i.e. OCM 'Klenovnik' is closed down, while OCM 'Cirikovac' is prepared for conservation. Western part of the basin, with the area of about 50 km^2, stretching from the current mines Klenovnik and Cirikovac to River Velika Morava represents a perspective coal mining area. Compared to the entire Kostolac Mining Basin, the level of geological exploration of this area is at this moment the lowest. Spatial division of Kostolac Mining Basin is shown on Figure2.

Figure 1. Location of Kostolac mining basin.

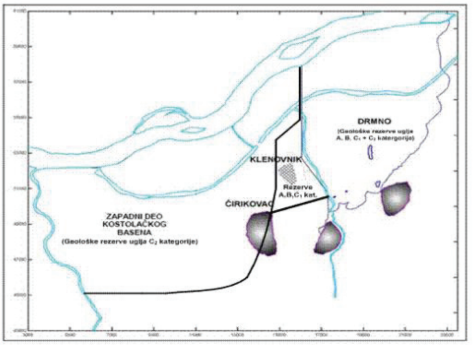

Figure 2. Layout of Kostolac mining basin with the exploration level.

Kostolac Mining Basin is characterized by the complex productive coal series of Pontian age. Within this series there are three coal seams of complex geological structure. Spatial position, i.e. extension of these three seams is uneven; however in general all three coal seams cover the basin area. The third coal seam covers the largest area, stretching over the entire basin. The second coal seam stretches above the third one, covering the smaller basin area, from its thin-out zone at OCM Drmno in the eastern part of the basin to the final western basin boundaries.

The first coal seam covers the smallest area, stretching over the central and western part of the basin. Geological tower of Kostolac Mining Basin is shown on Figure 3.

2 COAL TYPE, RESERVES AND QUALITY

Coal in the basin is represented by lignite of strip-like texture with expressed stratification. It is of light dark to deep dark colour and opaque to expressed shine. Analysis of the 3rd coal seam as the most productive seam in the basin indicated that its composition is variable, both in lateral and in vertical respect, whereby xylite litho-type, more specifically structural xylite (Zivotic, 2001) is more frequent than marsh litho-type.

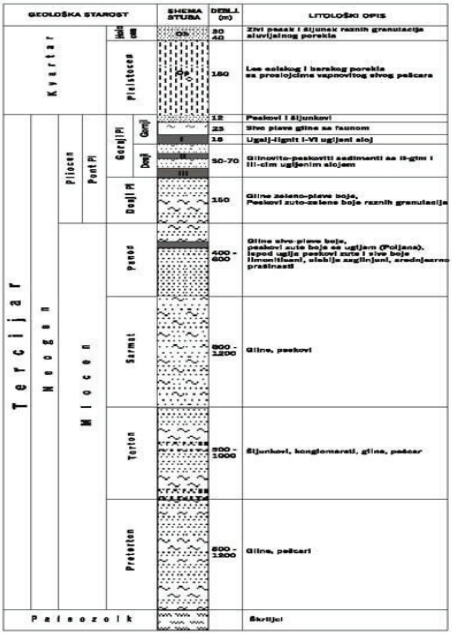

Figure 3. Geology of the Kostolac mining basin.

Structural-texture characteristics of coal seams from the eastern and the western part of the basin are not the same and they differ primarily in geological structure, inter-seam stratification with uneven thickness of coal seams and dirt bands. In the eastern part of the basin, this seam is compact, however, towards the west, slightly before the western boundary with Cirikovac it becomes stratified, whereas the thickness of clay-sand sediments inside the layers, as well as between layers increases towards west and north-west.

Exploration level of coal reserves within the basin is uneven. Based on existing categorisation of geological coal reserves and quality parameters it may be concluded that the exploration level of the eastern part of the basin is by far higher compared to the western part. Based on certified Studies on Coal Reserves in the western part of the basin, geological coal reserves of category $A+B+C_1$ amount to 800 million tons. In the central part of the basin, geological coal reserves of category $A+B+C_1$ amount to about 120 million tons, with potential reserves of C_2 category of about 120 million tons, while in the western part of the basin geological reserves are mainly of C_2 category amounting to about 1.3 billion tons. Overview of exploration sequence of Kostolac Mining Basin is shown on Figure 2. Both for Republic of Serbia and Electric Power Industry of Serbia, these geological coal reserves represent a significant energy potential. In this respect, Kostolac Mining Basin is also an essential coal resource for sustainable development of the Republic of Serbia.

Coal quality in the Kostolac Mining Basin is characterised by the following basic parameters: humidity content, ranging between 38% and 45%, lower coal calorific value is slightly higher in the eastern compared to the western part of the basin, ranging between 8.700 kJ/kg and 10.100 kJ/kg. Ash content in coal varies between 17% and 21.5%, while total sulphur content ranges up to 1.2%.

3 TECHNICAL-ECONOMIC ANALYSIS OF POTENTIALITY OF THE WESTERN PART OF KOSTOLAC MINING BASIN

This analysis is based on technical conditions for contour identification of the western part of Kostolac Mining Basin in which open cast mining may be founded as the basic assumption for economic analysis execution. It should be noted that this technical-economic analysis is carried out on insufficiently explored part of the basin, containing geological reserves of C_1 category. These reserves served for contour identification of potential operating coal reserves. In this respect, all analysis elements have been observed, starting from potential commercial coal reserves, ratio of potential overburden and coal, as the basic element of open cast mine formation, population level of the mining area, impact of infrastructure and water structures to the future coal mining, as well as energy needs of the Republic of Serbia in the forthcoming period.

Potentiality analysis of the western part of Kostolac Mining Basin from the aspect of workability was carried out on the basis of calculated potential coal reserves, shown on Figure 2. Potential commercial coal reserves within the contoured area amount to 466.4 million tons and they are related to 1st and 2nd coal seam, which has demonstrated acceptable technical elements for open cast mine opening. Basic technical parameters of the future open cast mine, stripping ratio – K_o have been obtained on the basis of the following parameters:

- Overburden thickness in the block
- Thickness of bed overburden in the block, for
  - Bed overburden (total) in 1st coal seam
  - Bed overburden between 1st and 2nd coal seam and
  - Bed overburden in 2nd seam
- Total thickness of all coal bands in 1st and 2nd coal seam, for each block.

The third coal seam is also completely included in the consideration of workability of various parts of this deposit, due to its dipping depth and high overburden amount.

Analysis of the above parameters (after calculation of the current stripping ratio) with an overview of relevant amounts provided an entirely acceptable ratio $K_o = 3.3$ t/m^3 and it corresponds to realistic parameters of profitable open cast lignite mines in Europe currently in operation.

However, despite such findings related to potential commercial reserves for coal consideration within the western part of Kostolac Mining Basin, additional geological investigations of the basin are necessary in order to obtain an adequate foundation for valorisation consideration of this significant mineral resource.

Major limiting factor in technical-economic mining feasibility analysis of this part of the basin is surely higher population level of this potential area. The following settlements are located directly above:

- Ostrovo
- Dubravica and
- Batovac

In addition to this, there are some significant water structures, such as Danube backwater serving today for cooling water supply of TPP Kostolac A whose relocation is inevitable. The existing water structures and settlements are shown on Figure 4.

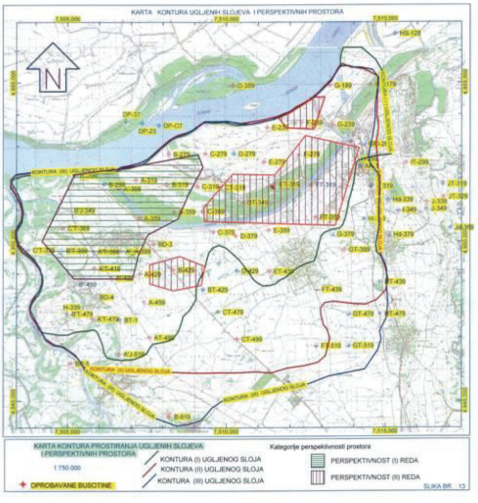

Figure 4. Existing water structures and settlements.

4 EVALUATION OF INFRASTRUCTURE FACILITIES, HOUSEHOLD FACILITIES AND OTHER FACILITIES

For the transfer of off-balance coal reserves into balance reserves, i.e. in order to enable coal mining in the western part of Kostolac Mining Basin, necessary funds need to be evaluated, serving for relocation of infrastructure facilities, compensations paid to households for civil structures and land located above coal reserves, as well as for other facilities and plantations.

Settlements located in the western part of Kostolac Mining Basin are Batrovac, Dubravica and Ostrovo. These three settlements have the population of 2952 inhabitants according to 2002 census, with the total number of households of 789, i.e. 3.7 inhabitants per household on average. Total estimated value of household structures for these three settlements is EUR 47 340 000, while the value of other citizen facilities is EUR 670 000.

Agricultural funds of households according to 1991 census include arable land of 2816 ha, forests 762 ha, barren land 988 ha and other land 230 ha, making the total area of 4796 ha. Estimated value of all land categories which should undergo expropriation is EUR 37 209 300.

Estimated value of plantations (orchards and vineyards) is EUR 10 000 000.

Total estimated real estate value in settlements was established with the increase of 20% compared to estimated values of household and other facilities and it amounts to EUR 9 468 000. This was done due to possible changes in construction level of settlements and legal obligation stipulating that compensation for expropriated immovables may not be lower than the market value at the moment of contract signing with the citizens.

Considerable funding is also necessary for relocation of Danube backwater and cooling water of TPP Kostolac A, supplied from this backwater. However the level of funding may not be anticipated at this moment.

There are other infrastructure facilities such as roads, transmission lines, railroad located on the coal deposit in the western part of Kostolac Mining Basin which will also have to be relocated. Estimated funds for this purpose amount to about EUR 10 000 000.

Total estimated funds, which will have to be provided, based on the open cast mine expansion schedule, amount to EUR 114 687 300 (without Danube backwater relocation).

5 SOCIAL – ECONOMIC FEASIBILITY ANALYSIS AND VALORISATION OF THIS RAW MATERIAL RESOURCE

Commercial reserves from the western part of Kostolac Mining Basin amounting to 466.4 million tons and their valorisation (coal supply to thermal power plants under the current price of 10 EUR/t) would provide feasibility of investments from the social – economic aspect.

In addition to this, in the western part of Kostolac Mining Basin, there are some other accompanying raw materials apart from coal as the most important resource, such as high gravel reserves of about 300 million m^3. Consequently, high benefits could be achieved through market placement of this raw material.

Based on previous experience from open cast coal mining at Electric Power Industry of Serbia, this expropriation scope of settlements should not affect the profitability of an open cast mine with the stripping ratio of $K_o = 3.33$.

Taking into consideration the outdated cadastre information, for this type of settlements, especially at this moment, considerations of all factors and indicators having an impact on social – economic feasibility analysis may not be established with a higher reliability level.

Provided there is interest in more detailed analysis of this part of deposit, expropriation issue will also be adequately considered. Therefore, it should be noted that objective assessment of value (especially households) requires a more extensive field work, primarily surveying of all immovables in settlements and polling of population providing a more realistic assessment of expropriation costs.

Based on experience in Electric Power Industry of Serbia in consideration and valorisation of raw material resources through land acquisition and relocation of settlements and infrastructure facilities, the same principle of valorisation, land acquisition may be applied in the western part of Kostolac Mining Basin. One of the ways to resolve this problem is provision of new resettlement locations and thereby balancing of considerable commercial coal reserves of 466.4 million tons.

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Reserve Estimation of Dereköy Copper Deposit Considering the Feasibility of the Project

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ABSTRACT Mining is a very risky business because it includes some uncertainties related with reserve estimation. Accurate reserve and grade estimation is the main basis of successful and profitable mining. Long, medium and short term mining planning are determined by reserve estimation of the mineral deposit. Today, different types of software are used for reserve estimation. The results obtained by a computer application are more accurate when compared to classical estimation methods. In this study, Dereköy copper deposit is evaluated by Micromine, mining software. At first, all deposit is evaluated and solid model and grade distribution of the deposit is estimated. Then, it is investigated that mining of all deposit is not feasible with today's market conditions. The southeastern part of the deposit is determined to evaluate independently with considering the grade distribution. To compare with the results of Micromine software, the southeastern part of the deposit is also evaluated by polygonal method.

1 INTRODUCTION

In this paper, Dereköy copper deposit is evaluated and the economical value of the deposit is discussed. This deposit is located near Dereköy town which is in Kırklareli province in Turkey and it is close to the boundary of Bulgaria.

Dereköy copper deposit is a typical porphyry type copper deposit and it is established in Istranca massive. The Istranca massive which is a part of Sredno-Gora zone, starts from Romania and passes over Yugoslavia and Bulgaria. This zone is important especially for porphyry type mineralization. Therefore, there are many porphyry, skarn, and vein type mineralization. Porphyry type copper reserve has two meanings as economically and geologically. Economically, porphyry type copper reserves has average copper grade of 0.8% and geologically reserve amount of more than 500 million tons containing small amount molybdenum, gold, and silver. They can be mined with open and underground mining methods (Erdem, 2008).

Doğan (1987) states that some copper presence clues had been observed and General Directorate of Mineral Research and Exploration (MTA) decided to make some exploration drilling at Dereköy location.

1.1 Exploration Drillings at the Location

The exploration was started at 1981 between Dereköy and Karadere with one drilling to investigate an ore reserve. The exploration was continued with one drilling at 1982, two drillings at 1983, 13 drillings at 1984, and eight drillings at 1985. Up to 1986, totally 25 exploration drillings was conducted with the total drilling length of 8,776 m (Doğan, 1987). Locations of the exploration drillings are presented in Figure 1.

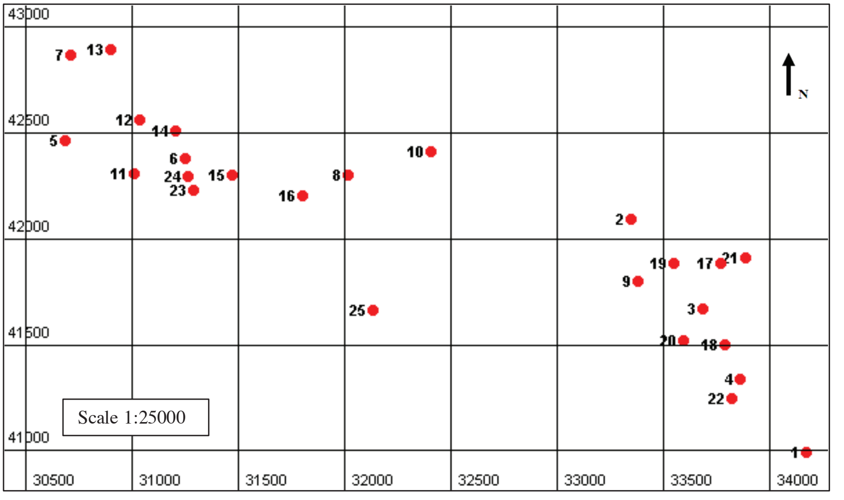

Figure 1. Locations of the drillholes.

2 RESERVE EVALUATION OF THE DEPOSIT

Reserve evaluation of the deposit, in general means the estimation of the average grade and tonnage of the deposit.

2.1 Average Grade of the Deposit

Metal content of an orebody is determined by using the average grade of the orebody. Therefore, determining the average grade of the orebody is very important step in reserve estimation. In this study two methods were applied to determine the average grade. In the first method, the grade distribution of the deposit is established using the data obtained from 25 drillholes. In the second method, the average grade was estimated by inverse distance method. Then, the results obtained from the calculation and the estimation was compared.

Every ten meters of the each drillhole were analyzed and grade of them were projected. First to conduct statistical analysis, grade histogram and frequency distribution are drawn using the data obtained from 25 drillholes as illustrated in Figure 2. From frequency distribution, it is seen that the grade has lognormal distribution. The average copper grade as shown on the figure is about 0.14 % Cu.

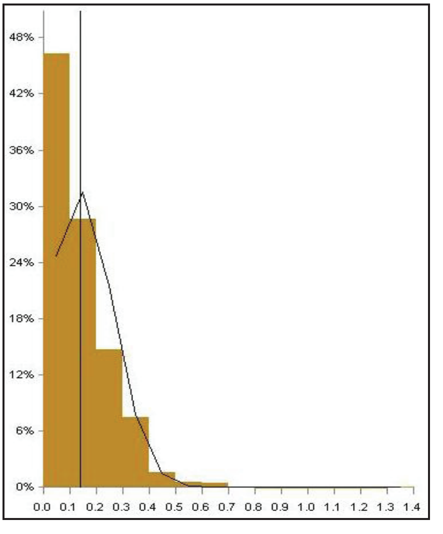

Figure 2. Grade histogram of the deposit.

In developing a mining project, an orebody model is generated from drillhole data to represent the deposit, usually through the application of geostatistical techniques (Ramazan *et al.*, 2005) such as kriging, inverse distance weighting and nearest neighbor method. Inverse distance weighting method was applied for modeling of Dereköy copper deposit as a second method. 3-D solid model of the orebody is generated and it is divided into blocks to apply inverse distance weighting method and the grade distribution of the deposit. The other name of this procedure is block model method. At this method, the deposit is divided into blocks and grade of each block is estimated then the average grade of the deposit is estimated by using the grades of the blocks. This procedure can only be done by using some mining software. In this paper, Micromine which is one of the mine design software is used. The deposit is divided to 40x40x20 m dimensions blocks and 4x4x4 m dimensions sub-blocks. The total number of blocks and sub-blocks together is 1,833,120. After the application of the block model method, average grade of the deposit is estimated as 0.106% Cu. One of the great advantages of the block model is that we can observe the grade distribution of the deposit. Therefore, we can see where the grade is high or low. Grade distribution of the deposit is indicated in Figure 3. After block model application, detailed analysis of the deposit is also estimated and the results are given in Table 1. Block model method also indicates the grade histogram of the blocks as seen in Figure 4.

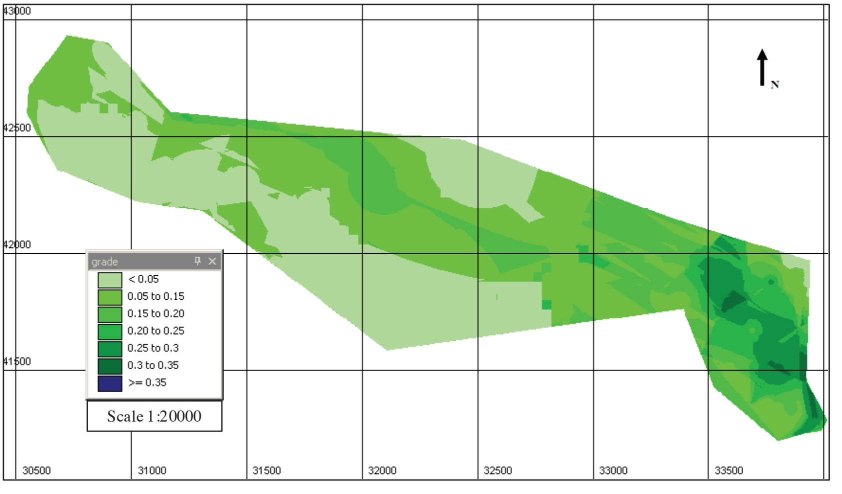

Figure 3. Grade distribution of the deposit.

Table 1. Grade distribution analysis of the deposit.

Grade From	Grade To	Volume, m^3	Tones, ton	Grade, %	Cumulative Volume. m^3	Cumulative Tones, ton	Cumulative Average Grade, %
0	0.05	152,305,536	411,224,947	0.031	152.305.536	411,224,947	0.031
0.05	0.15	344,626,688	930,492,058	0.091	496.932.224	1,341,717,005	0.073
0.15	0.20	71,947,008	194,256,922	0.173	568.879.232	1,535,973,926	0.085
0.20	0.25	39,109,696	105,596,179	0.221	607.988.928	1,641,570,106	0.094
0.25	0.30	22,900,992	61,832,678	0.275	630.889.920	1,703,402,784	0.101
0.30	0.35	10,784,128	29,117,146	0.318	641.674.048	1,732,519,930	0.104
0.35	100	3,068,736	8,285,587	0.397	644.742.784	1,740,805,517	0.106

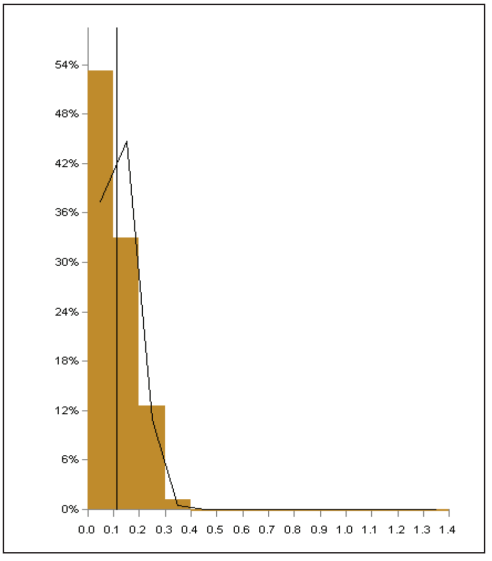

Figure 4. Histogram of the grades estimated by block model method.

Exploration drillings were conducted to investigate an orebody. Average grade of the orebody was determined by the grade data obtained from drillholes. During the exploration stage optimum distance was not considered. Therefore, the grade data may not represent all orebody. If the optimum distance is selected, the difference between mean of histogram constructed by the grade data from drillholes and estimated average grade with block model method is low.

The mean of histogram is 0.14% Cu but the estimated average grade is 0.106% Cu. There is a difference between them. The main reason of this difference is the much distance between the drillholes. This difference also indicates that more than 25 drillholes are required for this location for more accurate results. The estimated result by block model is considered for the reserve amount estimation because of this reason.

2.2 Reserve Amount of the Deposit

Reserve amount estimation is also done for the deposit by block model method as seen in Table 1. It is estimated as 1,740,805,517 ton.

As seen in Figure 4 and in Table 1, 77.1% of the reserve has a grade lower than 0.15% Cu. Figure 3 also indicates that the northwestern part of the reserve has low grade and there are Dereköy town and some international roads are located on the northwestern part of the deposit. Because of these reasons, the feasibility of the southeastern part of the reserve is evaluated at first.

3 RESERVE EVALUATION OF THE SOUTHEASTERN PART OF THE DEPOSIT

As mentioned at section 2.2, the southeastern part of the deposit is more suitable than the whole deposit. Therefore, the part is evaluated in detail. Besides the mine software usage, Micromine, for reserve evaluation, polygonal method is also applied to check the outputs of the computer software.

There are 11 drillholes, namely number 1, 2, 3, 4, 9, 17, 18, 19, 20, 21, and 22 which are located in the southeastern part of the mineralized zone. These drillholes are used for modeling and statistical analysis of the grade.

3.1 Reserve Evaluation by the Software

At the first step, solid model of the southeastern part of the deposit is created at the software environment and then the part is modeled in detail to get more accurate grade estimation. During block model application, dimensions of blocks are reduced to 10x 10x5 m and dimensions of sub-blocks are reduced to 2x2x2.5 m. 144,936 blocks and 522,352 sub-blocks are created by block modeling. Inverse distance weight method is applied again to estimate the grade of the each block and sub-block.

Average grade of the deposit is found by block model method as 0.244% Cu. Detail analysis of the southeastern part is shown in Figure 5, Figure 6 and Table 2. Table 2 indicates that only 11.9% of the southeastern part is lower than 0.15% Cu.

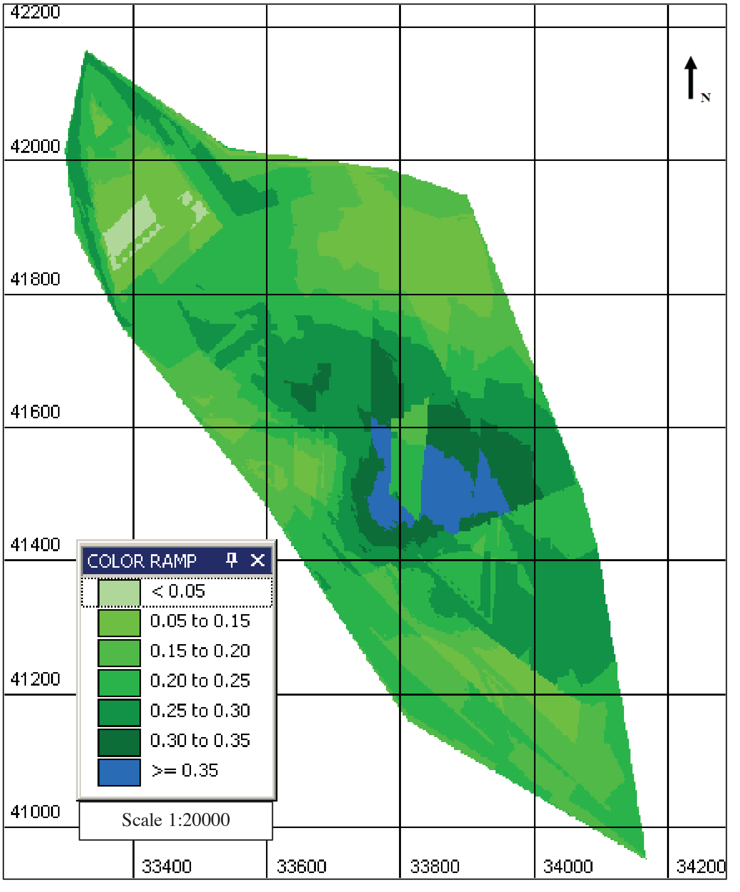

Figure 5. Grade distribution of the southeastern part of the deposit.

Table 2. Grade distribution analysis of the southeastern part.

Grade From	Grade To	Volume, m^3	Tones, ton	Average Grade, %	Cumulative Volume, m^3	Cumulative Tones, ton	Cum. Average Grade, %
0	0.15	8,784,450	23,718,015	0.124	8,784,450	23,718,015	0.124
0.15	0.20	14,519,640	39,203,028	0.178	23,304,090	62,921,043	0.158
0.20	0.25	20,539,540	55,456,758	0.226	43,843,630	118,377,801	0.190
0.25	0.30	17,305,750	46,725,525	0.271	61,149,380	165,103,326	0.213
0.30	0.35	9,303,630	25,119,801	0.324	70,453,010	190,223,127	0.228
0.35	100	7,238,510	19,543,977	0.406	77,691,520	209,767,104	0.244

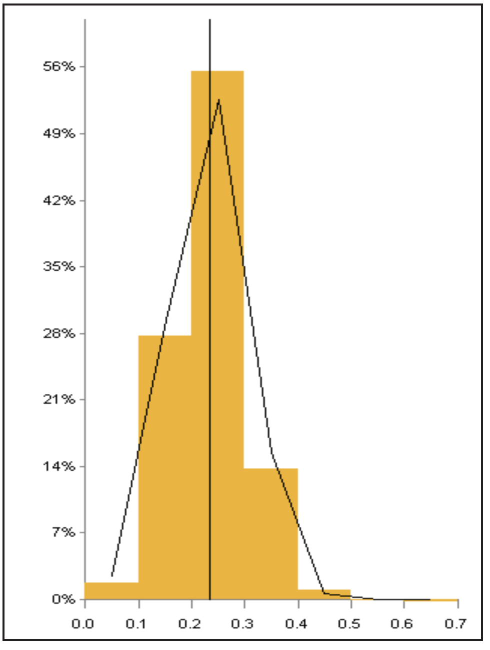

Figure 6. Histogram of the southeastern part's grades estimated by block model method.

Ore amount is also estimated by the software. It is found as 209,767,104 ton. The average density of the orebody is taken as 2.7 ton/m^3 by Doğan (1987).

3.2 Reserve Evaluation by Polygonal Method

Polygonal method was applied to the southeastern part of the deposit to estimate the amount of ore and average grade to check the outputs of Micromine.

The polygonal method of estimating ore amount and average grade is based on the area of polygons constructed around each drillhole. Many geometrical considerations are necessary when using the polygonal method of estimating ore reserves. An irregular boundary line is one of them. The main geometrical problem involved in the polygonal method of computing ore reserves is to make the sides of the polygons coincident with the boundary lines of the deposit. In this study, the boundary line is constructed by considering the position of the polygons and the influence area of drillholes. The area of influence of each drillhole and the boundary line is presented in Figure 7. In this study, a digital planimeter was used to find the polygon areas. To reduce the measurement errors, each polygon's area was measured three times and their average was accepted as the polygon area.

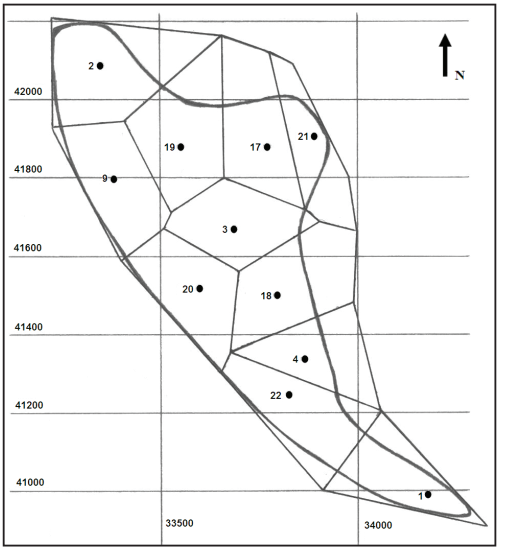

Figure 7. Area of influence of drillholes and the boundary line.

As indicated in Table 3, amount of orebody and the grade are 212,105,421 tons and 0.255% Cu respectively by polygonal method. When average grade of the deposit is estimated with polygonal method, weighed average method is applied with considering reserve amount of each polygon.

Table 3. Grade estimation obtained by polygonal method.

Polygon No	Ore Amount, ton	Average Grade, %	Cu Content, ton
1	5,103,063	0.229	11,686
2	23,880,582	0.199	47,522
3	25,201,296	0.372	93,749
4	24,279,948	0.232	56,329
9	10,286,955	0.222	22,837
17	22,394,736	0.108	24,186
18	33,605,226	0.325	109,217
19	39,600,630	0.283	112,070
20	9,285,633	0.236	21,914
21	3,953,664	0.158	6,247
22	14,513,688	0.240	34,833
TOTAL	**212,105,421**		**540,591**
AVERAGE		**0.255**	

3.3 Comparison of the Applied Methods Outputs

The outputs of the applied methods are compared to check the results of the computer software. The estimated amount of tonnage is about 209,767,104 tons by

Micromine but 212,105,421 tons ore amount is estimated by polygonal method. There is 2,338,317 tons difference between them. Estimation of average grade of the deposit is almost the same. Micromine estimates average grade as 0.244% Cu. The copper percent estimated by polygonal method is 0.255% Cu. Results are shown in Table 4.

Table 4. Outputs of Micromine and polygonal method.

	Reserve Amount, ton	Grade, %Cu
Micromine	209,767,104	0.244
Polygonal Method	212,105,421	0.255

Although there is a slight difference between Micromine and polygonal method, Micromine result is more accurate than polygonal method because polygonal method is more likely to include an error due to the fact that the boundary of the deposit was defined manually and the measuring error made while using planimeter. Results of Micromine are validated by polygonal method results. The slight difference indicated that the results are consistent.

4 CONCLUSIONS

Dereköy copper deposit is explored with drillholes and amount of deposit is estimated with Micromine and polygonal method. Dereköy copper deposit can be considered as a visible reserve in terms of traditional reserve classification.

The application of Micromine software gives us the chance to evaluate complete deposit or part of it. The grade and tonnage of complete deposit are found as 0.106% Cu and 1,740,805,517 ton respectively.

The grade and the tonnage of the richer part of reserve as mentioned in the previous are 0.244% Cu and 209,767,104 ton respectively.

Number of drillhole is not enough when compared largeness of the area of the studied location. Therefore, number of the drillholes may be increased to reach more precise results.

This copper deposit is not feasible with the today’s market conditions. Copper price was about 8,300 $/ton in the midst-2008 (Erdem, 2008). However, the current copper price is about 4,890 $/ton (London Metal Exchange, 2009). Dereköy copper reserve will be a profitable reserve when the copper price rise up to about 7,500 $/ton.

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New Product Development via Application of Ceramic Glaze and Decorative Techniques to Andesites of Afyon-Iscehisar

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ABSTRACT This report presents the initial results of the development of a new product using andesites from the Afyonkarahisar-İscehisar (Karakaya) region as an alternative to traditional ceramic wall and floor tiles used in the construction sector. A series of characterization tests (XRD, XRF, SEM, Dilatometer, DTA-TG, Heat Microscope, Mineralogical-Petrographical analysis and physico-mechanical tests) were conducted on andesite samples. Glazes were then applied to the samples for trial purposes. Analysis indicated that (i) the andesite samples consisted of opal-CT, mica, pyroxene, serpentine and hematite minerals and (ii) its apparent porosity, density, water absorption by mass and compressive strength values were 23 %, 2.55 g/cm^3, 7.43 % and 40.7 Mpa, respectively. Heat microscope measurements showed that maximum sintering was recorded at 1138 ºC. Linear expansion coefficient (α) of the andesite at 400 ºC was 325.82x10^{-8} 1/K. Firing using the prepared glaze recipe at approximately 1160 ºC gave good results in terms of body-glaze harmony. In addition, different decorative surface finishes -to be used in indoor and outdoor spaces- were obtained by the use of underglaze decorative technique.

1 INTRODUCTION

With its 4000-year history of marble production, Turkey is one of the oldest natural stone producers in the world. It is still one of the countries which have the largest natural stone reserves in the world. Natural Stone, in commercial terms, refers to rocks that can be cut and removed in blocks of economically-appropriate sizes; that can be cut smoothly at the desired sizes, and; to which polishing, ageing and other surface processes can be applied if required. Natural stones are classified according to their formation, as (i) metamorphic natural stones (marble, etc.); (ii) sedimentary natural stones (limestone, travertine, onyx, etc.); (iii) magmatic natural stones (granite, syenite, gabbro, trachyte, basalt, andesite and tuff). Andesites are classified within the volcanic rocks group, which constitutes a sub-group of magmatic rocks (Sentürk, 1996; Kulaksız, 2007).

The Afyonkarahisar (İscehisar) region of Turkey -also known as the “Docimenion Region” in historic periods- has always been an important area for marble production and processing. However, uncontrolled use of clean and high-quality marble resources utilized as natural stone, a gradual decrease in the natural stone reserves and the different product demands of the market, have obliged researchers to focus on developing new products which can be used in indoor and outdoor spaces (Sariisik, et al., 2008a; 2008b).

Andesite has long been used in Turkey and many other parts of the world, particularly in civil engineering and architecture, including in the production of pavements, kerb-stones, staves, coping, windowsills, jambs, and friezes. Andesite is a fine-grained volcanic rock, varying between grey and black in color, and including silica (53-63%). It has a porphyritic texture. It is composed of plagioclase and pyroxene microliths (clinopyroxene and orthopyroxene), feldspar, pyroxene and biotite phenocrysts in a glass matrix, and small amounts of magnetite minerals. Depending on the dark color mineral components, the color of the andesite varies from light grey to grey, dark grey, black and reddish-brownish-pinky tones (Williams, et al., 1982; Koca, et al., 2001; Aydar, et al., 1998, 2003),

The porosity rate of andesites is between 10 – 25%. Therefore, andesites to be used in indoor and outdoor spaces are exposed to various physical and chemical factors, such as cold, heat, moisture and chemicals commonly used at home, and; various impact-induced wearing factors. Andesite materials should be resistant to factors to which they may be exposed within specific environments (Sariisik et al., 2008a, 2008b).

Ceramic glaze is the vitreous layer produced by melting and solidifying silicate mixtures (according to the method and at the temperature required by the related technology) on ceramic bodies. Depending on the different temperatures at which firings are performed, ceramic glazes provide the underlying material with particular characteristics, such as mechanical strength, impermeability, electrical resistivity and resistance to acids/bases (Kartal and Gürtekin, 2002; Kartal, 1998). Serigraphy is the process of directly or indirectly transferring the prepared glaze (by using auxiliary tools) to the surfaces that will be decorated, using patterns prepared on special-texture nylon or silk sieves (Sevim, 2003).

A literature review identified several previous studies on the use of andesite and similar natural stones in glaze production (Çetin, 2005; Duran et al., 2002; Ercenk et al., 2006; Gal'perina et al., 1980-1981; Dvorkin and Galushko, 1969); however, no study has previously been conducted on the application of ceramic glazing on such natural stones. Therefore, the present study aimed to develop a new product by applying ceramic glaze to the surface of volcanic andesite. The following results are based mainly on data obtained from physical, chemical and thermal characterization of the andesite samples.

2 EXPERIMENTAL STUDIES

2.1 Material

2.1.1 Chemical characteristics

Andesite samples used in the present study were obtained from the quarry of İyigün Andezit Company, operating in Afyonkarahisar/İscehisar. The samples used in the tests were prepared by using the blocks provided from the quarry. The surface of the andesite samples were thin-grained, reddish in color and had macro-pores. Chemical analyses of the andesite were conducted at ACME Analytic Laboratory (Canada) via XRF (ICP-ES) method and by using an inductive compound plasma-mass spectrometer. Chemical analysis results of the K1 andesite samples obtained from Afyon-İscehisar region are listed in Table 1.

Table 1. Chemical analysis of Afyon (K1) andesite.

Chemical Analysis	*Unit*	Value
SiO_2	(%)	62.30
Al_2O_3	(%)	14.70
Fe_2O_3	(%)	4.04
MgO	(%)	2.78
CaO	(%)	4.26
Na_2O	(%)	2.95
K_2O	(%)	6.06
TiO_2	(%)	0.98
P_2O_5	(%)	0.81
MnO	(%)	0.07
Cr_2O_3	(%)	0.014
Ba	ppm	1526
Ni	ppm	<20
Sr	ppm	874
Zr	ppm	524
Y	ppm	22
Nb	ppm	28
Sc	ppm	12
LOI	%	0.6
Total	%	99.92

[a] LOI: Loss of Ignition

2.1.2 Mineralogical and petrographic characteristics

Andesites are classified by total alkali silica (TAS) diagram on the basis of their SiO_2 (in the range 49%-62%) and alkali oxide (Na_2O+K_2O) content (Le Bas et al., 1986; Aydar et al., 1998, 2003). The andesite sample used in the tests included 9.01% alkali oxide (Na_2O+K_2O) and 62.30% SiO_2. Figure 1 shows the chemical compounds from basalt to rhyolite in the total alkali silica diagram. The andesite used in the tests was found to be of the trachy-andesite type.

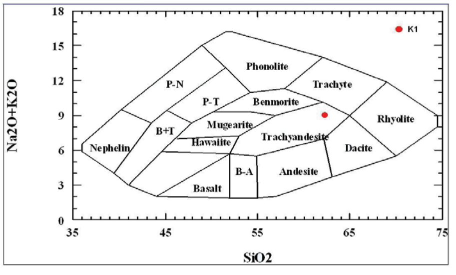

Figure 1. Total alkali-silica diagram.

Mineralogical and petrographic analyses of the K1 andesite sample were conducted at the Mineralogical-Petrographical Analysis Laboratory of DG Mineral Research and Exploration (MTA). According to these analyses, the K1 sample was defined as "Pyroxene Andesite". Mineralogical characteristics of the K1 sample were examined and, mineral composition of the sample is given in Fig.2, based on a thin cross-section.

Accordingly, K1 samples were of holocrystalline porphyritic texture. The main components of these samples were (by amount) biotite minerals, pyroxene-group minerals, plagioclase-group minerals and induced xenocrystalline mafic minerals (olivine) in phenocrystalline form; the paste was composed of the microlites of these minerals and the opaque minerals. Flow texture was observed in the plagioclase microliths of the paste. In addition, the K1sample had a porous structure, which was partially filled by zeolite minerals. Biotite phenocrysts and microliths were subhedral. Opacity formation was observed, mainly starting from their edges. A part of the microliths in the paste was completely opacified. Pleochroism formation from was recorded. While pyroxene phenocrysts and microliths were generally unhedral, trace amounts of pyroxene phenocrysts and microliths were present. Heavily cracked and fractured phenocrysts constituted the glomeroporphyric texture. Plagioclase phenocrysts and microliths were unhedral and the K1 sample was a karsbad-twinned andesite. Zoned-texture was observed in the phenocrysts, starting from the edges. Opaque minerals were recorded to at the micro-level, unhedral and heavily in scattering form.

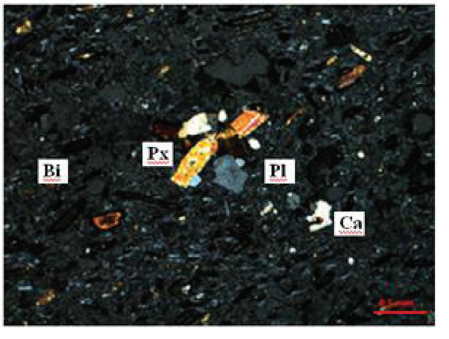

Figure 2. Thin cross-section of Afyon (İscehisar) andesite (Double Nicol). Pl: Plagioclase, Px: Pyroxene, Bi: Biotite, Ca: Calcite.

2.1.3 Physical and mechanical characteristics

The following tests were conducted at the laboratories of the Department of Mining Engineering of Afyon Kocatepe University, to determine the pre-glaze physico-mechanical characteristics of the K1 andesite sample: Volume Mass, Specific Mass, Porosity, Occupancy Ratio, Water Absorption, and Water Absorption in Boiling Water tests according to TS 699; Compressive Strength test according to TS EN 1926; and Bending Strength test according to TS EN 13161. In addition, the laboratories of the Construction Education

Department of Afyon Kocatepe University were used to conduct Ultrasonic Speed Tests according to TS EN 14579. The andesite samples used in these tests were 70.0x70.0x70.0 mm and 30.0x50.0x180.0 mm in size. Tests were conducted on a minimum of 5 samples, on average. The results of the physico-mechanical tests on the andesite samples are presented in Table 2.

Table 2. Physico-mechanical characteristics of Afyon (İscehisar) andesite.

Physico-Mechanical Characteristics	*SI Unit*	*Value*
Volume Mass	(kg/m^3)	1970
Specific Mass	(kg/m^3)	2550
Water Absorption by Volume	(%)	14.62
Water Absorption by Mass	(%)	7.43
Water Absorption by Mass in Boiling Water	(%)	11.84
Water Absorption by Volume in Boiling Water	(%)	23.08
Porosity	(%)	22.74
Occupancy Rate	(%)	77.26
Compressive Strength	(Mpa)	40.75
Bending Strength	(Mpa)	6.86
Ultrasonic Speed	m/sec	3190

Mineralogical analyses of the andesites were made by using "Rigaku Rint 2000" brand XRD device. Fractured Surface SEM (Scanning Electron Microscope) images of the raw and fired andesites were obtained via a Leo-1430 VP brand device. Samples were coated with carbon before the analyses. When the samples were within the 0-1200 ºC range, DTA-TG analysis was made by using a LINSEIS L-81 model device under normal atmospheric conditions, in an alumina crucible at 20ºC/min heating rate. A 20-30 mg sample was used in the analysis. A LINSEIS L-75/1000 model Dilatometer was used to detect the linear thermal expansion coefficient of the andesite sample. A computer-controlled dilatometer ran at a heating rate of 20 ºC/min. During this analysis, stick samples were used, each of which was 5.0x5.0x50.0 mm in size. In addition, sintering behavior was tested via an optical dilatometer and melting behavior was tested via a heat microscope. A Misura 3.32/ODHT-HSM 1600/80 model device was used in this analysis, where an optical dilatometer and heat microscope were used at 80K/min heating rate at 1400 ºC on the 5.0x5.0x15.0 mm size samples.

2.2 Preparation of Glaze Recipe

A transparent glaze-based, commercial recipe (Eczacıbaşı) was used as the basis to develop 2 types of recipes, by adding $CuCO_3$, CoO and PbO. Powders were obtained via the wet grinding method in a jet mill Jet. Approximately 100 gram of each of these powders was mixed with ~60 ml water for 5 minutes. The obtained glaze recipes were applied to the K1-Andesite sample (10.0x10.0x1.0 cm in size) via a spraying method. The glaze was fired in a REF-SAN brand, cabin-type ceramic kiln with a firing volume of 0.5 m^3 at 1160 °C. Glazes colored with different pigments were also tested. In the 1160 ºC kiln regime, the temperature was raised to 573 ºC over 3 hours; kept at 573 ºC for 30 minutes; raised to 1160 ºC over 3 hours; kept at this temperature for 15 minutes; and reduced to 100 ºC over 42 hours (Fig. 3).

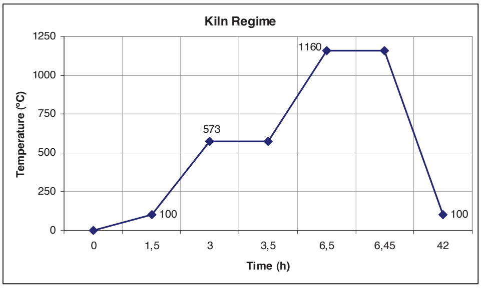

Figure 3. Kiln regime applied to the glazed andesites.

2.3 Process of Serigraphy Print Application

A paint/medium ratio of 2:1 was ground using a jet mill or manual spatula to obtain a homogenous mixture. The last part of the serigraphy process, he printing application, was performed on a serigraphy table. The sieve frame to be used for printing was mounted on the table. The andesite plate (15.0x15.0x1.0 cm), on which the print would be applied, was placed under the pattern of the sieve frame. The clearance between the sieve and andesite -which was necessary for the printing process- was checked; the paint was applied to the pattern with the help of a stripper, in such as way as to ensure that the paint was transferred to the andesite surface. This process was repeated for each print.

3. RESULTS AND DISCUSSIONS

3.1. Analyses Results

XRD analysis of the K1-Andezit sample showed the presence of Sanidine ((K-feldspar), $(K,Na)(Si_3Al)O_8$) and Mika (K(Mg,Fe), $K(Mg,Fe)_3$ $(Al,Fe,Si_3)O_{10}(OH)_2$) minerals. The XRD graph of the K1-Andezit sample is given in Fig. 4.

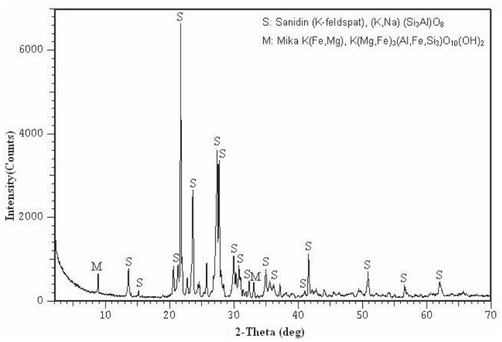

Figure 4. XRD pattern of the K1-Andezit sample.

XRD analysis of the K1 andesite was made after the sample was subjected to a 6-hour thermal treatment at 1000 and 1055 ºC (Fig. 5). Comparison of the XRD analyses of the sample which was thermally treated at 1000 and 1055 ºC with the XRD analyses of the original samples produced no significant change in terms of the mineralogical phase structure. However, examination of the $2\theta=20^o-40^o$ range of XRD analysis indicated a reduction in the crystallinity level of the sanidine mineral. This is an indicator of an increase in the amorphous phase amount. This is an indicator that feldspathic andesite has been subjected to some local melting. Taking into consideration the feldspathic structure of the andesite, it can be concluded that its physical and mechanical characteristics change when it is subjected to high temperatures. It is known that alkali feldspaths vitrify easily at high temperatures (Bernardo et al., 2006).

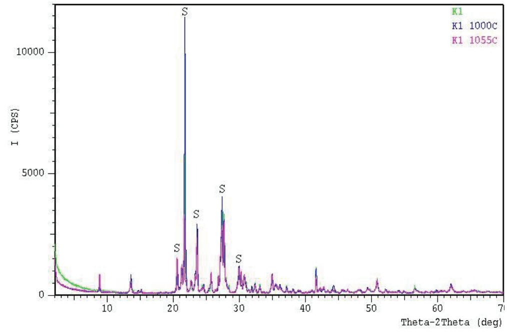

Figure 5. K1 XRD patterns of the andesite sample at 1000 and 1055 ºC.

After a 6-hour thermal treatment at 1000 and 1055 ºC, the micro structures of the K1 andesite were analyzed using SEM images (Fig.6). Fig.6 indicates there was a weak mass bond between the particles before the thermal treatment. There are clearly large pores between the particles throughout the entire structure. No vitreous phase formation was observed in the inner structure of the original andesite. The bond between the particles seems to have become slightly stronger at 1000 ºC; however, large particle edges were still surrounded by large pores, and porosity decreased to some extent when compared to the normal condition (before the thermal treatment).

In addition, a silicate melted and started to form a vitreous phase, due to the effects of the alkalines present at 1000 ºC. It was observed that, at 1055 ºC, the sintering increased in parallel with the increase in the liquid phase amount; that the pores between the particles (porosity) became smaller; and thus the resistance increased. Therefore, it is possible to conclude that the resistance increase created by the thermal treatment resulted from the vitrification caused by the thermal treatment, rather than from the newly-developed phases. Condensation effect, which resulted from the vitrification effect, can also be clearly observed from the changes recorded in the volume mass, porosity and water absorption values. The condensation effect resulted in a decrease in porosity value and a considerable reduction in the water absorption values.

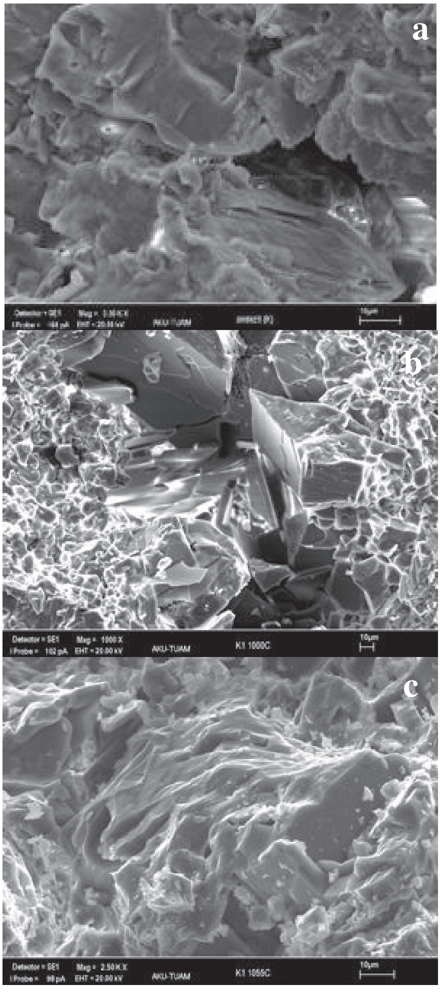

Figure 6. Fractured surface SEM photos of the K1 andesite before (a) and after thermal treatment at 1000 ºC (b), 1050 ºC (c).

The results of the DTA-TG analyses of the K1-Andesite are given in Fig. 7.

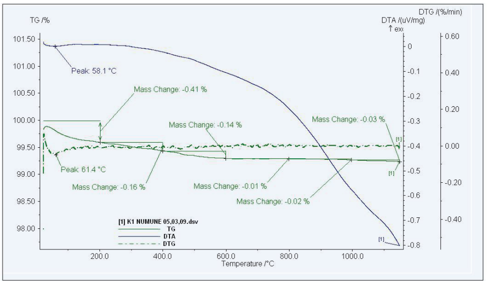

Figure 7**.** DTA-TG curve of the K1-Andesite.

It is recorded that the K1-Andesite lost approximately 0.41% of its mass in the 0-200 ºC range; 0.16% of its mass in the 200-400 ºC range; 0.14% of its mass in the 400-600 ºC range; and 0.05% of its mass in the 600-1200 ºC range. The total mass loss of the K1-Andesite was calculated to be 0.76%. It can be concluded that this mass loss resulted from the removal of moisture and impure volatile organic substances (possibly present in the structure in low amounts) due to thermal effects. LOI (loss of ignition) value, CaO, MgO and P_2O_5 amounts determined via the chemical analysis were found to be at similar levels to the mass loss recorded in the TG curve. This finding supports the above conclusion. Chemically-bound water present in the Andesite was removed at 542 ºC. A low-density exothermal reaction was recorded at the highest level at 754 ºC. This may be due to crystallization of the second phase of the vitreous residue phase. The endothermic peak at 1056 ºC shows the melting point.

One of the most important factors that ensure a good harmony and bonding between the glaze and andesite body is the harmony between the thermal expansion coefficients of the glaze and andesite body. Knowing the thermal expansion behavior of the andesite body is of great importance for the selection of a newly-developed adaptable glaze. To this end, before the thermal treatment, the thermal expansion behaviors of the andesite samples were measured between room temperature and 700 ºC. The dilatometric curve of the K1-andesite sample (Figure 8) shows that the shrinkage rate of the sample was 0.36% at 360 ºC; that there was a linear increase in the shrinkage rate in parallel with the temperature increase; that the linear shrinkage rate reached 0.85% at 900 ºC. The linear expansion coefficient (α) of the andesite at 400 ºC was 325.82×10^{-8} (1/K) (Fig.8).

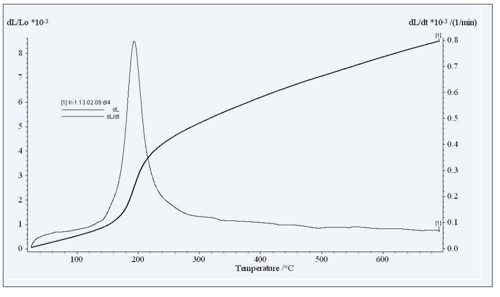

Figure 8. Dilatometer curve of the K1 Andesite.

Optical dilatometer test results of the K1 andesite sample are given in Fig.9. The optical dilatometer analysis showed that the sintering process of the K1 andesite sample started at 1134 ºC.

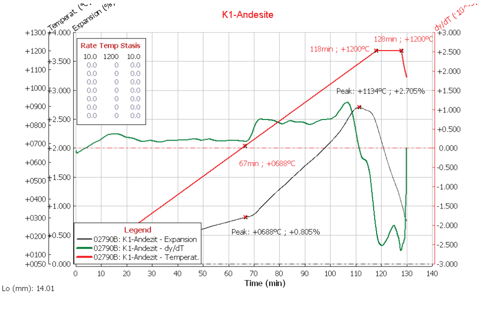

Figure 9. Sintering curve of the K1 andesite sample.

Figure 10 presents the heat microscope analysis results of the AS-1 and AS-2 glazes with K1-Andesite sample. According to the heat microscope measurements of the K1 andesite sample, maximum sintering was recorded at 1138 º C.

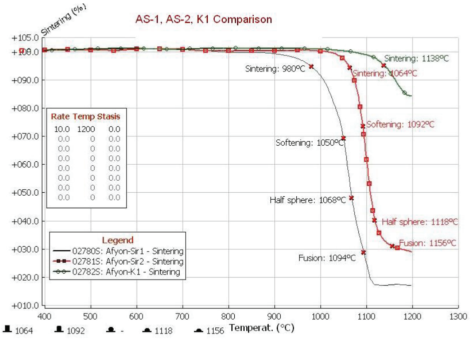

Figure 10. Heat microscope comparison of AS-1, AS-2 and K1-andesite sample

The AS-1 glaze mixture started to melt at 1055 º C and the AS-2 mixture at 1092 ºC (softening point). The AS-1 glaze sample was observed to start sintering at 980 ºC and the AS-2 glaze sample at 1064 ºC (Fig.11).

According to these findings, the AS-1 glaze sample sintered at a temperature 158 ºC lower than that of the K1 sample, while the AS-2 glaze sample at a temperature 74 ºC lower than that of the K1 sample.

3.2 Glaze Application

Different surface finishes were developed by applying underglaze and overglaze decoration techniques to the andesite samples up to 1160 ºC, by paying attention to the harmony of the samples with the artistic glazes. The aim of these applications was to enable use of the andesite material in indoor and outdoor spaces for artistic and decorative purposes, by applying ceramic clay and decorative techniques. Glaze recipes to be applied to the andesite surfaces are listed in Table 3.

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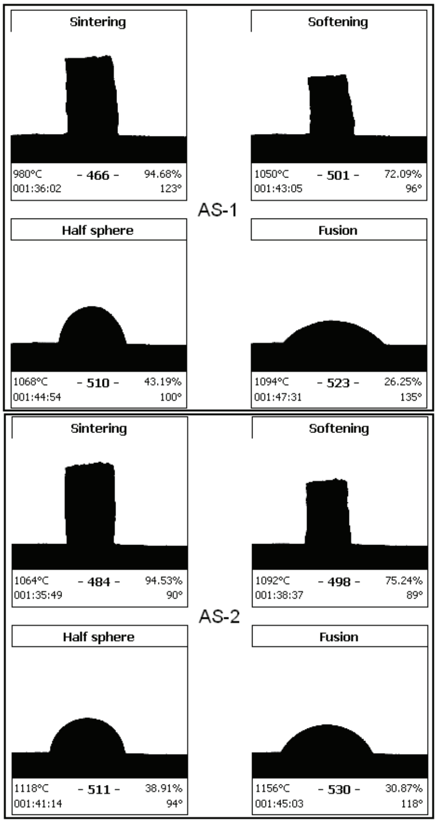

Figure 11. Heat microscope views of AS-1 and AS-2 glaze samples

Firing shrinkage value is of great importance for glazes (Kurama et.al., 2006). Particles get closer to each other and the size of the pores becomes smaller as a result of the sintering process. These dimensional changes result in firing shrinkage. Removal of the water in the chemical materials constituting the recipe and in the andesite; degradation of the crystallized structure, and; removal of the organic substances and hydroxyls during the firing process of the glazes, resulted in glaze shrinkage and weight loss in proportion to the increase in firing temperature (in ºC). As a result of the degradation of the $CuCO_3$ in the AS-1 glaze recipe, the firing shrinkage value of the AS-1 recipe (2.48%) was higher than that of the AS-2 recipe (Table 3).

Table 3. Glaze recipe applied to the andesite surfaces and firing shrinkage at 1160 ºC

Glaze Code	*Glaze Recipe*	*Rate (%)*	*Temper. ºC*	*Firing Shrinkage*
AS-1	Transparent Glaze	80	1160	2.48
	$CuCO_3$	10		
	PbO	10		
AS-2	Transparent Glaze	80	1160	1.40
	CoO	10		
	PbO	10		

There was no problem in terms of glaze and base harmony in the AS-1 sample. Some andesite-induced pores developed. A metallic effect was not observed in the glaze, which had a glossy surface appearance. It produced a smooth surface, which was metallic green in color. Purple points were observed on the blue background in the AS-2 glaze. There were andesite-induced point-shaped holes on the surface (Fig.12). The andesite samples fired at 1160 ºC were shown to have good glaze-base harmony and to hold the glaze firmly. Glazes developed at the same temperature provided the andesite surfaces with a glossy and opaque appearance. Andesite surfaces with solid colors and artistic effects were also created.

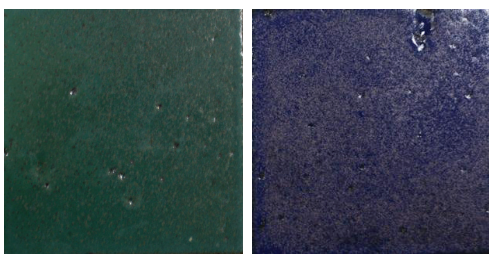

Figure 12. Glaze appearances on the andesite surface.

3.3 Serigraphy Print Applications

A serigraphy technique was applied by using the pattern paint/medium mixture prepared for decorative purposes to perform the printing process (Figure 13). At the end of the trial, very rare aggregations were detected on the surface. Red paint was observed to turn dark red at 1160 °C, due to firing color.

Figure 13. Serigraphy application via underglaze method.

4. CONCLUSIONS

The results of the experimental studies are summarized as follows; (i) XRD analyses made of the K1-Andesite sample revealed the presence of sanidine and mica minerals. (ii) Loss of ignition value determined in the chemical analysis of the K1-Andesite was 0.6% and the mass loss recorded in the DTG-TG curve was 0.76. Thus, these two losses were observed to be at a similar level. (iii) The liquid phase amount increased at the end of the thermal treatment at 1055 ºC. This increase resulted in an increase in sintering and a decrease in porosity. (iv) The sintering process of the K-1 andesite sample was found to start at ~ 1160 ºC in the two analyses made via optical dilatometer and heat microscope. Accordingly, the most appropriate temperature in terms of glaze-base harmony was determined to be 1160 ºC. (v) The AS-1 glaze sample was observed to start sintering at 980 ºC and the AS-2 glaze sample at 1064 ºC. (vi) The harmony of paint/medium mixture was achieved at 1160 ºC in the tests of serigraphy printing using the underglaze method.

ACKNOWLEDGEMENTS

This study was financially supported by the TÜBİTAK–1001 Project (Project No: 108M027). We would like to thank TÜBİTAK for its valuable contributions.

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Geochemical Study of Effects of Lignite Firing in Power Stations: Kostolac, Serbia

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ABSTRACT Lignite of the Neogene Kostolac Basin in Serbia, mining by the open pit Drmno (present production 6 Mt/y, up to 12 Mt/y 2012, working up to 2042), is fed into two power plant (PP). The complex geochemical, chemical and minarolgical studies of coal seams in the basin idicated that there are signicant variations of calorific value and geochemical features (contents of heavy metals/harmful trace elements: Pb, Cr, Ni, As, Hg ...) of coal and inorganic matter in the coal, vertically and laterally.

The geochemical studies of slag/ash produced in PP discovered several minerals (quartz, rutile, hematite, magnetite, vüstite, metal/alloy, glass) and indicated potential distributions of toxic and harmful elements in the system slag/ash-fly ash-gasous products during lignite firing in power stations. Variability in both mineralogical and geochemical features have been found. Based on these, efficiency of firing processes and their impacts to the environment have been considered.

1 INTRODUCTION

Large lignite deposits appear to be over the world subjects of massive lignite mining to feed large electric power stations (EPSs). Lignite consists of fossil organic (coal) matter and rather significant content of inorganic (mineral) matter, which appear in variable contents across the lignite deposits.

Lignite, and coal in general, may also vary significantly in its geochemical features, as may contain some major and numerous trace elements harmful to the environment, as sulfur, arsenic, lead, nickel, etc. (Dangić, 1992, 2001, Dangić & Dangić 2007, Dangić & Dimitrijević 2001, Dangić & Putnik 1998, Swaine & Goodarzi 1995, Wang et al. 2008, Yan et al. 1999, Zubovic et all. 1960).

As lignite is a low-rank coal, its firing in EPSs produces large quantities of elecric power also large masses of gasous and solid waste products which appear to be dangerous to the environment. Large masses of solid waste products appear to be deposited in the geo-environment and may pollute surrounding geologic media – soil and ground and surface water.

Both the production of electric power and the environmental impacts depend in general on coal quality and its monitoring.

Rationality of lignite usage in EPSs depend on both efficiency of lignite firing process and expenses of environmental protection from the coal firing waste products. Accordingly, it is very important to monitor quality of coal feeding EPSs and waste products of coal firing and their impacts to the environment (Dangić, 1992, 1995, Dangić & Dangić 2007, Dangić et al, 1993, 1998, 2008, Mardon & Hower 2004, Vassilev & Vassileva 1996, Ward, 2002).

The Neogene Kostolac Basin in Serbia (600 km^2) is one of biggest lignite basins in Europe. It is also one of most significant energetic resourses of Serbia and is a base for operation two EPSs feeding lignite, situated in the basin. At the present the most important lignite resources in the basin belong to the Drmno deposit, situated in the eastern part of the basin. The Drmno deposit is mining by an open pit which is at present only one lignite producer in the basin.

The solid waste products of coal firing in EPSs are depositetd in the nearby geological environment to form a large ash deposit. The polluted water forming in the ash deposit drains into the Danube River.

2 GEOLOGY, MINING, POWER STATIONS

The Kostolac Pliocene Basin is located around 80 km at east of Belgrad, the capital of Serbia, and is situated at the right bank of the Danube River (at south of the Danube). It is as large as 600 km^2. In the basin there are three coal (lignite) seams: seam I, seam II and seam III (from the surface downstears).

The geological exploration and mining exploitation of lignite in the basin started more than hundred years ago. However, the exploration have been mostly concentrated to the eastern part of the basin and coal was and is mining in this part of the basin only. Most of the coal resources in the eastern part of the basin belongs to the Drmno coal deposit.

The Drmno deposit contains two coal seams: the coal seam II and the coal seam III. The coal seam III is more significant, as it is developed continually accross the deposit and is as thick as up to 20 m. In rim parts of the basin MCM is shalow and homogenous but towards inside of the basin it becomes progresively deeper and thicker and intercallated with increasing amount of barren rocky materials. The coal seam II is less important, due to its restricted development and small thickness.

The present and future operations of EPSs in the basin (up to 2042) are based on the resources of the coal seam III in the Drmno deposit, which is 16-20 m in thickness and has a small slope toward NW and slowly decline toward the north. It is mined by the open pit (OP) Drmno.

In the OP Drmno, from 1987 (started the operation) up to the end of 2007, total 83x10^6 t coal was excavated and the present usable coal resources are as much coal as over 400x10^6 t.

Lignite mining in the OP Drmno is feeding to fed both EPSs in the basin: Kostolac A and Kostolac B, with the following install powers: 310 MW and 697 MW, respectively (in total 1007 MW).

The EPSs fire as much coal as 8.5x10^6 t/year, producing large masses of gaseous and solid wastes. The solid coal firing waste products - coal ash and slag, , as much as around 3x10^6 t/year, appear to be transported from EPSs hydraulically (as a water suspension) and deposited in adjacent geological environment to form large ash deposits. Due to specific geochemical features of ash, they appear to be dangerous sources of pollution of surrounding ground and surface waters, soil and air.

3 MATERIAL AND METHODS

Both organic and inorganic matter of the coal seam III have been studied. Bulk lignite samples, represented large intervals of the vertical profile of the coal seam as well as several individual samples and subsamples have been studied.

Field and lab studies involving macroscopic, microscopic and several physicochemical investigations have been carried out. In petrographic and mineralogical studies of coal and mineral matter the optical (binocular, polished and thin section microscopy) and X-ray diffraction (XRD) have been applied.

In geochemical studies, major and trace elements have been analysed by X-ray fluorescence (XRF), atomic apsorption spectroscopy (AAS), and clasical chemical analyses have been applied.

4 GEOCHEMICAL FEATURES OF LIGNITE FED TO EPS AND ITS FIRING PRODUCTS

4.1 Geochemical Features of Lignite

Lignite in the Drmno deposit significantly varies in its technical features, in vertical as well as in lateral profiles of the coal seam. The moister, ash and combustible matter vary as much as (in wt. %): 14.39-52.60 (mean 38.15), 0.28-52.60 (17.21), and 20.16-69.27 (44.63), respectively (Table 1).

Table 1. Technical features of coal in the deposit.

Parameter	n*	Range	Mean**
Moister, %	1217	14.39-52.60	38.15
Ash, %	1217	0.28-53.48	17.21
Combust.,%	815	20.16-69.27	44.63
GCV, kJ/kg§	1086	4195-19557	11823
NCV, kJ/kg§	1188	1112-18221	10399

* Number of analyses. ** Arithmetic mean.
§ Calorific values: GCV= gross, NCV=net.

The calorific values, gross and net ones, are in the ranges (in kJ/kg): 4195-19557 (mean 11823) and 1112-18221 (10399), respectively.

Petrographic composition of the coal matter as well as mineralogical composition of the inorganic (mineral) matter in lignite varies also significantly.

Lignite seam consists of the following coal lithotypes, which appear in variable proportions across coal seam profiles: pit (dominates) textite (xylithe) and doplerite coal (gelified plant tissue).

Table 2. Chemical composition of 15 samples of coal (wt. % of dry matter)

	Range	Average*
SiO_2	11.11 – 28.78	14.91
TiO_2	0.04 – 0.30	0.19
Al_2O_3	0.38 – 12.49	6.05
Fe_2O_3	0.73 – 3.69	1.90
MgO	0.13 – 1.42	0.66
CaO	1.73 – 3.69	2.71
Na_2O	0.01 – 0.20	0.09
K_2O	0.02 – 0.46	0.23
SO_3	3.37 – 6.52	4.88
P_2O_5	0.02 – 0.04	0.03

* Arithmetic mean.

Table 3. Trace elements abundance in 15 samples of coal (mg/kg of dry matter)

	Range	Average*
As	16 – 70	41
B	390 – 1130	727
Ba	280 – 320	297
Be	<1	<1
Cl	270 – 590	368
Co	2 – 6	4
Cr	50 – 60	53
Cu	10 – 20	11
F	10 – 40	26
Hg	0,11 – 0,41	0,27
Mn	80 – 430	190
Ni	10 – 40	19
Pb	3 – 32	10
Se	<1	<1
Sr	80 – 230	107
Th	<1	<1
U	<10	<10
V	20 – 50	37
W	<10	<10
Zn	10 – 30	12
Zr	9 – 140	97

* Arithmetic mean.

The microscopic petrographic studies found thet there are also wide variations of contents of coal micro lithotypes: detrite, gelite, fuzite and resins, as well as mineral admixures of pyrite and clayey matter, in both vertical and lateral profile of the cola seam across the deposit. Most typically coal consists of gelled xylithic ground mass which contains coal macerals (gelinite, cutinite, microspores), impregnations of clayey matter and dispersed pyrite (Fig. 1). Pyrite appears to form isolated grains (in size mostly up to 0.1 mm) and mineral agregates.

Mineral matter in the coal seam appears mainly as irregulary distributed admixtures and interbeds of clay, aleurite and marl and, localy, as thiny epigenetic calcite, gypsum and calcedone mineraliaztions.

The clayey admixtures consist of kaolinite, smectite, quartz and thin particles of coal matter. The marly materials consist of 45-54 % clay, 34-46 % calcite and 9-12 % coal matter. The aleurite consists of 12-30 % sandy, 42-74 % aleuritic and 15-29 % clayey fractions; the sandy fraction consists of quartz, coal matter, rock fragments, micas, carbonates and traces of heavy minerals.

Geochemical features of coal vary also significantly (Table 2 and 3).

In chemical composition of coal silica (SiO_2) appears to be most abundant (11.11-28.78 %, on dry bases), and is followed by alumina (0.38-12.49 %), sulfur (as SO_3 3.37-6.52 %), iron (as Fe_2O_3 0.73-3.69 %), CaO and MgO (1.73-3.69 and 0.13-1.42 %, respectively); Na_2O and K_2O appear in contents up to 0.46 % and P_2O_5 up to 0,04 %.

Numerous heavy metals and other trace elements appear in coal in variable contents: As, B, Ba, Cl, Co, Cr, Cu, F, Hg, Mn, Ni, Pb, Sr, V, Zn and Zr.

Most abundent are B and Cl (390–1130 and 270–590 mg/kg, respectively). Mn, Ba, Sr and Zr appear in ranges 80–430, 280–320, 80–230 and 9–140 mg/kg, respectively.

As, Cr, and V are as abundant as up to 50-70 mg/kg. Ni, Zn, Pb, Cu, and F are as abundant as up to 20-40 mg/kg. Co appears in contents 2-6 mg/kg and Hg 0,11-0,41 mg/kg.

The following elements appear to be bellow analythical detection limits (in mg/kg): Be, Se, Th (all <1) , U and W (both <10).

4.2 Geochemical features of waste solid lignite firing products

Massive lignite firing in EPS produces large masses of waste gaseous and solid products. Both waste gases and ash/slag appear to be dangerous to the environment. The solid products – ash/slag, appear to be collected by electro-filters and afterwards removed from EPS by a hydraulic transportation (as a water suspension) into the ash deposits, which is situated in the geo-environment.

Three samples of solid firing waste products have been complexly mineralogically and geochemically studied: one sample of electro-filter (EF) ash and two samples of ash/slag transported to the ash deposit. In further text all these materials are treated ash.

It was found that ash is a material of variable mineralogical, geochemical and physicochemical features.

Microscopic studies indicated that ash consists mostly of particles and aggregates of silicate glass which are accompanied by diverse silicate minerals, some anhydrite/gypsum, metallic minerals and thin particles of unfired coal.

The X-ray powder diffraction (XRD) analyses have been carried out for one sample of electro-filter ash (from the block 2 of EPS) and ash/slag transported to the ash deposit (from blocks 1 and 2 of EPS). The following mineral phases have been determined by XRD (Tab. 3): quartz (SiO_2), feldspar ($CaAl_2Si_2O_8$), mullite ($Al_6Si_2O_{13}$), gellenite ($Ca_2Al_2SiO_7$), magnetite (Fe_3O_4), hematite (Fe_2O_3), anhydrite ($CaSO_4$), and gypsum ($CaSO_4 \cdot 2H_2O$). Anhydrite appears in EF ash and gypsum in ash transported to the ash deposit. Gypsum appears to be a secondary product, formed by hydration (in interaction with water used for ash hydraulic transportation).

Table 3. Results of X-ray powder diffraction analysis of electro-filter ash (EF) and ash/slag from the ash deposit (AS1, AS2).

Mineral	Formula	EF	AS1	AS2
Quartz	SiO_2	+	+	+
Feldspar	$CaAl_2Si_2O_8$	+	+	+
Mullite	$Al_6Si_2O_{13}$	+	+	+
Gellenite	$Ca_2Al_2SiO_7$		+	+
Magnetite	Fe_3O_4	+	+	+
Hematite	Fe_2O_3	+	+	+
Anhydrite	$CaSO_4$	+		
Gypsum	$CaSO_4\ 2H_2O$		+	+

The metallic minerals have been studied microscopically, in several samples of polished sections. They appear as small grains (up to few tens of μm in size) dispersed in silicate matrix. Several sulfide, native metal oxide and oxy-hydroxide phases have been determined (Tab. 4).

Table 4. Metalic mineral phases in coal ash/slag from EPS, blocks 1 (AS1) and 2 (AS2).

Mineral phase	Formula	AS1	AS2
Pyrrhottite	FeS		+
Covellite	CuS	+	
Iron	Fe	+	+
Copper	Cu	+	+
Wüstite	FeO	+	+
Magnetite	FeO_3O_4	+	+
Maghemite	Fe_2O_3	+	+
Hematite	Fe_2O_3	+	+
Goethite	FeOOH	+	+

Among metallic minerals, iron minerals predominate. The iron minerals have been formed mainly by pyrite transformations during coal firing process.

The following iron minerals are identified in both EF ash and deposit ash: pyrrhottite (FeS), iron (Fe), wüstite (FeO), magnetite ($FeO{\cdot}Fe_2O_3$), hematite (Fe_2O_3) and maghemite (Fe_2O_3).

In deposit ash also goethite (FeOOH) appears as a secondary product, formed by hydration of hematite and maghemite (in interaction with water used for ash hydraulic transportation).

Table 5. Chemical composition of coal ash/slag and electro-filter ash (EFA) from EPS Kostolac (in wt. %).

	Ash/Slag	EFA
SiO_2	49.84-52.1	50.03
TiO_2	0.58	0.60
Al_2O_3	17.98-20.22	18.68
Fe_2O_3	8.13-9.67	7.71
MgO	1.69-1.86	1.77
CaO	6.59-7.92	6.85
Na_2O	0.34-0.39	0.39
K_2O	0.83-0.93	0.79
SO_3	7.87-9.17	11.42
P_2O_5	0.04	0.04
Sum		98.28

The chemical composition and contents of heavy metals and other trace elements have been analyzed for one sample of EF ash and two samples of ash/slag from the ash deposit (Table 5).

Chemical compositions of EF ash and deposit ash are rather similar (Table 5). The predominate component is SiO_2 (around 50 % based on dry matter), wich is followed Al_2O_3 (cca 18-20 %), Fe_2O_3 (cca 7-10 %), CaO (cca 6,5-8 %) and SO_3 (cca 8-11 %).

Due to significant variations of chemical composition of lignite in the deposit (Dangić et al., 2009 b), one can expect also significant variations of chemical composition of ash producing in and transporting from EPS during the time.

In EF ash and deposit ash the following heavy metals and other trace elements have been analyzed and detected: As, B, Ba, Cr, Cu, Ga, Li, Mn, Mo, Nb, Ni, Pb, Rb, Sc, Sr, V, Zn, Zr i Y (Table 6).

The most abundant among these elements are (in mg/kg): manganese (as abundant as around 1000), boron (up to cca 800), zirconium (up to near 700), and chromium, copper, scandium and strontium (up to cca 300).

Table 6. Contents of heavy metals and other trace elements in coal ash/slag and electro filter ash (EFA) from EPS Kostolac (in mg/kg of dry matter).

	Slag/Ash	EFP
As	120-125	148
B	709-770	667
Ba	401-428	425
Cr	200-207	242
Cu	182-234	203
Ga	19-20	15
Li	63-74	54
Mn	622-995	672
Mo	2-3	2
Nb	11-13	15
Ni	107-131	168
Pb	31-33	25
Rb	20-25	15
Sc	227-232	222
Sr	268-301	287
V	187-200	193
Y	10-13	15
Zn	106-134	188
Zr	575-670	672

The rather abundant are vanadium (up to 200 mg/kg), arsenic, nickel and zinc (up to cca 150 mg/kg) and lithium (up to near 100 mg/kg). The significantly less abundant are lead and rubidium (up to 35 mg/kg) and gallium, niobium and yttrium (up to 20 mg/kg). Molybdenum appears up to 5 mg/kg.

5 DISCUSSIONS AND CONCLUSION

The complex geochemical and physicochemical studies of the Drmno deposit indicated that there are significant variations of lignite quality in the deposit.

There are variations in coal petrographic and mineralogical composition of both organic and inorganic matter, moister, ash content, calorific values and chemical composition and contents of heavy metals and other trace elements.

It was found that both coal organic and inorganic matter may contain heavy metals and other trace elements which may be dangerous risk to the environment during/after coal firing in EPS.

The variations of presence and interrelations of diverse lithotypes and microlithotypes of coal organic matter as well as mineral and chemical composition of inorganic metter (clay, sandy and marly components; pyrite/gypsum ratio) may significantly impact to the process of coal firing in EPS and thermical effects.

The variations of contents of heavy metals and other trace elements may produce also variable risks to the environment.

The studies of ash/slag indicated that the solid waste products of coal firing in EPS are rather complex in mineralogical and geochemical features.

Compared with (mother) coal, the ash/slag material is characterized by ocurrences of several new (secondary) non-metallic and metallic minerals and by concentrations of most of heavy metals and other trace elements.

We found that geochemical studies of metallic mineral phases in coal and ash/slag, especially iron minerals, may be used to indicate the geochemical features of the coal firing process in EPS (Dangić et al., 2008).

Pyrite is the main or only iron mineral in lignite (Fig. 1). During coal firing in EPS, pyrite appears to be transformed in several secondary minerals (Fig. 2).

One can conclude that ways and products of pyrite transformation depend on thermodynamic-geochemical conditions of coal firing.

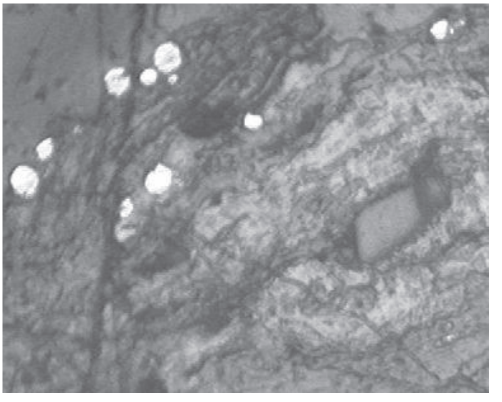

Figure 1. Microphotographs of coal from KB: the gelled xylithic ground mass containing coal macerals (gelinite, cutinite, microspores), impregnation of clayey matter and dispersed pyrite (bright grains). Size of picture 2x1.5 mm (magnification: 40 X).

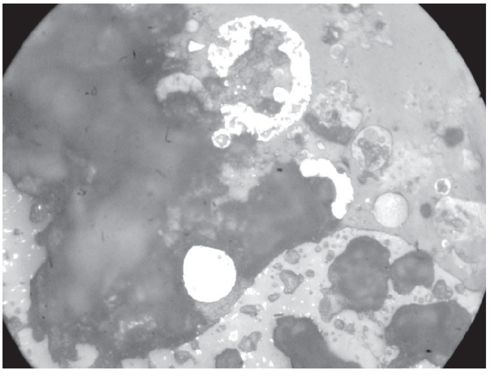

Figure 2. Microphotograph of coal ash from EPS: magnetite and wüstite (bright) pseudomorphs after former spheroid and worm-like gel pyrite in a silicate ground mass. Size of picture 0.8x0.6 mm (magnification 100 X).

Accordingly, based on thermodynamic-geochemical conditions of secondary iron mineral stabilities, we used these minerals as indicators of redox conditions (oxygen regime) during coal firing in EPS (Table 7).

Finding of pyrrhottite indicates that local conditions have been much reduced, i.e. characterized by a significant oxygen deficiency. Namely, pyrite fired but due to oxygen deficiency sulfur remained in the form of sulfide which binds iron to form FeS. Accordingly, in that system all coal particles could not be entirely fired.

Table 7. Mineral phases formed by transformation of pyrite during coal firing in EPS and redox conditions of their formation.

Mineral phase	Compo-sition	Redox conditions
Pyrrhottite	FeS	very reduced
Iron	Fe	reduced
Wüstite	FeO	slightly reduced
Magnetite	Fe_3O_4	imperfect oxidized
Hematite	Fe_2O_3	oxidized

The metallic iron indicates less reduced conditions, under which sulfur have been oxidized to form SO_2 but iron was reduced from Fe^{2+} into Fe^o state. Also, coal could not be entirely fired.

Wüstite indicates slightly reduced and magnetite less reduced but imperfect oxidized conditions.

Appearance of hematite, on the other hand, indicates the oxidized conditions, wich enabled full oxidation of Fe^{2+} into Fe^{3+}. Under that conditions all coal particles need to be fired.

Taking into account all presented results and their discussion, one can conclude that for more rational utilization of coal in EPS it would be necessary to systematic and complex study and monitor quality of both coal feeding EPS and its solid firing waste products. It needs to involve petrographic composition of organic matter, mineral composition of mineral matter and geochemcial features (major and trace elements) of coal feeding EPS as well as relevant studies of coal firing products.

The proposed approach may enable that manage of both the process of coal firing in EPS and protection of the environment be more efficient and more profitable for the mining company producing coal as well as for the EPS consuming coal.

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Velenje Coal Mine (VCM) Mining Method and Modern Mechanized Faces

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ABSTRACT One of the thickest seams of lignite coal (up to 170 m) in the world has fostered the development of innovative mining methods. This paper explains high productive Velenje longwall mining method. Its process does not leave empty areas behind, as the hanging wall layers collapse into the unfilled spaces as the mining progresses.

The longwall faces in which only the lower excavation sector is mined reached a length up to 210 m. In year 2008 first mine face with such length successfully started. Face is equipped with modern state of the art equipment.

Paper presents the safety and technology information system (VTIS) which is a surface operated monitoring system of the pit combining the parameters of safety and technical control and management of the mine: machine operation, ventilation, drainage, electrical energy, compressed air, water etc. Automation of main coal transport and data transfer through optical network to the control center is described.

1 VELENJE COAL MINE (VCM) PRESENTATION

1.1 Velenje Coal Mine Presentation and Strategy

Velenje Coal Mine is a company with a tradition over 130 years long, a solid present and an enterprising orientation toward the future. This joint stock company, the basic activity of which is production of lignite coal, is inseparably connected with the constant supply of electrical energy to Slovenia as well as with the current and future development of the Šalek Valley. The primary engagements of the company also include environmental care and improving the quality of life for people who are directly and indirectly related to the company.

Velenje Coal Mine is one of the largest and most modern deep coal mining sites in Europe. Its operation and strategic orientation comply with Slovenia's National Energy Programme. The related long-term mining plan for Velenje Coal Mine projects a gradual consumption of the available coal reserves in accordance with the energy demand of the Šoštanj Thermal Power Station up to their complete exhaustion by 2040.

The annual output of Velenje Coal Mine is approximately 4 million tonnes of lignite coal, which is used up entirely by the Šoštanj Thermal Power Station for the production of electricity and heat. Thus, the mine represents one of the main pillars supporting the Slovene energy network, as together with the Šoštanj Thermal Power Plant it provides for a third of all domestically produced energy. As such it is an important and reliable link in Slovenia's electricity supply.

Velenje Coal Mine consists of the parent company and several companies in which the Mine holds equities.

The mine and company's operation is based on four strategic goals:

- Rationalization of the coal excavation process.
- Providing safety and humane conditions of work throughout the working process.
- Solving environmental problems.
- Intensive restructuring of affiliated undertakings and expert services of Velenje Coal Mine, as well as the foundation of new companies.

The greatest asset of the company is its people, who employ their knowledge and skills to successfully manage the very demanding working conditions. They are distinguished for their expertise, innovativeness and loyalty, and connected by both a comradely spirit and tradition.

1.1.1 Mission

The mission of Velenje Coal Mine, the leading Slovene company in the field of mining, is long-term production of coal for the purposes of generating electrical energy in the Šalek Valley in accordance with the principles of sustainable development, as well as maintenance and development of the knowledge of deep mining works.

1.1.2 Vision

Thanks to the long-term orientation of its business and modern production process, Velenje Coal Mine will jointly with the Šoštanj Thermal Power Station (ŠTPS) manage a suitable exploitation of Slovenia's only strategic energy source, lignite coal from the Šalek Valley, while all the time providing safety and humane work conditions.

1.1.3 Values

The satisfaction of the key public owners, employees, buyers and the local environment lies in creative harmony:

- Ensuring the owners a suitable yield from the capital invested.
- Providing the employees with quality jobs and development of personal potential.
- Ensuring safety at work, humanized jobs and continuous training.
- Promoting knowledge, expertise and the spirit of innovation, guaranteeing to the buyers a consistent fulfilment of contractual obligations in coal supply in order to ensure long-term economic success.
- Maintaining a good relationship with the local environment in accordance with the principles of sustainable development.

1.2 Velenje Coal Mine is a Company with Corporate Governance

The production of lignite coal and electrical energy stimulated the undreamt of development of the Šalek Valley, leaving on it indelible imprints with numerous changes to the living space. Mining as a process of underground excavation of coal is often viewed as environment unfriendly. Given the considerable economic and social usefulness of its results, in many cases the environment is sacrificed in favour of this important primary energy resource. However, the hostile effects of mining on the environment can be greatly reduced if the mining process is based on the principles of sustainable development.

Throughout its operation, Velenje Coal Mine has been greatly changing the image of the Šalek Valley and constantly altering the quality of life in it. It has taken a lot from the Valley, but has also given it a great deal in return. In fact, due to the direct and indirect impacts of the development of the mining business numerous jobs have opened in the Šalek Valley, residential and many other types of buildings have been constructed, infrastructure has been built, public services have developed.

Since the early 1980s, Velenje Coal Mine has been systematically remedying the consequences of the mining operations, which once were reflected primarily in the sinking and degradation of the surface. Velenje Coal Mine engagement has also stimulated a methodical spatial planning, which has resulted in new building areas in Velenje, Šoštanj and Škale.

A decade ago, Velenje Coal Mine started to actively participate in the municipal programme for improving the condition of water, soil and air in the Šalek Valley. On the banks of the lakes that had formed as a result of mining arose a modern tourist recreational centre that is continually updating and expanding its offer.

The environmental policy of Velenje Coal Mine originates in its strategic goals. One of them is being the care for a healthy working and living environment and the health of our employees as well as all inhabitants of the Šalek Valley, and the other concerning the prevention of environmental strains and the remedy of negative consequences of coal mining. Company has thus been constantly monitoring the impacts on the environment caused by the mining process, preventing its detrimental effects on the environment, abiding by the legislation requirements as well as internal and environmental protection regulations, and cultivating a constant communication with our employees and other publics with the purpose of preserving and improving mutual trust.

2 VELENJE LONGWALL MINING METHOD AND MODERN MEHANIZED FACES

One of the thickest seams of lignite coal in the world has fostered the development of innovative mining methods. The production of lignite coal in Velenje is studied scientifically, professionally planned and executed with state of the art technology. The entire process is based on the respect of natural characteristics, provision of safety and anticipation of impacts on the environment.

2.1 Natural Characteristics

The wide area geographically known as the Šalek Valley can be classified geologically as a tectonic depression that had been sinking between the Šoštanj, Velenje and Smrekovec faults. The valley was shaped to its current dimensions by sinking and the simultaneous accretion of sediments, part of the mass locked in it being the coal seam.

2.1.1 Geology, hydrogeology and geomechanics

The approach to the exploitation of a determinate ore deposit largely depends on the natural features in the ore layer and its environs. The basis for the planning and subsequent execution of mining works is geological data, which Velenje Coal Mine has been acquiring and processing since the beginning of the mine's operation.

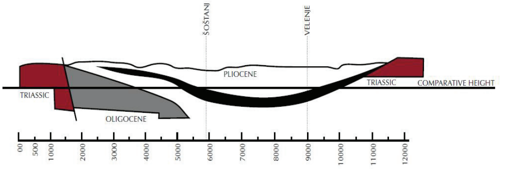

Figure 1. Position of the coal seam in the Šalek Basin.

The majority of geological data is gathered through exploratory drilling. More than 620 drill holes have been bored from the surface into the coal seam and the surrounding rock, which translate as "210 km of geological data." The geological database is rounded off by some 2100 shallower drill holes bored from underground workings.

2.1.1.1 Geology

The coal seam under the Šalek Valley stretches 8.3 km long and up to 2.5 km wide, at a depth between 200 m and 500 m. Its average thickness is 60 m, with maximum values reaching up to 170 m. The coal called lignite is relatively young. The origin of the Velenje lignite coal dates back to the period of late Tertiary, early Pliocene, of 2.5 million years ago. The framework of geological data processing also includes the calculation of mineral reserves. We estimate that the currently known technologies could recover from Velenje Coal Mine at least another 163 million tonnes of lignite coal.

2.1.1.2 Hydrogeology

Hydrogeology studies the mechanisms of the underground water flow in rock and soil aquifers.

The knowledge of hydrogeological parameters is of extreme importance for mining, as they provide the basis for planning safety measures against the intrusion of water into underground workings, while also enabling us to determine and preserve the drinking or process water abstraction points.

The process of lignite coal mining is accompanied by intensive drainage of aquiferous layers. In the hanging wall above the coal seam Pliocene sands are drained, while in the footwall the drainage of Triassic substratum and Lithotamnion limestone is necessary. In the past, the drainage activities established a stationary condition in the aquifers, which will have to be maintained till the mine is exhausted. For a safe and undisturbed operation of the mine at present 0.7 m^3 of water per 1 tonne of coal produced is drained.

2.1.1.3 Geomechanics

The knowledge of rock and soil behaviour is closely connected to laboratory and in situ studies as well as with computer or mathematical modelling of rock and soil behaviour in altered stress and strain conditions resulting from mining operations.

Seeing that the production (excavation) of coal is concentrating on an ever narrower area, an increasingly active involvement is necessary with:

- Safety pillar designing.
- Studies of the caving process.
- Prevention of sudden coal and gas outbursts.
- Stability of underground workings and face support systems.

The geotechnical measurements with which these processes can be tested represent constituent parts of coal mining technology.

2.2 Velenje Longwall Mining Method

The exceptional thickness of the lignite coal seam in the Šalek Valley prompted the Velenje miners to investigate the possibilities of extracting as much coal as possible while investing the least necessary amount of mining work. Therefore, they divided the coal seam into several levels at heights still manageable in view of the related natural characteristics and technical possibilities. After 1955, the longwall mining method with classic understructure prevailed. An increased level height was provided by blasting off the ceiling section, and with the mechanised hydraulic face support system gaining ground the development of the mining method picked up even more speed.

The basic concept of the Velenje longwall mining method is that the area of exploitation extends above the supported roof coal of the face, while the natural forces of the higher lying geological strata are exploited for breaking and crushing the coal seam above the hydraulic support system.

The longwall face is divided into the upper and lower excavation sections. The lower excavation section is supported up to a height of 4 m by a hydraulic prop system, which enables mechanized extraction cutting the coal seam with a shearer and removing it via high performance chain conveyors.

The upper excavation section, reaching up to 12 m high, is subject to dynamic rock stresses, which can in combination with loosening and tightening of the hydraulic prop system crush the upper excavation section of the coal seam, and the crumbled coal from it then only needs to be dropped onto a conveyor. Retreat mining has always been the preferred system of coal extraction in the coal mine of Velenje. For this method mine roadways have to be built first, and then the line of the faces retreats from the boundary toward the main underground structures.

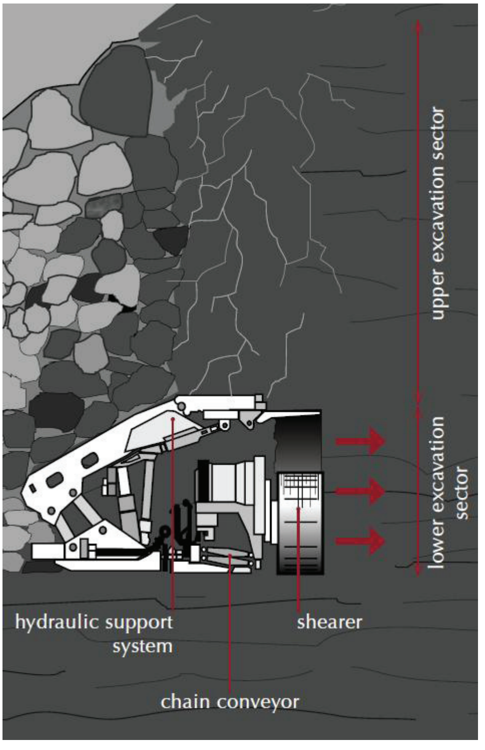

Figure 2. Lower and upper excavation sector.

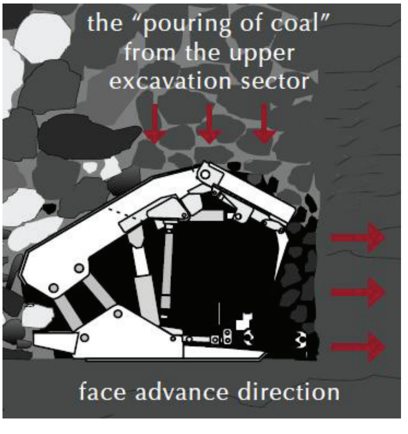

Figure 3. "Pouring of coal" from the upper excavation sector.

This process does not leave empty areas behind, as the hanging wall layers collapse into the unfilled spaces as the mining progresses.

In thick coal seams, the highest located levels are mined first. The mining operations on the upper levels do not disturb the lower laying coal to any considerable degree, while mining in the direction from the hanging wall towards the footwall maintains relatively good conditions for setting up underground constructions at lower levels.

2.3 Longwall Faces Extension

The allowed face height at the long wall depends on the thickness of clay insulating layers in the hanging wall, which protect the face from the inrush of sand and water. In the area of the VCM Pesje pit, where the insulation layers are extremely thick, the face height is limited by the technology of controlled coal extraction and the restraint of mining losses.

In the area of the VCM Preloge North pit the insulation layers are extremely thin, so the height of the face is set according to the criteria for safe mining. This means that only the lower excavation section is mined; that is, only the coal that can be cut with a shearer.

With the development of the mining equipment the longwall faces have become more and more efficient. Presently, the annual production is around 4 million tonnes and the amount is recovered from only three longwall faces. Owing to the improved capacity of the mining equipment longwall faces extended and the daily advancement increased. The current average length of the faces is 140 m, with the daily advancement ranging between 4 and 4.5 m. An average daily production at such a face runs between 8,000 and 11,000 tonnes.

In year 2008 Velenje Coal Mine established 210 m long face were only the lower excavation sector was mined. This was done for the first time in VCM history. Face was successfully excavated and also modern state of the art equipment was used. Before that the longwall faces in which only the lower excavation sector is mined reach a length of up to 160 m long. Daily advancement is ranging between 5 and 5.5 m. An average daily production at such faces runs between 4,000 and 6,000 tonnes.

3 VELENJE COAL MINE SAFETY AND TECHNOLOGY INFORMATION SYSTEM AND COAL TRANSPORT AUTOMATION

3.1 Safety and Technology Information System

The safety and technology information system (VTIS) is a surface operated monitoring system of the pit combining the parameters of safety and technical control and management of the mine: machine operation, ventilation, drainage, electrical energy, compressed air, water, number and names of the workers currently in the mine etc. The system is also connected with information support of determinate points located on the surface: fire control of surface structures and plans for all possible extraordinary events inside and outside the pit. All controls come together at the duty controller's post, whose responsibility is to monitor the data and take appropriate measures when necessary. The measures concern the coal excavation process and extraordinary events occurring in the area of Velenje Coal Mine. VTIS is also used for managing determined automated parts of the mining process.

The system was set up as early as 1975, and has been continually updated. Its conceptual design and implementation are the result of knowledge and work of Velenje Coal Mine experts. Every second VTIS receives immense amounts of data. Much of that is archived and later used for analyses and planning by technical services, while a great deal of data is, in addition to giving valuable reports on safety, used for technological operations. From here, information is also forwarded outside.

3.2 Automation of Main Coal Transport and Data Transfer Through Optical Network

At VCM main coal transport is done through several belt conveyors. Such transport is done from the mine to the surface. VCM goal was to modernize coal transport and enable remote control and management over conveyors.

3.2.1 Introduction

Control system of main coal transport through conveyors was previously done in relay technology. System was still reliable but remote control of automatic conveyors wasn't possible. Because of that and goals of VCM to rationalized working process and costs proper solution was find.

Solution is including automatic and remote control of six so called main transport conveyors. That is from conveyeor number 60 to number 10 which is running to the surface. Remote control is also possible on surface conveyor K2/1.

Automation and control is based on connections between process computers (PLC). Each conveyor has his own PLC which producer is OMRON. They are connected to optical knot through optical line. Optical knot is located in surveillance control center and serves as server for communication. Each PLC has his own IP address therefore communication between them is running through TCI/IP protocol. All PLC's are connected with central computer which is remotely controlled and managed. Schematic transport line is shown in Figure 4.

Each conveyor has simple command unit on which beside HMI terminal (touch screen) control elements with keys for service control are. Direction of coal transport is from conveyor T60 to T10. Direction of conveyors start is from T10 to T60. We use two technological expressions regarding mentioned directions:

- Previous conveyor. That means that for example T20 must be switched on before T30. Therefore operation of T20 is determined by T30 operation.
- Next conveyor. That means if for example T30 is next conveyor of conveyor T20 because it follows it in transport direction.

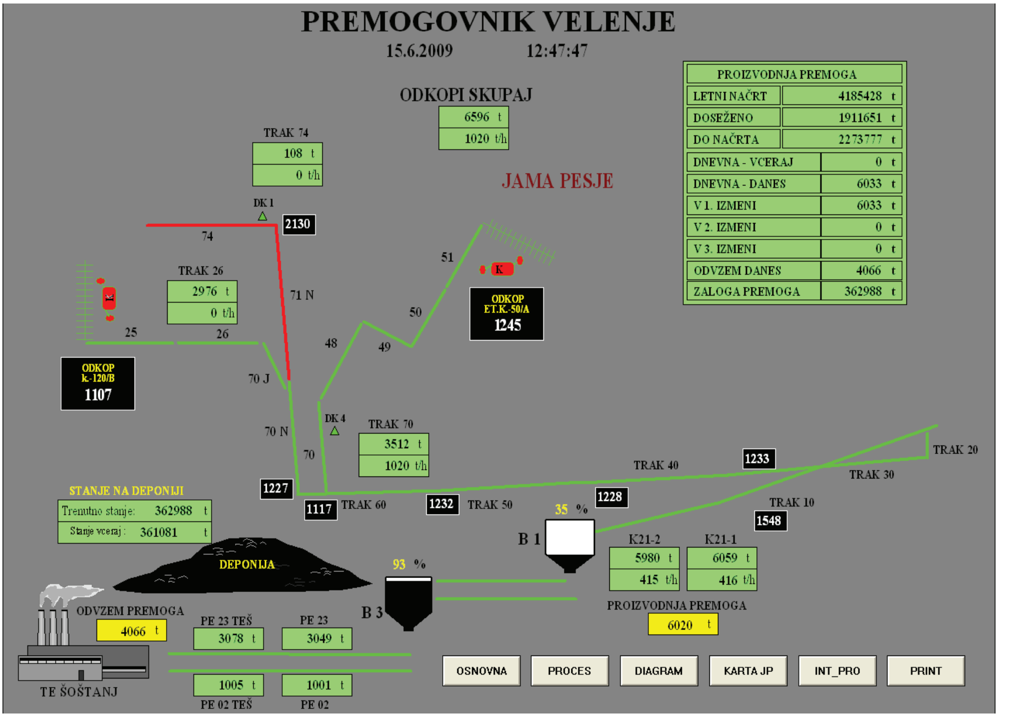

Figure 4. Schematic plan of coal transport from the mine to the surface.

3.2.2 Technological process control

Software on PLC enables operation of devices with consideration of all needed protection criteria's and interacting dependence regarding chosen operation regime. PLC is receiving information's about devices state through digital and analog input units. On base of received information's and program management of devices is performed. From control computer PLC's receives commands for line on or off switching and on base of that proper operations are performed. PLC's are receiving also demands for parameters change (device starting time, limitation for warnings or disconnection at analog signals, time delays etc.) on which base control is done. Software program on PLC's includes all digital and analog signals, counts devices working hours, counts number of motor and high voltage switches and generate warnings and alarms on central control computer.

For conveyors from T10 to T60 automation porpoise six process controls are installed (PLC). Each PLC takes care for one conveyor.

For example PLC 1 is in charge for conveyor T10 (E-1400) automation. Through modem NT terminal is connected to PLC 1. Because of that control and management of T10 conveyor is possible. Management through NT terminal is reserve option. It is used in case of control computer failure or at communication failure. The rest of conveyors are equally equipped.

Modern PLC's which are used for transport automation have integrated Ethernet port for smooth connection. On Ethernet port special connector Ethernet/Optic is connected. Through it connection of PLC to the Velenje Coal Mine optical spine is enable.

Velenje Coal Mine has mine information communication network which is build on base of copper cable connections. Through mentioned networks safety, technological and communication data's are entered. Through existing networks electrical circuits are lead. They are in different explosion protection. With development of technological solutions and ICT systems development trend of solutions which are already common in computer world are transferring to the mining field. Need to transfer more data's, fast transfer, transfer without looses on long distances, big reliability and system insensitivity for external disturbances bring optical technology to the mining field. Because of that optical spine was built at Velenje Coal Mine. It is shown on Figure 5.

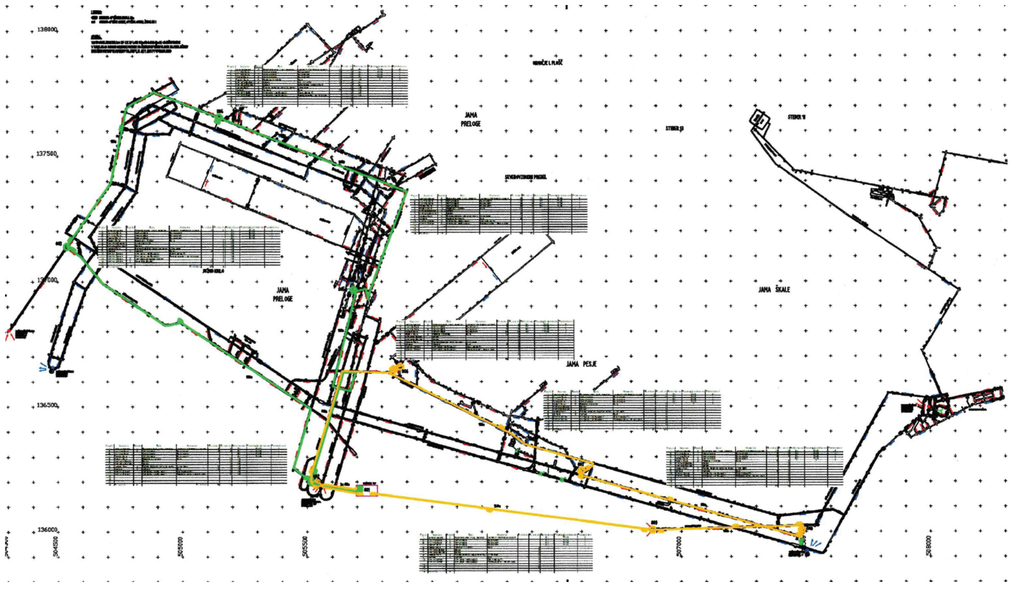

Figure 5. Velenje Coal Mine optical spine in the mine (pit).

3.2.3 Control computer

Control computer purpose is mostly automation line of conveyors T10 to T60 switching on and off.

Because of its high importance control computer is powered through UPS which assures continuous power supply. We can see schematic review of conveyor T10 on Figure 6. Software on control computer enables us:

- Schematic review of whole transport system line from conveyor T10 to T60 and remote control of line.
- Schematic review of individual conveyor.
- Schematic review of devices condition and measured values (temperature, currents and levels, working hours, number of motor switches, number of high voltage field switches etc.).
- Print of requested data's (alarms, levels, measures, on/off state).
- Review of current state (devices operation).
- Change of boundary values and operating parameters for each individual conveyor.
- Program bridging of specific parameters.
- Notice about failures, warnings, alarms and data archives.

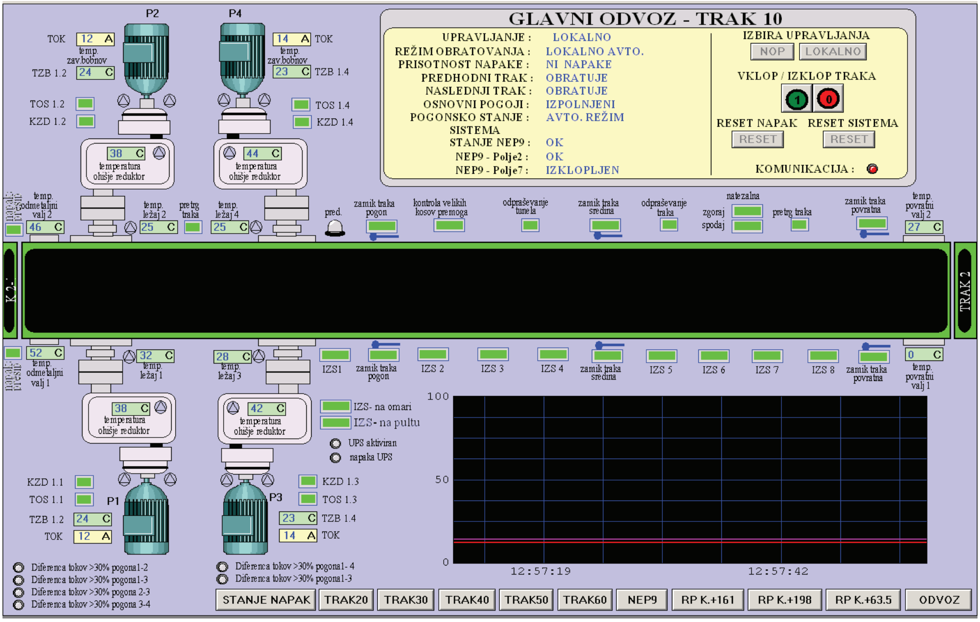

Figure 6. Schematic plan of T10 conveyor on control computer.

3.2.4 Conclusions

With coal transport from mine to surface automation remote control and management with permanent control over transport line from one place is assured. On one place all key data's about system operation are gathered. Such operation assures easier system maintenance, faster fault solving, less jams and optimal burdening of transport line.

Project is result of knowledge and experience of Velenje Coal Mine workers. It is additional contribution to fulfill the vision of Slovenian leading company on mining field. Velenje Coal Mine is orientated to modern production process implementation with contemporary health and safety assurance.

Implementation of main transport line automation system represents base for upgrade with central system of coal loading to the main transport line. With that optimal burdening of whole transport system and cost reduction will be assured.

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Ellatzite Mine – The Biggest Open Pit Mine for Copper and Gold Production in Bulgaria and in the Balkan Peninsula

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ABSTRACT Ellatzite deposit is located near the town of Etropole, about 80 km to the East of Sofia. Ellatzite open pit mine is exploited by traditional classic development system using up-to-date high capacity mining equipment manufactured by leading world producers. Massive vertical copper-gold ore body of cylindrical shape is mined with low grade payable elements in the ore.

Daily run-of-mime extraction is about 153 000t; the share of ore is about 35 000t. At the moment the pit wall height is about 520m, decreasing to the east as a function of terrain topography. Now the actual overall slope angle of mining operations is 45 degrees, for both bench heights, i.e.15m and 30m benches.

Ellatzite deposit is operated by ELLATZITE MED AD, a company of the group GEOTECHMIN AD, SOFIA.

The last third expansion of Ellatzite open pit mine lowers the final pit floor to an elevation of 880m, thereby the maximum slope angle to be increased to 48°. At the time of mine operation completion Ellatzite mine length would be 1400 m north-northeast-south-southwest, 1900m in west-northwest-east-northeast direction. Thus, last enlargement effected in an increase of copper ore reserves by about 60M t more as to 01.01.2009. Thereby the pit walls will be some of the longest and steepest in the world.

1 ELLATZITE DEPOSIT, GENERAL INFORMATION

Ellatzite deposit is located about 80km to the East of Sofia on the Northern flank of Etropolska Stara Planina, near the town of Etropole. The distance from Etropole to Ellatzite Mine is 14 km.

Etropole area was known as an ore-mining district since the Middle Ages. There exist historical documents where it can be seen that town of Etropole has been one of the major suppliers of iron and gold to the Ottoman state.

First information on geological formations and their lithological species presents the Frenchman Amy Bouie, who while making his tour over Etropole area in 1837, found that there is a presence of Paleozoic material, massive rocks that he has defined as syenite, also and these are of an earlier age than their host rocks. Considerably later, during 1880 Franz Toula researched over the Etropolski Balkan. N.Puskarov and G.Bonchev have examined the geological texture of Ertopole area; and yet the entire geological prospecting has made B.Kamenov during 1933-1934. As a result “Geology of Etropole area” came into sight, where he presented comparatively full information on stratigraphy and tectonic structure of the researched area.

Not until 1951-1954 systematical prospecting work of rather large scale has been undertaken, started by M. Konstantinov, continued by A. Ushev and finished by

S.Kalaidjiev. Copper mineralization has been found that S.Kalaidjiev relates to the porphyry copper specie. The original information on Ellatzite deposit appears as a result of geologic mapping of a scale 1:25 000, in 1954. Geological prospecting work commenced in 1959 with mapping and revising work, continued with mining and drilling activity in 1960, so detailed exploration has been completed in 1968 when the then Committee of Geology approve Ellatzite deposit reserves in a record No.839, dated 19.12.1968. Basic stock of fundamental material for Ellatzite deposit was completed and summarized to these two reports: in 1968 (Hadgijski & Team); in 1980 (Georgiev & Team).

2 GEOLOGIC TEXTURE

2.1 Geology and Structure Description

Conceptions for Ellatzite deposit geologic texture, also the role of structural factor for ore mineralization go through evolution. According to Professor Zh. Ivanov the structural evolution of the deposit is a product of deformations within the range of right-hand strike-slip system of crustal shear of regional nature. The system has worked for a long time and have controlled not only Upper Cretaceous hypo abyssal to sub volcanic magmatic bodies but also distribution of late magmatic and/or hydrothermal solutions and mineralization processes, being related to these. Upper Cretaceous mineable ore mineralization of Ellatzite deposit is porphyry disseminated, hosted in magmatic and metamorphic rocks of different age – Paleozoic and Late Mesozoic; related to Paleozoic rocks are contact altered to hornfels green schist metamorphites of "diabasephyllitoid complex" and Vejen Pluto granodiorites, as well. Mesozoic (Late Cretatious Age) magmatic bodies are represented by monzosyenitic and quartzdioritic porphyrites

2.2 Morphology of the Ore Body

Ellatzite deposit ore body is one of stock work type without sharp geological boundary lines. Its boundary lines are defined on basis of chemical analyses of the major mineable component - the copper. It is considered as ore body that part of mineralized host rocks, where copper grade concentration comes up to 0.18%. It has the shape of distorted pillar-like ellipse with horizontal sections. These have a long axis of max 1400m and short axis max 850m. The industrial importance of Ellatzite deposit ores is determined by grades of three major components - copper, gold and silver. Besides above mentioned components extracted by metallurgical ore processing, also, the ore contains many other valuable components that could be considered to be "potentially" mineable. These are PGE (platinum group elements), molybdenum, and rhenium. The non-metallic minerals of Ellatzite deposit are presented by quartz, potassium feldspar, hydrobiotite, carbonates, zeolites, barite, chrorite and many others.

3 GEOLOGICAL AND EXPLORATIN ACTIVITY

The geological exploration activity is divided into two stages:

1st stage: detailed exploration during the period of 1959-1968

2nd stage: operation exploration after 1977 up to now.

Methods of exploration of the deposit are in conformity with the copper-porphyry nature of mineralization and potentialities for surface mining.

Detailed exploration has been completed principally by vertical core drilling; drill holes arranged in drilling pattern 71m x 71m in the central part or 100m x 100m drilling pattern in the periphery. On the basis of drilled holes Ellatzite deposit has been completely explored in area, also in depth, up to an elevation 805m. For testing of the drilling record from sampling, there have been cut exploration workings. These have been used to take technological samples, too. The drill core in mineralized sections has been continuously sampled by taking sectional (single) samples.

Besides single samples, group samples have been constituted over 15m sections corresponding to exploitation levels. The single samples have been assayed for copper and sulphur, group samples for copper, sulphur and molybdenum, a portion of them for gold, silver and selenium.

In general, in the course of detailed exploration there have been carried out significant specific activities and geological exploration work to the extent, as follows:

- Drilling : 74 021 m drilled
- Mine exploration workings : 4 115 m drilled
- Number of core samples taken from drill holes : 26 546
- Number of trench samples, workings : 3 431
- Number of group (batch) samples : 3 274
- Number of mono mineral samples : 140
- Technological samples, number : 18

As a result of the detailed exploration, reserves have been estimated after the geologic block method of then existing B, C_1 and C_2 categories. Total reserves accepted to be a basis for designing and development of Ellatzite Mine, average copper grades over 0, 18% come up to 407 823,5 Kt, with 1 544 911 t metal and average copper grade 0,379%. First reserve re-estimation was made as to status 01.02.1980, when after a new block construction an index was introduced to reduce the average copper grade in periphery blocks.

Operational exploration of the developed deposit part is carried out by regularly sampling of blast drill holes. Density of sampled drill holes varies between 5 – 15 meters depending on copper grade of sampling section.

4 ORE RESERVES AND MINING OPERATION

Mining & engineering conditions and potentiality for using opencast mining method to exploit Ellatzite deposit for the first time have been well-grounded simultaneously from both institutes NIPRORUDA/ Sofia and GIPROTSVETMET/ Moscow, in 1969 and in 1970.

The projects have been developed on the basis of approved by the then State Committee of Reserves: 212 972 Kt proven reserves, 0,387% copper grade, 824 569 t Cu metal, as to state 01.07.1968. Preliminary mine design of GIPROTSVETMET, Moscow has been approved by Order No.176 dated 04.09.1974 of Counsel of Ministers; thereby Ellatzite open pit technical contour has been approved. Above mentioned Order is the approval of following basic technical parameters to affect directly the ore reserves:

- Annual ore extraction : 10 000 Kt
- Annual copper concentrate based on 20% copper : 150 480 t

During 1976-1977, KNIPPI "NIPRORUDA" has undertaken an optimization of the technical pit outline. In April 1980 came out a new "Report on Reserve Estimation of Ellatzite Copper-porphyry Deposit, as to state 01.02.1980". This report shows proven reserves as approved reserves amounting to 247 406, 6 Kt ore, copper grade 0,375% and 927 102 t of metal. Here the increase is a consequence from re-estimation of sulfide and mixed probable reserves to proven reserves and enlargement of the pit in depth by one more benches (where the pit floor lays at an elevation 910). The late proven reserves are in conformity with last optimized contour of the pit approved in 1979 from the then Ministry of Metallurgy & Mineral Resources.

5 RUN-OF-MINE, OVERBURDEN AND ORE EXTCTION

As it was mentioned above, the first projects for opencast mining of Ellatzite deposit have been designed by KNIPPI "Niproruda" and "Giprotzvetmed" – Russia. The plans for subsequent reconstructions and modernizations of production capacities during the time period after 1992 have been designed by "GeoTechmin – Consult & Engineering"OOD, Sofia in cooperation with mining specialist who were exploiting the deposit.

The mining method is the classic drilling and blasting technology using up-to-date mining equipment; drilling and blasting of 15 m benches / own production cartridges & emulsion explosives. Broken ROM is loaded by Hydraulic or electric excavators /11 m^3 -

15 m^3 buckets on 80t - 130t dump trucks. Starting September 2003, an automatic Sky Links GPS control is in operation to monitor and control dump trucks. The Canadian Opteh ILRIS-3D laser scanner, with the 1200 m range is the pride of mining survey department and it is most powerful in the world, also irreplaceable when working under complicated mining and technical conditions or inaccessible mine sites.

Extracted ore is dumped into bunkers of primary crushing plants. Crushed ore goes by underground rubber conveyors into Mirkovo Processing Plant that is located on the south flanks of Stara Planina Mountain. The tunnel, passed by the rubber conveyor is about 6.5 km long. It crosses the main Stara Planina Range, and thereby is unique equipment.

Construction of ore mining complex started in 1976. At the very beginning of operation the highest point of the pit was at an altitude 1 562m that was already mined. Mining activities cover 520 m now – from an elevation 1510 to elevation 985. The first 5.3 Mt ore have been mined by the end of 1981. Run-of-mine extraction in that same year was 8.2 Mt, figures increase by every year. From the beginning of exploitation till 01.01.2009. 613 Mt ROM have been yielded, more then 268 Mt ore with copper grade 0.386% and 1 035 116 t metal (Cu). In 2008 was achieved a record in Ellatzite Mine's history - 56 Mt ROM extracted, 13 Mt ore thereof. Thus, Ellatzite Mine turns into the biggest open pit mine in Bulgaria and in the Balkan Peninsula.

By virtue of the Law of Ore and Minerals, the ore and minerals are exclusive state property. Extraction of ore is possible after received granted concession rights, i.e. buy off the necessary license rights permitting the extraction of ore and minerals under defined terms for a fixed period of time. By Resolution No. 594/02.09.1999 of Council of Ministers, Ellatzite Med EAD is granted a concession, subject to special rights to use ores and minerals – copper-porphyry auriferous ores. Concession rights are for 17 years time period.

On 02.10.2006 was signed a supplementary agreement No. 2 to "Contract for Granting a Concession for ores and minerals – copper-porphyry auriferous ores, by extraction from Ellatzite deposit, Sofia district, signed on 15.11.1999.

The supplementary agreement term of granted concession for extraction of copper-porphyry auriferous ores from Ellatzite deposit was amended by extension of 5 years towards the original term.

Ellatzite Ore Mining Complex, town of Etropole is one of both major business branches of Ellatzite Med AD private stockholding company – leader of Bulgarian mining industry, one of the biggest concessioners, investors and tax payers in Bulgaria. This is the biggest company in the group GEOTECHMIN OOD, Sofia. Ellatzite copper and gold open pit mine is one of the biggest in Bulgaria and Europe with annual output more than 56 Mt ROM, 13 Mt ore, inclusive; 92 % metal recovery allows more than 42,000t copper production, more than 1.6 t gold and about 5.5 t silver. Modern up-to-date mining equipment from world class leading manufacturer is in operation, like HITACHI, BELAZ, KOMATSU and many others. Own explosive production plant built under American license was erected to manufacture emulsion explosives and cartridges, equipment for production of wide range of inert materials.

Reserves of the deposit assure profitable extraction at least until year 2022.

6 THIRD ENLARGEMENT OF ELLATZITE MINE

After all said, according to the report of 1980, ore reserves of Ellatzite ore body would be depleted. And yet, during recent 4-5 years the Company's strategy and all the efforts of ELLATZITE MED AD mining professionals and Geotechmin EOOD involve further development of mineral resources.

Design teams developed dozens of technical contours to search for increasing of reserves in the deposit, with minimum quantity of overburden, at optimized overall slope angles. In 2008 we brought to an end one of best technical contours, developed by GEOPROJECT EOOD, a subsidiary of GEOTECHMIN OOD, resulting in an ultimate slope angel of 48°, pit floor at 880m altitude, length 1900m and width 1400m. Above pit outline ensures ca.150 Mt proven reserves (Category 111) as of state 01.01.2009, there by increased life of the pit until 2022, thus to satisfy the last concession term completely.

Figure 1. General view of Ellatzite mine.

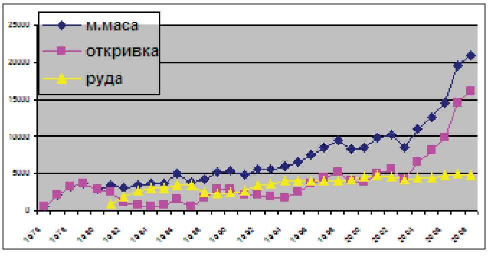

Figure 2

In fact, that is the third Ellatzite Mine expansion of large dimension where with its 150 Mio t proven reserves it offers a new open pit. With the new enlargement mining operation gets into depth and haulage distances increase. Under existing circumstances it became necessary to optimize transportation of the ore that has been realized by construction of our new in-pit primary crushing plant and transportation of crushed ore by rubber conveyor belt through a tunnel. Erection of the new conveyance system shortened the average haulage distances by some 1.5 km, thereby to save from haulage activity almost 150 Mt/km.

The lowering of pit floor imposed new answers to pit drainage. For that purpose a new drainage tunnel has been constricted under the pit floor, at an elevation of 840m, 2100m long.

Configured that way Ellatzite Mine will have one of the longest and steepest slopes in the world. As example, the well-known Chuquikamata in Chile, South America has slopes of a length 645 m at 46° slope angle inclination; Palabora Mine in South Africa uses slope length 500 m, at 58° slope angle

REFERENCE

Tzatzov, V., Director Ore Extraction, Ellatzite Ore Mining Complex/Ellatzite Med AD

Characteristics of the Open Cast Mine "South Field"

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ABSTRACT Opening and building up of the new coal mines, construction of the new thermo blocks is a big investment project and represents an imperative and turn over in the development of Electric power industry of Serbia. The condition for the maintenance of the present level of coal production and capacity enlargement of the new thermo blocks is opening and building up open cast mines Veliki Crljeni, Field E, South Field and Radljevo. The future open cast mines will have considerably bigger and more complex problems for opening and building up in comparison to all the other open pit mines which have been opened in Kolubara open mines up to now. First, bed separation of coal seams series, great depth of occurrence, significant hydro geologic ground water, with the important infrastructure object, which occupy the area, are the only basic parameters which show the real problems of new open pit mines. Open pit mine South Field itself with all the problems it comprises, is the biggest challenge for the designers and the executors of work.

The limitation of open pit mine South Field has been carried out based on the consideration of the complete coal layer (geological fields E, F, G, Šopić, Veliki Crljeni and Tamnava East Field), and also based on the position of the important infrastructure objects and coal quantities required for the supplying of electric power plant. By analyzing of the front development on the overburden and coal, as well as stockpile capacity for waste in two variants of vertical division, it has been is decided to excavate only overlaying and coal groundmass, while the substratum is not going to be dug out. The main reason for defining of the open pit boundary depth in such a way is the lack of stockpile capacity for overburden dumping.

1 INTRODUCTION

The mining basin of Kolubara is the greatest producer of coal in the Republic of Serbia, and in the total coal production, its participation amounts 65%. From the total of electric power 42% is on the basis of the mining basin of Kolubara. In this mining basin three are there working open pits – field B, field D and Tamnava West field. Exploitation is finished on field A and Tamnava East field while its in plan opening open casts Big Crljeni, field E, South Field and Radljevo. Open casts in future will have considerably bigger and more complex problems for opening and development in comparison with all the other open pit mines, which was been opened so far. Certainly, open pit South Field represents the greatest challenge because its problem represents the most challenge for the designers and workers on that open pit.

The limitation of open pit has been carried based on the consideration of complete coal layer, and on the basis on the position of the important infrastructure objects and coal quantities needed for the supplying of electric power plant. In the central area of the future open mine pit three are a lot of rivers, such as the river Kolubara which is the main water course which runs south – north across the coal bed of South Field. On the left side of Vranicina and Skobaljski potok flow into the Kolubara, and on the right side river Pestan and Lukavica. Also the important roads (G. Milanovac–Beograd main road, railway Beograd–Bar) as and industrial inherent objects on coal layer disposed the contour of the future open pit.

The limitation of open pit is, beside it, conditioned by separation and wedge out of coal on south and west part, while there are barren, unusable part and open pit Tamnava East field on the north. Eastern ending slopes of the open pit South Field is determined by important industrial and infrastructure objects. It could be said that large amounts of coal between east boundary of the open pit South Field and defined boundaries of open pits field D and field E present reserves out of balance.

New open pit mine is limited:

- on north – by open pit Tamnava East field and unworkable part of coal bed,
- on east – by railway Beograd – Bar,
- on south – by separation and wedging of the coal layer,
- On west – by wedge out coal layer, increasing of coefficient of stripping ratio and the final position Kolubara River.

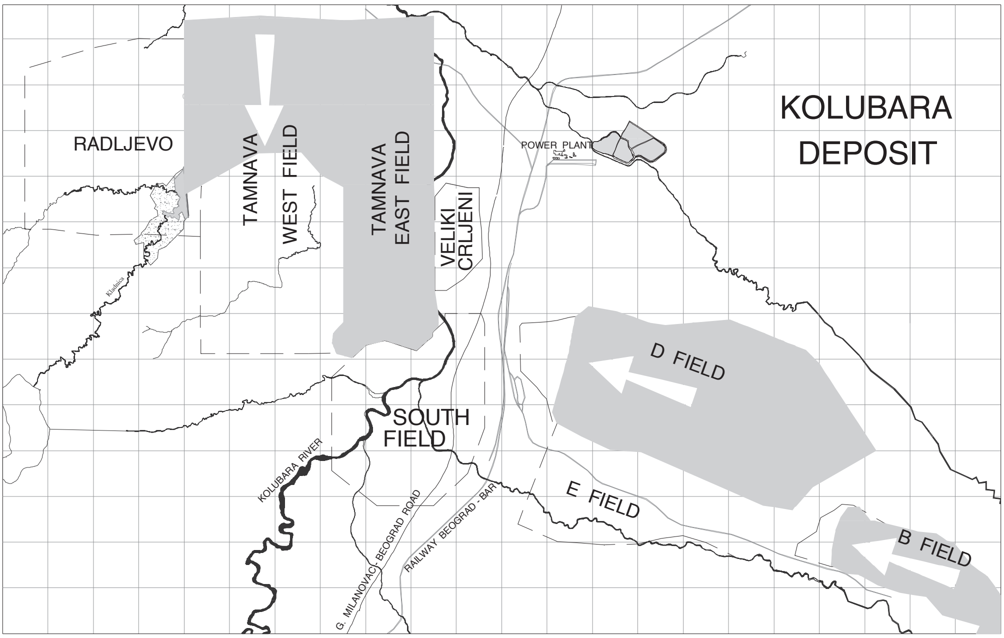

Figure 1. Working and planned open pit.

By realizing geomorphologic characteristics of the coal layer, we can conclude that the best solution is to open new coal pit South Field from south to north. Namely, the coal layer goes upward and wedges out to south, the advance from south to north in first year of exploitation with the smaller coefficient of stripping ratio and at the good of favorable angle of the slope of the opening cut, even the third coal layer can be reach.

However, considering everything, it is obvious; this solution does not have any advantages. Having in mind the chosen construction of the open cut mine, position of rivers Pestan, Lukavica and Kolubara, the amount waste and intra–layer waste and coal, as well as possible location for waste, possible ways for transporting waste and coal, opening open cast from south to north would create irrational solutions for the transport of waste and coal. In addition, opening open pit mine from south to north would cause very unfavorable dynamics of dislocating river, which would not coincide with the dynamics needed roar opening the open cast mine.

The chosen variant of opening open pit was by making the inside cut from north to south, with phase dislocation of the river and the mine road. After finishing exploitation of the waste in the future open pit Veliki Crljeni, bucket wheel excavator would advance for southwards and start opening the South field.

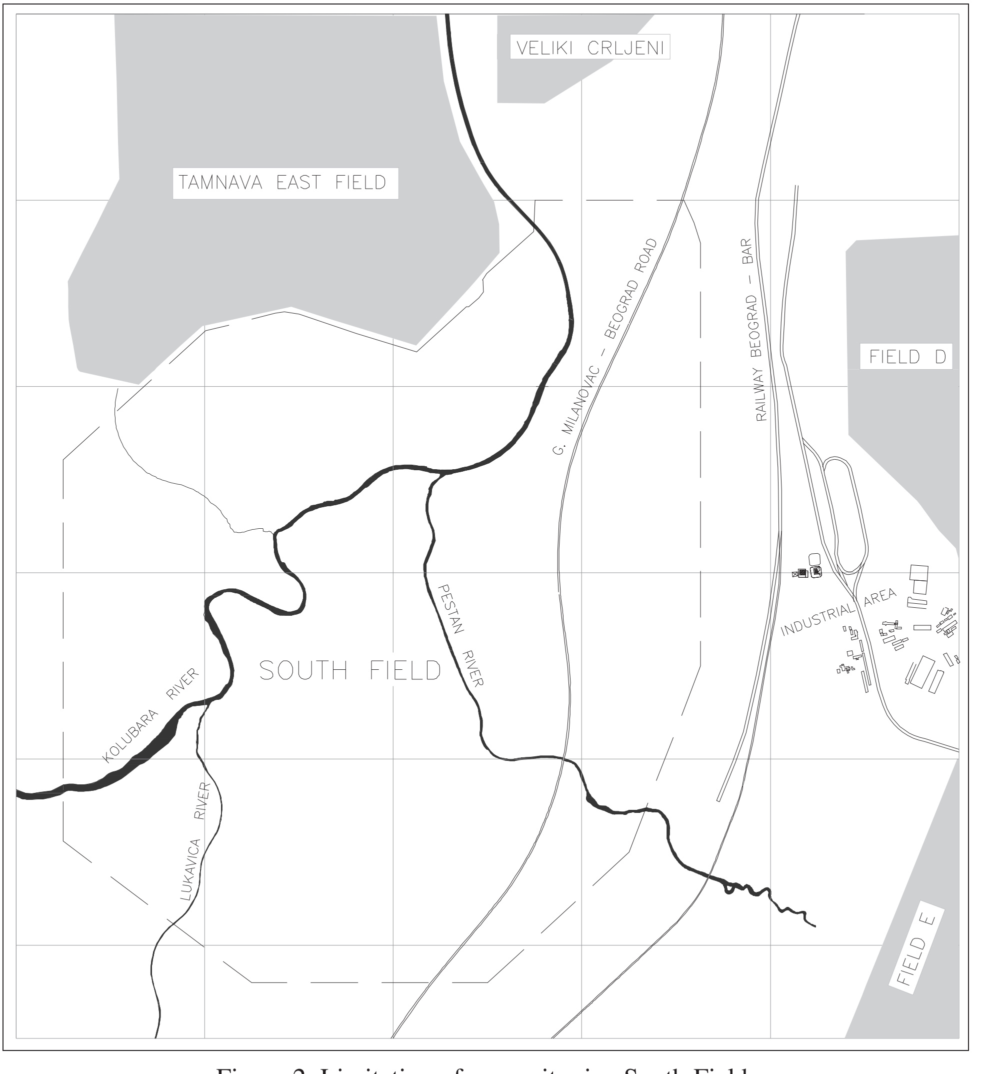

Figure 2. Limitation of open pit mine South Field.

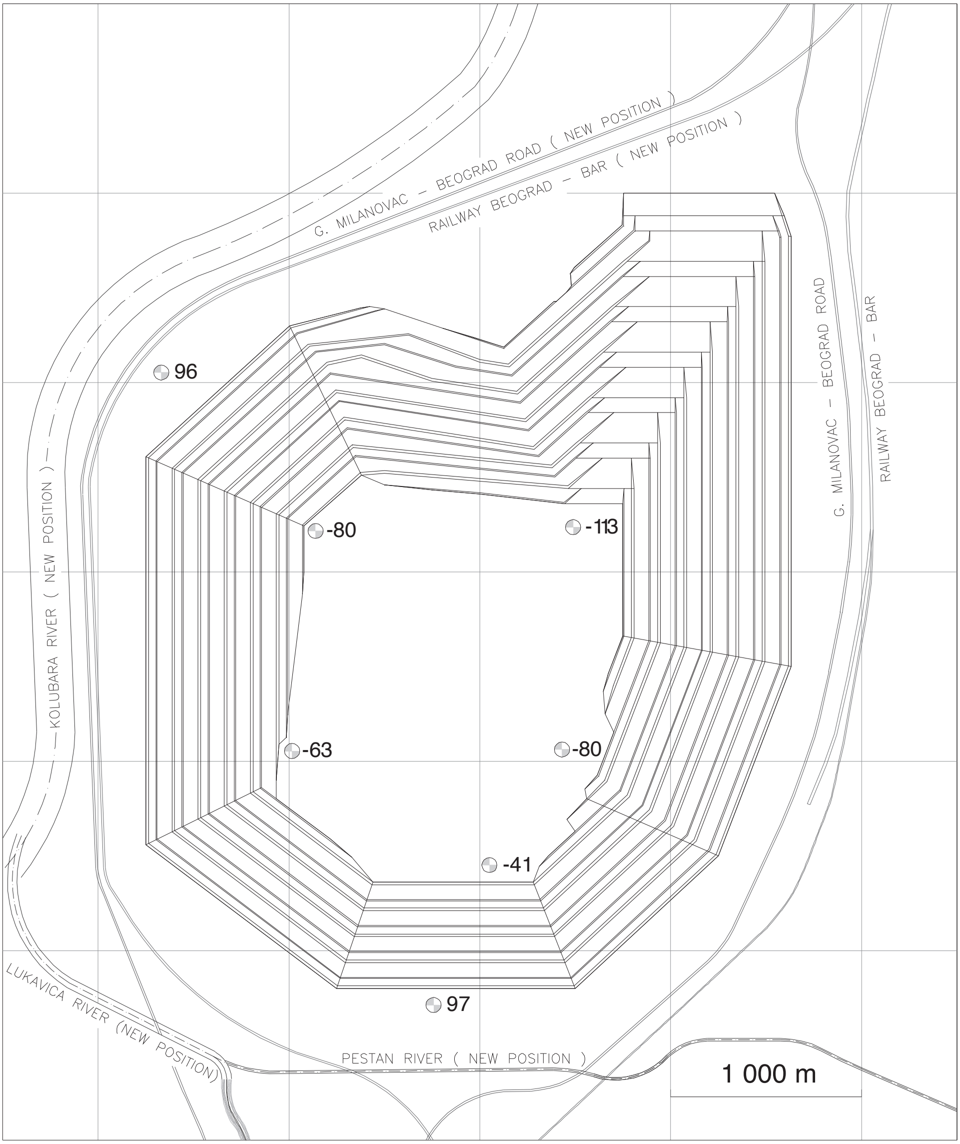

Figure 3. Overlaying and coal ground mass excavation.

All waterways must be dislocated outside the contour of the designed open pit and they must have optimal distance from the boundary of the open pit. Dislocated rivers have to be watertight, meaning that technical solutions must prevent infiltration of the water from river into the open pit.

Kolubara river bed dislocating is intended to be carried during two phases – in the first phase the length of the dislocated riverbed would be 2580 m and in the second phase, 6230 m. Riverbed Lukavica would be dislocated for 1860 m and riverbed of Pestan for 3300 m.

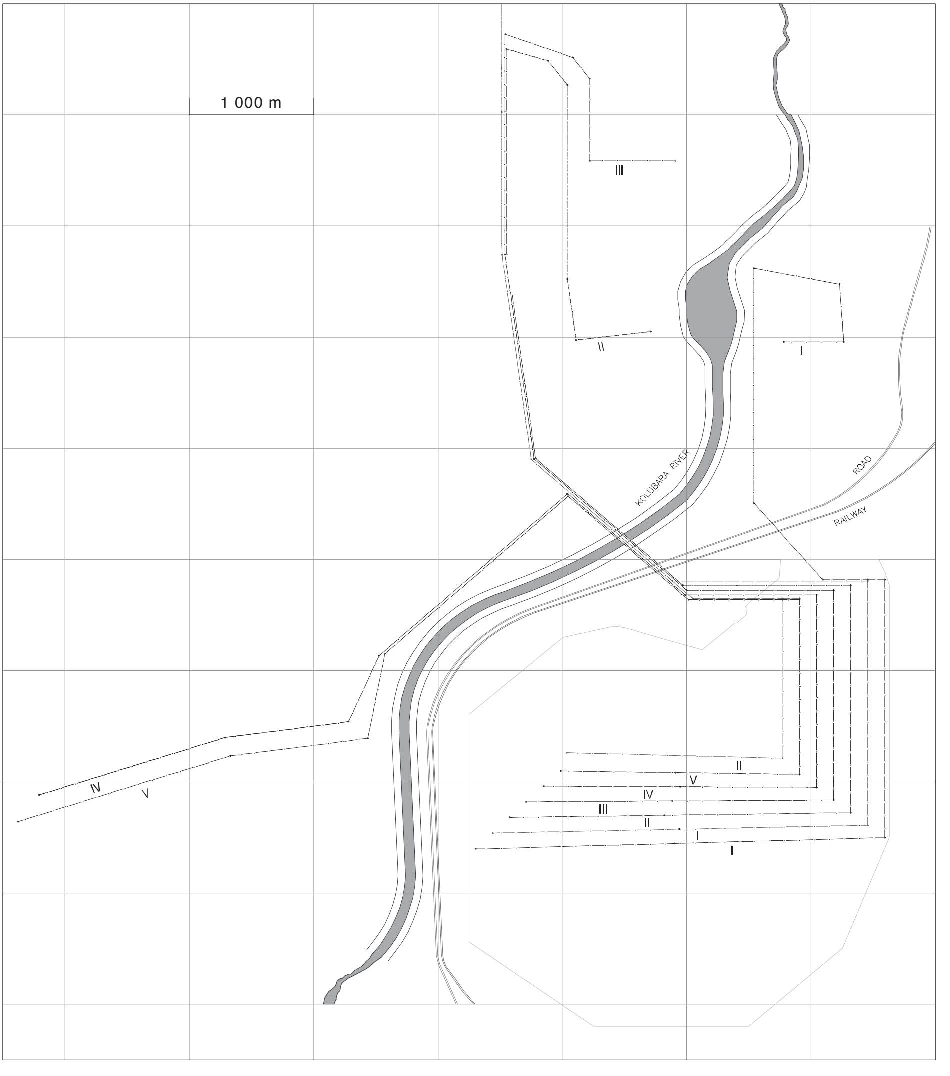

Figure 4. Transportation roads of waste and coal.

For the excavation open pit mine, we need five systems: excavator-conveyor–spreader on waste and two systems on coal bancs. In the first phase of building open pit mine putting away of waste is planned on three out overburden dumping until making are created the conditions for overburden dumping on inside.

Standard stripping ratio is 3.62 m^3/t, and the standard for coal production is 8.4 x 106 t for 35 years. Overtook amount of coal has the quality of coal of 7100 kJ/kg.

By dislocating the main road G. Milanovac – Beograd and railway Beograd – Bar would be the create conditions for digging large amount of coal on the east of the boundaries of the south field open pit, which would include even the coal stretching to open cast mine field E.

Alternative Technological Solutions in Underground Ore Mining

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ABSTRACT In generating multiple variants applicable for mining thick ore bodies it is appropriate to use the matrix recording. Based on that system it has been established that the mining technologies can be classified in two big groups: single-stage and multi-stage. The sublevel stopping mining methods are typical cases of the multi-stage technologies. Characteristic variants are presented which can be applied under conditions of stable and medium-stable rock masses. A graphic-analytical method has been developed for assessing the effectiveness. The analysis is carried out by using a three-dimensional coordinate system. It is proved that the coordinates of the optimal technological solution correspond to a point which is inscribed in a body with a definite stereometric characteristic. Its walls are planes or curvilinear surfaces constructed based on the limiting conditions of parameters with natural and value measurements.

1 INTRODUCTION

The reconstruction and modernization of the existing underground ore mines is a topical issue in modern mine design technology. There are numerous alternatives along the technological chain: opening-up, development and mining operations. Their systematization is a first step toward constructing a general model which can "cover" the whole combination of natural, mining, economic, social and environmental factors. The dynamics in the variation of indices characterizing the interaction of the mining technology with the real natural environment shows that without making use of the principles of the systems approach it is impossible to solve such problems. Not less important is the problem of defining appropriate criteria for assessing the technological solutions. In terms of the general formulation of their systematization as metric, time and value parameters, finding the n-cube involving the optimal mining technology is a very specific task. Its successful solution depends on the level of mechanization and market and socio-political situation in the country.

2 ALTERNATIVE TECHNOLOGICAL SOLUTIONS IN OPENING UP THE OREFIELD

2.1 Selecting the Location of the Main Opening

When opening up a group of stocks within an ore field, the location of the main opening is determined according to the relative weight of the load flows and haulage operations on the crosscut levels. A gathering point is set up whose location depends on the interaction of the mining operations in each ore body (stock). The task is set in a 3D space, the z-axis coinciding with the vertical direction and axes x and y oriented in the horizontal plane. An expression is used which combines the haulage operations of hoisting equipment W_d

in the underground workings W_t, and transportation on the surface W_s. Obviously, W_t=f(x,y); W_d=f(z); W_s=f(x,y,z). Then the functional U_{al} for finding the total operation will have the following form (Zlobin et al 2007):

$U_{al}=W_t+W_d+W_s$, i.e.
$U_{al}=F(x,y,z)$. (1)

The assessment of the set of alternatives defined by the factor "selecting the location of the main opening" has a static character since the quantity of ore reserves and the coordinates of transfer points on the crosscut levels are constant values.

2.2 Selecting the Opening-up Method

When developing the mining operations in depth, the alternative opening-up methods are reduced to two types: vertical shafts and inclined shafts. Practice has shown that under modern conditions the ratio of using these two types of openings varies considerably. If we assume that the limit depth of opening by inclined shafts is H_1 = 400 m, and the depth at which the vertical shaft opening method has no alternative H_2 = 800 m, then apparently a range exists within which the selection of the opening-up method should be based on a detailed feasibility study. We know from practice that the hoisting equipment now being used is capable of lifting loads from a depth H_3 = 1200-1300 m. Therefore, H_3 characterizes the limit of introducing multi-stage opening-up methods.

The determination of the opening-up method on the basis of defining alternative variants with vertical shafts and inclined shafts is based on the functional

$$U = F(A,Z,H) \qquad (2)$$

where A is the annual mine output, Mt/an; Z - quantity of opened-up reserves, Mt; H - depth of opening, m.

Practice has proved that the functional (2) has a clearly expressed extremum which is an advantage when using this approach in the feasibility study.

2.3 An Optimal Opening-up Step; Assessment Criteria

High-output mines are characterized by using underground dump trucks and conveyor belts. The capacities of these vehicles to move along inclined openings as well as the introduction of the first stage of ore crushing underground resulted in basically different approaches to determining the optimal opening-up step. The dynamics in changing the metal prices considerably increased the risk factor thus forcing mining companies to avoid radical opening-up variants. From this point of view it has become particularly relevant to determine the optimal opening-up step aiming at a non-conflict connection with the existing infrastructure.

Figure 1 shows a 3D opening-up method when the transfer points $Q_{1\alpha}$, $Q_{2\alpha}$ and $Q_{3\alpha}$, lie in plane α, and the gathering point P is located at a certain distance above plane α. Figure 2 shows a 3D scheme when load flows from transfer points are formed which are located in two planes $Q_{1\alpha}$, $Q_{2\alpha}$ and $Q_{3\alpha}$; $Q_{1\beta}$, $Q_{2\beta}$ and $Q_{3\beta}$.

The gathering point P is found between the two planes. Figure 3 shows a 3D scheme when the transfer points are located in three planes α, β, γ. A combination of three types of openings: vertical shaft, inclined shaft and ramp, is used between the separate planes. The formulation of the problem is based on finding the minimum distance between the three points using an original solution to Ferma's problem for that purpose (Mihaylov & Georgiev 2007).

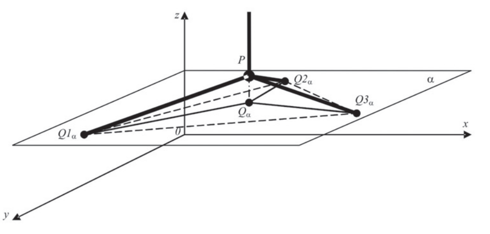

Figure 1. Gathering point lies at a given distance above plane α.

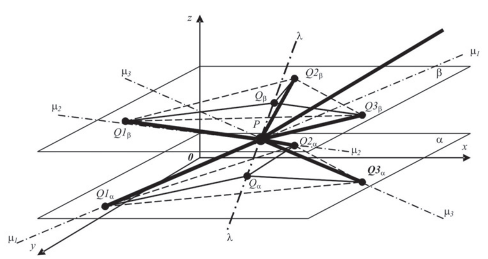

Figure 2. Gathering point lies between two horizons.

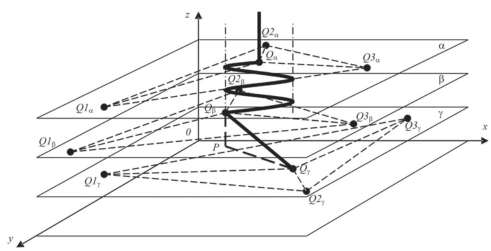

Figure 3. Combination of mine workings between three horizons consists blind incline shaft, ramp and shaft.

The necessary investments for the implementation of a reconstruction project include the following strategic goals:

- change in the mine output;
- reconstruction of the opening-up method including the surface mine complex;
- changing over to another type of mining technology;

These goals are realized in a long-term perspective thus making the time factor and the period for assessing the economic efficiency of investment projects indicators of primary importance. The problem formulated in this way presupposes the use of the following expression for the profitability assessment:

$$\Pi_j(\tau)=\sum_{t=P_j}^{\tau}\left[\left(Ц_{tj}-C_{tj}\right)\Delta_{tj}-e_2K_{tj}^{//}\right]\frac{(1+q)^{\tau-t}}{(1+e_н)^{t-1}}-e_1\sum_{t=1}^{P_j}K_{tj}^{/}(1+q)^{P_j-t}$$

(3)

where t is year of feasibility assessment, $t=\overline{1,\tau}$;

τ – assessment period; generally corresponds to the variant with longest duration of reconstruction;

$Ц_{tj}$ – selling price of 1 t of mineral during the tth year of the jth variant;

C_{tj} – production costs for extracted mineral during the tth year of the jth variant;

$K_{tj}^{/}$ – investments spent during the reconstruction stage during the tth year of the jth variant;

$K_{tj}^{//}$ – investments spent after introducing the reconstruction stage during the tth year of the jth variant;

e_1, e_2 – coefficients determining the costs for redeeming the loans during the construction and reconstruction stage, respectively;

q – indicator determining the costs for redeeming loan interests;

$e_н$ – coefficient for discounting future profits;

Δ_{tj} – quantity of extracted mineral during the tth year of the jth variant;

P_j – year of putting the reconstruction into operation;

Finding an optimal variant of reconstruction and modernization is in itself a complex problem. Therefore, besides expression (3), criteria are used which characterize the flow capacity: load matching coefficient $К_c$, loaded capacity coefficient $К_н$, and indicator accounting for the level of system organization Y_c.

The following parameters are to be used for assessing the effectiveness of the load flows formed:

- possible flow capacity, $Q_в$;
- effective flow capacity, Q_e;
- flow capacity of the reserve capacity (if any), Q_p;
- 24-hour mine output, $A_д$;

The parameters given have a natural dimension, i.e. they depend mainly on the technical capacities of the units (machines) and the adopted work organization. A necessary condition for the results to be representative is that each transport chain should have a minimum of three links. Therefore, if *i* is the number of the link, then $i=\overline{1,N}$, where N_{min}=3.

The matching coefficient $К_c$ is determinate according to the expression:

$К_c = Q_e/Q_в$, $К_c \le 1$

The loaded capacity coefficient $K_н$ is determined according to the expression:

$K_н = Q_в / A_д ; \quad K_н \geq 1$.

When studying the transport chain consisting of *N* links, we can find link *i* for which the loaded capacity coefficient takes a minimum value, i.e.

$K_{н_i} \to min$

This link corresponds to the bottleneck in the technological chain. Therefore, by using another variant or replacing the units in the existing transport chain variant it is possible to change the value of $K_{н_i}$. This means that for every variation in $K_{н_i}$ an array of values for K_c will be obtained corresponding to the number of links in the technological chain. It is necessary to introduce.

The indicator accounting for the level of organization of the system Y_c can be determined by the expression:

$$Y_c = \frac{\prod_{i=1}^{N} K_{c_i}}{\frac{1}{N}\sum_{i=1}^{N-1}\sum_{j=i+1}^{N} K_{c_i} K_{c_j}}$$
(4)

where N is the number of links constructing the transport chain.

If we consider a total of M variants, we will obtain an array of M values of $Y_{c_k}, k = \overline{1,M}$. Then $Y_{c_k} \to max$, will correspond to the variant with the highest flow capacity, i.e. it should be assumed as the optimal variant.

3 ALTERNATIVE TECHNOLOGICAL SOLUTIONS IN DEVELOPING THE OREFIELD

The alternatives in the stope development are determined by the spatial position of the three links: loading-delivery-drawing. The loading depends on the type of mechanization used and the delivery on the elements of orebody occurrence. Therefore, here we place an emphasis mainly on the method of drawing. Three ore drawing methods exist: draw point, draw cone and undercut draw each determining a different structure of the stope bottom. The draw point has the advantage of permitting control of the contact between the ore and caved rock by the draw doses. The draw cone has the advantage of decreasing the volume of development openings but does not permit effective control of losses and dilution. The undercut draw provides the best conditions for displacing the contact ore-caved rock but is accompanied by problems of geomechanical nature which, in turn, decreases the level of technical safety during operation.

When working thick ore bodies, very often the question of applying multi-stage technologies arises. According to the classifications of mining methods known until now, these technologies can be referred to the combined methods. In this particular case the number of stages of mining a given ore deposit/ore body proves to be a suitable classification feature for generalizing the set of mining technologies. Apparently, each stage of mining has its own specificity in the structure and location of the development openings.

The first question that arises is to determine the number of stages *s* for the ore body development and the relative portion of each stage D_r, observing the condition: $\sum_{r=1}^{s} D_r = 1$.

As we have already mentioned, one indicator on which the conception of the effectiveness of the mining technology is built is ore dilution. The average value of dilution b_{cp} when using multi-stage mining will be:

$$b_{cp} = \sum_{r=1}^{s} D_r b_r ; \qquad \sum_{r=1}^{s} D_r = 1 . \qquad (5)$$

Taking into account the specificity of development openings for each mining stage, the average value of dilution $\overline{b_{cp}}$ will take the following expression:

$$\overline{b_{cp}} = \sum_{r=1}^{s} D_r \left[\left(1 - K_{доб_r}\right) b_{o_r} + K_{доб_r} b_r \right]$$
(6)

Then, the production costs $C_{П2}$ for the extracted ore for the whole stope when using the multi-stage mining technology will be:

$$C_{П2} = C_{бл} + Щ_b = C_{доб} + C_{ПНР} + Щ_b. \qquad (7)$$

If an *s* number of mining technologies with a relative portion D_r $\left(\sum_{r=1}^{s} D_r = 1\right)$ is used, then the expression for $C_{П2}$ will finally take the form:

$$C_{П2} = \sum_{r=1}^{s} D_r \left(C_{доб_r} + C_{ПНР_r}\right) + Щ_b \qquad (8)$$

In this case $Щ_b = \psi_1\left(\overline{b_{cp}}\right)$. The character of this function implies that we must introduce the ore value, the valuable ore constituents, respectively, and the prices on whose basis the end product is sold (in this case the concentrate). The multi-stage mining presupposes the use of technologies with different losses *a*. By analogy with dilution, the average value of losses a_{cp} and the average value of losses with accounting for the portion of development openings $\overline{a_{cp}}$ can be determined.

$$a_{cp} = \sum_{r=1}^{s} D_r a_r, \text{ if } \sum_{r=1}^{s} D_r = 1.$$

(9)

$$\overline{a_{cp}} = \sum_{r=1}^{s} D_r \left[\left(1 - K_{доб_r}\right) a_{0_r} + K_{доб_r} a_r\right]. \qquad (10)$$

Then, the total costs $C_{П3}$ for the extracted ore for the whole stope (ore body) with taking into account the damages caused by losses and dilution in multi-stage mining will be:

$$C_{П3} = C_{бл} + Щ_a + Щ_b = C_{доб} + C_{ПНР} + \\ = + Щ_a + Щ_b \qquad (11)$$

$$C_{П3} = \sum_{r=1}^{s} D_r \left(C_{доб_r} + C_{ПНР_r}\right) + Щ_a + Щ_b$$

(12)

In this case $Щ_a = \psi_2\left(\overline{a_{cp}}\right)$. The functions $\psi_1\left(\overline{b_{cp}}\right)$ и $\psi_2\left(\overline{a_{cp}}\right)$ have a considerable effect on the accuracy of the solution obtained. The theory and practice of studying the effect of the damages caused by losses and dilution show that in order to derive functions $\psi_1\left(\overline{b_{cp}}\right)$ and $\psi_2\left(\overline{a_{cp}}\right)$ as a basic variant, we use function: $b = \psi_3(a)$. It is often represented as a linear function. The concrete analysis with taking into the sources of losses and dilution and the resulting damages should prove whether the linear approximation is reliable enough when considering the site in question.

4 ALTERNATIVE TECHNOLOGICAL SOLUTIONS IN SELECTING THE MINING TECHNOLOGY

The approach to designing mining technologies based on the mining stages is grounded on three characteristics:

- number of mining stages;
- rock strata control method;
- method of supporting the face working area;

One condition on which the multivariant approach is built is the relationship between factors of different gradation: binary-alternative, on the one hand, and discrete-continuous, on the other. Within this framework, it is reasonable to use a matrix record. For single-stage and two-stage mining technologies it will take the following form: $\|DT\|_{ij}$, where $i = \overline{1,3}$; $j = \overline{1,3}$. The digits 1-3 correspond to the rock strata control method: 1 – pillar support (open stope); 2 – waste filling; 3 – caving. The matrix members which are on the main diagonal, i.e. i=j, identify a single-stage mining technology. The combination (ij) determines the type of technology used during the first and second stage. Apparently, the matrix $\|DT\|_{ij}$ is not symmetrical, i.e. it is a kind of a generator of possible mining technologies in single-stage and multi-stage mining: a total of 9, of which 3 single-stage and 6 two-stage. In this generation of variants combinations are formed which seem to be difficult to apply. For example, DT_{31}– caving during the first stage and open stopping during the second one; DT_{32} – caving and waste filling.

When having three or more stages, the matrix record is not appropriate and then the symbols of logical algebra are use, in particular, the logical function conjunction "$\wedge$" (Ehrenberger & Fajkos 1990). It is used to identify every mining technology involving three or more mining stages. In this case the automatic generation of a set of variants is not expedient. It is formed manually proceeding from the specificity of the natural conditions and mining characteristics of the site.

Figure 4 shows the classification of mining technologies according to the "mining stages" factor. The correspondence to existing classifications is pointed out. It can be noticed that some of the well-known mining methods such as shrinkage stopping, top slicing, stall-and-breast stopping have not been mentioned in the classification. For the first two methods we can find explanation in the modern tendency of decreasing their relative portion. The low level of technical and economic indices achieved makes them uncompetitive versus the alternative mining technologies (open stopping or waste filling). The stall-and-breast method is a typical two-stage mining technology of the DT_{12} or DT_{13} type. The presence of alternatives is convenient to represent by the logical function disjunction "$\vee$", i.e. $DT_{12} \vee DT_{13}$.

When mining bedded deposits the room-and-pillar method is widely used. If the supporting pillars are not extracted, it is a typical single-stage technology of the DT_{11}. If the pillars are extracted, then it becomes a multi-stage mining technology. The following alternatives are possible:

- back filling and pillar extraction as open stopping;
- back filling and pillar extraction by the waste filling technology;
- back filling and pillar extraction by caving;

5 CRITERIA FOR ASSESSMENT AND LIMITING CONDITIONS

Creating a mining technology model based on a generated set of variants requires that limiting conditions be defined. On the one hand, a procedure is available which guarantees that solution sought is found in a certain closed contour. The calculation process is impossible without imposing objectively existing limiting conditions. They should have a complex character in order to ensure the formation of the so-called set of admissible variants as an object of further detailed analysis. This approach provides an answer to the question of the need for constructing a mining technology model taking into account the large variety of constituting elements. The following groups of limiting conditions are formed on the axes of the spatial coordinate system:

- axis of natural (mining and geological) limiting conditions;
- axis of the economic limiting conditions;
- axis of the organizational limiting conditions;

In this way a parallelepiped is formed in space which is a limit of the volume with the optimal solution sought. Thus, the necessary combination is achieved between the objective and subjective factors presented as natural and economic, on the one hand, and organizational, on the other. The advantage of this approach in comparison with the use of a complex criterion in the *n-dimensional* Euclidean space is that, the subjective factor is reduced to a minimum: practically it is not necessary to use either a set of criteria or an expert evaluation for determining their weight in the overall procedure. Figure 5 shows a diagram of the parallelepiped of limiting conditions. The axis of natural limiting conditions includes the variation in the ore body thickness M as a factor for the applicability of a given technology. Here we can use other factors as well: stability of ore and wallrock, hydraulic radius determining the stable dimensions of the open spaces, angle of slope and depth of mining (character of the stressed state of the undisturbed rock mass).

The axis of economic limiting conditions includes the Net Present Value as the most frequently used assessment method in referring the profits and investment costs to a given moment of time, usually to the moment of starting the investment process.

Single-stage mining technologies

Name	Symbol	Correspondence to existing classifications
1. Open stoping	DT_{11}	I.A. Pillar stoping I.B. Sublevel open stoping
2. Waste filling	DT_{22}	II.A. Waste filling II.A.1. Slicing II.A.1.1. Horizontal slices II.A.1.2. Inclined slices II.A.2. Sublevel slicing
3. Wallrock caving	DT_{33}	III.A.Sublevel caving III.B. Spontaneous caving

Two-stage mining technologies

Stage I	Stage II	Symbol	Correspondence to existing classifications
1. Open stoping	Waste filling	DT_{12}	IV.A. Combined pillar stoping and waste filling
2. Open stoping	Caving	DT_{13}	IV.B. Combined pillar open stoping I. C. Induced caving IV.A. Stoping and caving
3. Waste filling	Open stoping	DT_{21}	IV.C. Combined stoping
4. Waste filling	Caving	DT_{23}	IV.D. Combined stoping
5. Caving	Open stoping	DT_{31}	No correspondence
6. Caving	Waste filling	DT_{32}	No correspondence

Figure 4. Classifications of mining technologies according to the "mining stage" factor.

Depending on the particular procedure for selecting the mining technology, it is possible to use other economic categories. However, each category will be based on the structure of production costs *R(q)* mentioned above. The axis of organizational limiting conditions reflects the selected order of development of the mining operations within the production section. On the basis of the mining technology used and the schedule drawn up, the time for mining the reserves will vary within very wide limits (for comparison, we can use waste filling or stope caving technologies). The indicator Internal Rate of Return (IRR) can be used as a suitable limiting condition when accounting for the change in the quality of extracted ore, metal prices on the World Metal Exchanges and the profitability of the investment projects. This indicator equalizes the algebraic sum of the discounted positive and negative cash flows.

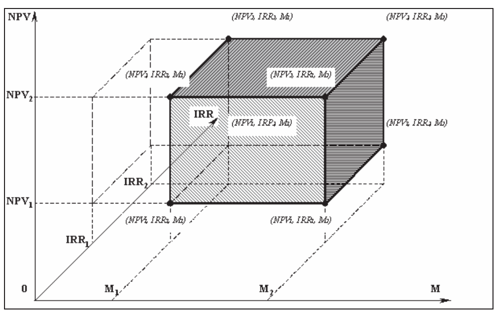

Figure 5. Defining limiting conditions for determining the optimal solution.

The determination of IRR is of special significance for the general analysis of the applicable mining technologies. Based on the set schedule for mining a given section as well as the anticipated rates in changing the prices on the World Metal Exchanges we can evaluate another organizational factor, namely, the adaptability of the mining technology to a possible change in its structural elements. At present the problem is reduced to determining the volume of the parallelepiped of limiting conditions. The equations of its walls are derived by linear interpolations of the functional relationships which inevitably exist between the quantities plotted on the three axes of the spatial coordinate system. The further development of this procedure is reduced to analyzing the ratio between the individual sides of the parallelepiped (analysis and assessment of the shape, volume size, spatial orientation, etc.). A relevant continuation is the use of polynomials of a higher degree for deriving suitable analytical expressions for spatial bodies as an alternative of the parallelepiped.

6 CONCLUSION

The numerous alternatives of mining technological solutions can be effectively used only on the basis of the systems approach. The construction of a mining technology model is based on the three-stage procedure including the ore field opening-up, development and mining operations.

The structure of limiting conditions developed enables us to find optimal solutions in both designing new mines and reconstruction of existing production capacities.

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Prospects for Development of Geo-Technological Production of Copper In Situ in Bulgaria

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ABSTRACT The paper contains the main information about the copper deposits in Bulgaria. A principal scheme about the technology and the mining method for undergrount leaching, production of productive solution and hydro-metallurgical production of copper in situ. The ecological problems are minimized and is ensured total protection of the environment. Two projects are presented.

An innovative strategy was prepared about the developments up to the year 2040-50 and further into the future.

1 PRINCIPAL TREATMENT

The physical-chemical geo-technology is in principal the new and leading trend in mining science during the 21st century.

Geo-technology as a science and practice has been developing systematically during the second half of the 20th century and it really represents a contemporary scientific revolution in mining sciences and the industry in its development during the 21st century (Arens, 2001).

The essence of geo-technological production (GTP) of metals from ore deposits and from non-balance raw materials could be described as (Danov, V., 2006):

(1) leaching technology (chemical dissolution) of the metals in place (in situ) in the deposit, production of metal solutions and

(2) with subsequent ion-exchange (hydro-metallurgy) processing of the productive solutions to marketable metals and salts of metals in situ.

During the period 1960-70 up until now (2009) this high technology was developed very quickly, around the World, mainly for the production of uranium and copper and some accompanying metals.

GTP of copper began in 1950-60 in the US and Canada by leaching of non-commercial waste dumps and heap piles with ore which was difficult to enrich. The methods for underground leaching of shrinked ore were developed gradually as well as the methods of leaching in open pits and artificial installations (tanks) and special heap piles with commercial grade ore.

At present about 25-30% of the World copper production comes from GTP – over 80% from open pit mines with commercial grade ores and non-commercial grade waste dumps and very limited quantities from flotation tailings.

In Bulgaria GTP of copper comes as follows: 1) Installation "Elshitsa" (since 1970) for leaching of overburden (waste dump) and production of copper concentrate. 2) Installation "Medet" (since 1988) for heap leaching of difficult to enrich commercial grade ore and production of copper concentrate. 3) Installation for overburden leaching on a waste dump "Tsar Asen" (since 1985) and production of copper concentrate (until 2003) and after that production of copper. 4) Installation for leaching difficult to enrich commercial grade ore ”Asarel” (since 1992) and production of copper concentrate (until 2007), at present production of copper. 5) Installation for reactor leaching of flotation tailings “Medet” and production of copper (from 2004) – this is another achievement of the “MSB”(Bulgaria) company, the capacity of the geo-technical installation is 5 000 t of copper annually, not achieved yet!

The current experience with GTP of uranium and copper shows that the method is up to 5 times more efficient than the conventional underground and open pit mining of ores (+ beneficiation + transportation + metallurgy). In addition it is with minimum ecological problems and full rehabilitation of the environment and the waters.

On an industrial scale is developing also the GTP of precious, rare and disseminated metals – mainly from non commercial grade ore, metallurgy and the chemical industry, as well as difficult to enrich ores and as well as associated metals in copper, uranium and lead and zinc deposits.

Scientific research accomplished in Bulgaria (Danov, V., 2008).

In the beginning of the 60’s of the 20th century in “Nipropuda” was established a new scientific department for leaching copper from difficult to enrich ores and non-commercial grade ores under the managements of senior researcher Dr. eng. N. Semkov and with the associates N. Shtirkov, M. Mihailov, P. Nesterova and others.

During approximately 13-15 years were performed laboratory, technical and economic studies for copper leaching (from the above mentioned types of raw materials) for all copper deposits: Medet, Prohorovo, Asarel, Vlaykov Vrah, Petelovo, Elshitsa, Radka and some from the Burgas region.

A long term program was approved for further development of the studies for industrial application up to 1985-90 in the directions of leaching in open pits and underground mines.

But the Nipropuda program gradually faded away. The underground direction was not developed.

In the Higher Institute for Mining and Geology (HIMG) (now Mining and Geology University (MGU) “St. Ivan Rilski”) in 1973-74 began laboratory studies for bacterial-chemical leaching of copper from difficult to enrich ores and non-commercial grade raw materials (deposit Vlaykov Vrah) under the leadership of Dr. St. Grudev (research assistant at that time). The good results were implemented only in the project for underground leaching of copper from shrinked ore in the underground mine “Izdremets” (near the Bov rail station) – project leader prof. V. Danov. The necessary construction was completed but the start of the installation was stored for subjective reasons.

The last problem (developed by HIMG – project leader prof. V. Danov), was the national project for staged application of geo-technological methods for mining copper deposits in Bulgaria and from the waste dumps (heap piles and flotation tailings) during the period 1985-1995, with continuation. This project was also stopped for subjective reasons.

The general conclusion from the above analysis is: for 40 – 45 years, with the existence of significant copper reserves in deposits and the non-commercial grade raw materials (suitable for copper production mainly by leaching) the industrial scale applications are relatively limited, even symbolic compared to the possibilities.

2 RESERVES OF COPPER IN DEPOSITS AND IN NON-COMMERCIAL GRADE RAW MATERIALS

2.1 Commercial Reserves in Deposits(MEW, 2008)

The registered by 01.01.2009 copper deposits and the copper reserves are shown on Table 1.

The reserves in steeply inclined vein deposits (mainly in Zidarovo) are approximately 21 100 000 t of ore or approximately 151 501 t of copper (approximately 10.5%).

Almost all reserves of copper 1 302 200 t (approximately 90%) are contained in ore stocks, which are not economical to mine by open pit, mainly because of the very low metal content and low effectiveness of the floatation, particularly for the largest deposit Prohorovo.

Table 1

Parameter	Ore 000.t	Copper content, %	Copper metal t
1. Elshitsa, sulf. ore	1470	1,23 .- 1,30	18740
2. Radka	1528	0,78 – 1,03	13280
3. Popovo Dere withc Zn and Pb	75000	0,21	155870
4. Medet (below mining level) *	86000	0,32	250000
5. Petelovo (Kominsko Chukarche)	3050	0,37 – 0,18	9620
Total in the Panagyurishte region	**167000**	**-**	**447500**
6. Meden Rid –sulphide ores	250	1,73	4300
7. Varli Bryag	210 1200	2,15 1,00	4570 11630
8. Propadnala Voda	870	1,16	10300
9. Cherni Vrah	460	1,06	4820
10. Strandzha (Propada)	210	1,57	3200
11. Bradtze Scarn sulphide ore Supphide and sulphide oxidised	55200	1,37-1,32 0,26 – 0,21	124480 33400
12. Gradishte – sulphide ores	1720	2,13 – 1,33	30780
13. Zidarovo	10780	1,27 – 1,23	130700
Total in the Burgas region	**80900**	**-**	**381320**
14. Prohorovo – Nova Zagora Sulphide – oxidized ore	250000	0,25 – 0,22	625000
Total for Bulgaria	**497900**	**-**	**1453820**

**Asarel and Elatsite (below mining level) after closing there would remain reserves of sulphide ores (for underground mining) analogous to those in Medet.*

2.2 Extractable Reserves of Copper from Noncommercial Grade Raw Materials (MEW, 2008)

The quantities of copper in non-commercial grade raw materials in Bulgaria could be summarized, as follows:

1) In heap piles of open pit copper mines: total rock mass of over 1 billion t with average content of copper 0,10-0,08% or 800-1000.10^3 t; of those extractable through (forced!) leaching for 30 to 50 years are over 450-550.10^3 t.

2) In old copper tailing ponds: Medet (approximately 250000 t of copper), in Burgas Copper Mines (60000 t), in Panagyurishte (40000 t), in tailing ponds of lead and zinc flotation mills there are approximately 120-150.10^3 t of copper: A total of over 500000 t, extractable reserves 250-300.10^3 t/year.

Actually in the currently operating flotation mills at Elatsite and Asarel after they are closed down around 2022-25, there would be over 1 bil. t of tailings and approximately 1 mil. t of copper (+molybdenum, nickel, gold and silver); extractable over 550-600.10^3 t. copper/year.

In Chelopech after the closing down during 2025-30 there would be about 50 mil. t of tailings with 100-150.10^3 t copper, approximately 50 t of gold, 300 t of silver, 100-150 t of germanium and significant quantities of other rare metals, the extractable reserves are about 60-75%.

3) In the underground mine waters of 10 copper mines with 80-130 mg/l are lost 350-400 t copper annually; extractable 280-320 t/year.

The mine waters from the heap piles of Meted, despite the recultivation (but without a clay insulating layer) are acid with 60-80 mg/l copper and at least 150-200 t/year; extractable 120-160 t/year.

The waters from Elatsite and Asarel with 80-120 mg/l, carry out of the heap piles 700-800 t copper/year; extractable 550-650 t/year.

4) In metallurgical waste (waste dump) of the plant in Pirdop are piled over 20 mil. t (2008) with 1-1.5% of copper or over 250-300.10^3 t; extractable 180-250.10^3 t. Near the Lead and Zinc plant in Kardzhali, in the waste dump, there are 50-60.10^3 t copper and as much zinc; extractable are 70-75% .

In total by 2009 the raw materials (non-commercial grade ore, flotation tailings, metallurgical waste) in Bulgaria contain 3800-3900.10^3 t copper; extractable 1500-1700.10^3 t; significant quantities of gold, silver, germanium, cobalt and other rare metals (rhenium, vanadium, rare earths).

The main objective for the future is the ecological safety of these raw materials sites by applying geo-technological methods. Simultaneously it would be possible to produce metals and salts of metals and to cover a significant portion of the costs; it is possible even to clean them at a profit.

2.3 Total Quantity of Reserves for Effective Geo-Technological Production of Copper "In Situ" in Prospective for up to 2040-50 and Further into the Future

a) in copper deposits: the commercial reserves are 1454.10^3 t copper.

With the technology for underground leaching of shrinked ore and the long degree of extraction (with the innovative technology) 70-80%, the extractable reserves would be minimum $1020 \div 1150.10^3$ t copper.

b) In non-commercial grade raw materials: the commercial reserves are $3800 \div 3900.10^3$ t of copper.

With the technologies and methods for forced leaching (with the objective of ecological safety!) and long term degree of extraction 40-50%, the extractable reserves would be minimum $1500 \div 1700.10^3$ t of copper.

c) total extractable reserves for effective geo-technological production of copper "in situ": minimum $2500 \div 2850.10^3$ t of copper.

3 TECHNOLOGIES AND METHODS FOR UNDERGROUND LEACHING OF COPPER DEPOSITS IN BULGARIA

The large scale of application and the significant scientific experience confirms that in the future the geo-technical production of copper in situ from deposits would develop increasingly as a much more effective and (particularly important!) ecologically safe method compared to the conventional underground and open pit methods for production of ores + beneficiation +metallurgy + significant and long term damages to the ecology and the waters.

3.1 Effective Technologies for Underground Leaching of Shrinked Ore (Danov, 2006)

a) sulphur-acid technology for copper oxidized and oxidized-sulphide ores: consumption of H_2SO_4 5-10 g/l water, pH=1,5-2,5; intensification is recommended with bacterial solution of Fe and S- dissolving tiobacillus bacteria (produced in a bio-reactor, activity 4-5 mil. cells in 1 ml) in order to increase the kinetics and reduce the cost of the production process..

b) sulphur-acid technology for sulphide-oxidized and sulphide copper ores: consumption of H_2SO_4 10-15-30 g/l, pH=1-1,2-2; mandatory bacterial and intensification with chemical oxidant and by other methods (see Section 5 below).

c) technology with complex acid reagents according to patent, which have been commercially tested and new technologies in the future: with bacterial and by other methods intensification.

d) alkaline technology: for ores with carbonate content above 3-4%. consumption of carbonate reagents 30-50g/l of water, pH=7,5-9,5; various types of intensifiers are required.

3.2 Effective Methods for Underground Mining by Leaching of Blocks with Shrinked Ore (Danov, 2006)

In every production block the ore mass is blasted at once by long blasting boreholes in order to achieve optimum size of the shrinked ore (size 80-120 mm), i.e. with a fraction up to 200-250 mm.

The experience (mainly in the USA and Canada) had proven the following mining methods – with high technological effectiveness and degree of copper extraction 70-80%:

a) in steeply inclined deposits – veins or stocks (developed in vertical production blocks):

On Figure 1 is shown the principal variant of a mining method with underground leaching of shrinked ore.

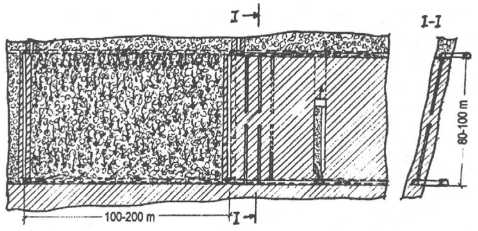


Figure 1

This method could be applied for the vein deposits in the Burgas region – mainly Zidarovo (130 000 t of copper) and in the other deposits with small reserves.

Copper-pyrite stocks in the Panagyurishte region – Medet (below the open pit with 250000 t copper) and others with smaller reserves, as well as the stock of Prohorovo (over 600 000 t of copper), they could be mined effectively and ecologically safe in the future only by underground leaching of blocks with shrinked ore ("in situ") – according to the variant on Figure 1.

The technological regime is gravitational with continuous sprinkling in a closed cycle for every production block.

b) for deposits with low inclination (up to 45-50^0) it is rational to apply room and pillar method (Figure 2) for underground leaching of shrinked ore.

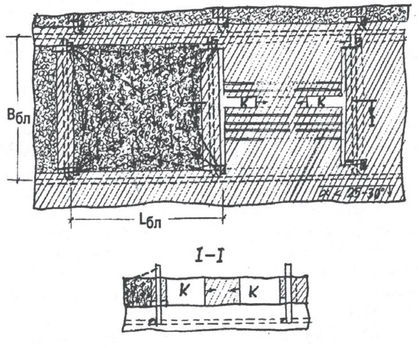


Figure 2

Technological regime hydrostatic cyclic in a closed cycle for every production block.

c) on Figure 3 is shown the design of the authors for a combined method for underground leaching of shrinked ore with a blasting boreholes and sprinkling from the surface under the conditions of the Bradtse mine (Malko Tarnovo – Burgas region) with reserves of approximately 160 000 t copper.

The openings preserved from the former underground mine, which was closed 1998, would be used: the rock heading at the lowest level for accumulation of product solution and shaft No. 3 for hydraulic transportation.

4 TECHNOLOGIES AND MINING METHODS FOR GTP OF COPPER WITH OPEN PIT WORKS FROM WASTE DUMPS AND COMMERCIAL GRADE ORE

In the World practice for GTP of metals with open pit works the following mining methods have become standard for leaching commercial grade ore and non-commercial waste dump raw materials.

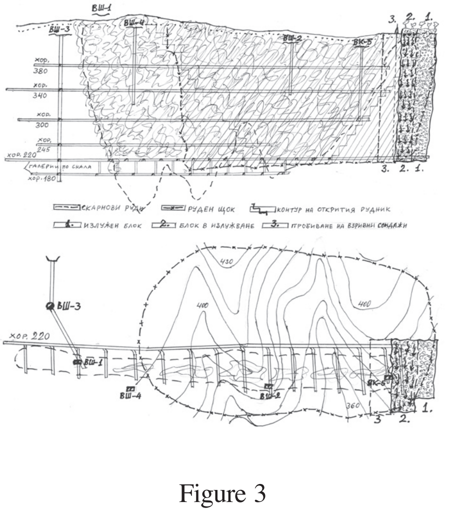


Figure 3

1. Methods for leaching of ore in men-made installations – variants for thin layer leaching in tanks (Figure 4), stage leaching, trenches, leaching in reactors. Metallurgical waste is leached in such installations as well.

2. Methods for leaching old waste rock heap piles (without hydro-insulation of the stage), (Figure 5) and newly build heap pile stages (with such insulation). Here are

included old and new heap piles and also waste from metallurgy.

3. Methods for leaching flotation tailings in reactors.

4. Methods for autoclave leaching of metal concentrates.

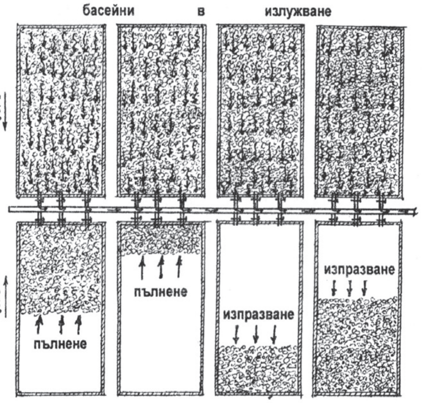
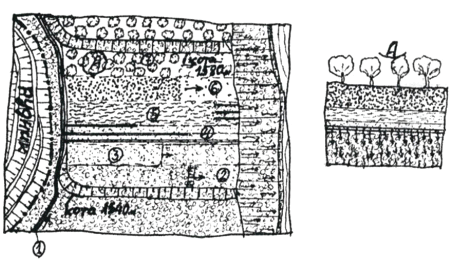


Figure 4



1 – магистрален тръбопровод за излужващ разтвор; 2 – сектор в булдозерно подравняване; 3 – сектор в насипване на пясъчен слой; 4 – сектор в полагане на оросителни тръби; 5 – сектор в разстилане на глинест слой; 6 – сектор в насипване на рекултивационен слой; 7 – сектор в засаждане на дръвчета

Figure 5

The large reserves in waste dump sites (2500÷2850.10^3 t copper!) allow for and is possible to implement them up to refined copper, mainly with the objective of ecological safety.

The long experience in Bulgaria and abroad proves that forced leaching is the only method make them ecologically safety. In addition it is economically effective!

On Figure 6 is shown the scheme for a waste dump – based on the example of a section of the western dumps of the Elatsite open pit.

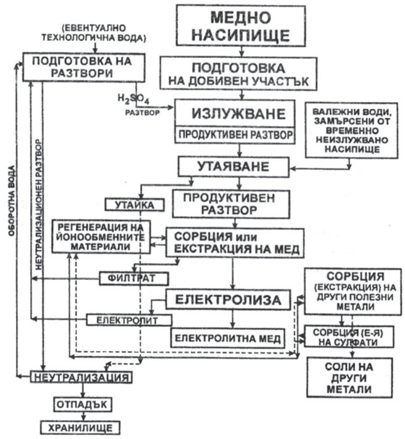


Figure 6

On Figure 7 is shown a combined scheme of a geo-technological plant for copper production with sub-systems for heap pile leaching, thin layer leaching of ore in a tank and autoclave leaching of copper concentrate and hydro-metallurgical plant. This is approximately the Nchanga, Zambia installation.

In the future geo-technological plants in Bulgaria (for example Prohorovo and in the Panagyurishte region) the development ore would be leached on the surface using the methods shown on Figures 4 and 5.

With reference to the technology of hydro-metallurgical processing of the productive solutions.

The long experience has proved:

1) Selective sorption with ion-exchange resins with a subsequent de-sorption and concentration of copper sulphates above 15-20 g/l for electrolytic production of refined copper.

Figure 7

This technology is applied effectively for productive solutions with relatively low content of CuSO4: from 80-100 mg/l (the lowest concentration) and up to 400-500 mg/l.

2) Selective extraction with liquid ion-exchange extraction agents with subsequent re-extraction and concentration of CuSO4 above 15-20 g/l – it is applied with higher content, above 400-500 mg/l up to 1-3 g/l solutions.

With both technologies the number of ion-exchange resins depends on the number of components which are contained in the productive solutions.

With the development of innovations we expect improvements of these technologies – particularly with respect to the effective extraction of the associated precious and rare metals in the copper ores.

5 INNOVATIVE STRATEGY FOR THE DEVELOPMENT OF FUTURE GEO-TECHNOLOGICAL MINING COMPANIES

The main objective of the innovative tasks is to increase the technical and economic effectiveness and to ensure ecological safety of the geo-technological methods for mining copper deposits and waste dumps.

In principal, our view about the innovative tasks is as follows:

1) It is advisable that a new and detailed study is made of the ores mineral and chemical composition (possibly the bad rocks as well) and the technological waste, particularly about associated metals.

2) A study should be made to improve the acid reagents and for the creation of new reagents for complex leaching of copper and the associated metals.

3) Studies about additional chemical oxidants to the main acid reagents with the aim to increase the kinetics in the ore peaces and to increase the degree of extraction of copper and the other metals.

4) Studies of the tio bacteria aiming at (eventual) selection of new types for direct oxidation of the primary and secondary sulphates of copper, in addition to of the Fe- and S- dissolving the pyrite tio bacteria, producing oxidizing ferrisulfate, intended for the sulfate ores.

5) Studies for improving the technological regimes and maintenance of effective filtration in the blocks with shrinked ore by suitable physical methods: utilization of blasting borehole or hydrodynamic periodic shacking of the shrinked ore, application of high frequency electric current or electric field with normal frequency; it is possible that other physical and physics-chemical methods are discovered.

To improve the variants of the shrinkage mining method by the methods of development and cutting, the construction elements and parameters of the production block and the parameters of the mass borehole blasting of the ore.

6) The same for the waste dump raw materials.

7) To improve the existing and eventually to create new ion-exchange materials for highly efficient extraction of copper and the associated metals and the harmful components (arsenium, sulfates etc.) and with the objective to reduce the costs of their production.

8) To improve the machines and other equipment (sorption columns, extraction tanks, electrolytic tanks, control and measurement devices, process automation, etc.) in the hydrometallurgical installation (plant).

9) To develop an automated systems for design and control of the copper geo-technology plants.

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Underground Workings Over 13 km Total Lenght Serve for Ellatzite Open Pit Mine Operation

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ABSTRACT There are eight underground workings of various lengths, from 163m to 6 343m, to serve Ellatzite open pit mine and their total length exceeds 13 125.27m. Internal cross-sections vary from 5.1m^2 to 12.4 m^2 respectively. All these workings solve various ecological or technological problems associated with the ore mining.

1 ELLATZITE DEPOSIT UNDERGROUND WORKINGS

Ellatzite deposit is located about 80km to the East of Sofia on the Northern flank of Etropolska Stara Planina, near the town of Etropole.

In 1976 the construction of ore mining complex started. The deposit is worked out by open cast method. The technology of extraction is conventional - drilling and blasting using modern mining equipment; drilling and blasting of 15 m benches with cartridges & emulsion explosives our own production.

In virtue of the Law of Ores and Minerals, the ores and minerals are exclusive state property. Extraction is possible upon granted concession, i.e. buying out needed license rights giving permission to extract ores and minerals under definite terms for a fixed period of time. By Resolution No. 594/02.09.1999 of Council of Ministers, Ellatzite Med AD Co. (that is part of the “GEOTECHMIN OOD” group of companies’) was granted a concession, subject to special rights to use ores and minerals: porphyry copper gold bearing ores.

Concession period is over 17 years time period. On 02.10.2006 a Supplementary agreement No. 2 to the Contract has been signed. In accordance with above mentioned supplementary agreement the term of granted concession to extraction of porphyry copper auriferous ores from Ellatzite deposit was extended by 5 years towards initially set term.

During 2008 56 million t ROM at Ellatzite Mine have been extracted, including 13 million t of ore, to make Ellatzite Mine one of the biggest open pit mines in Europe.

In spite of working out of the deposit by open cast method, the entire period of time – from construction to the present moment was marked by cutting more or less unique underground workings to solve various ecological or technological problems associated with ore mining.

The first underground working is in connection with Ellatzite Mine development and its unique features being the result of Ellatzite deposit location on the Northern flank of Stara Planning and a processing plant built on the South flank of Stara Planina, also advantageous topography to construct needed appropriate tailing dams. Extracted ore from the deposit after crushing enters Mirkovo Processing Plant that is located on the south flanks of Stara Planina Mountain through underground rubber conveyors. **The length of the tunnel passed by the rubber conveyors is 6343 km,** internal cross-section 11.16m^2 or it crosses the main Stara Planina Range. This facility has been constructed for less than three years, from 1976 to 1979.

The following underground working is related to the underground drainage of Ellatzite Mine, environment protection and partly Malak Iskar water. The project and construction have been realized by "Geotechmin" OOD Company. Basically, it is a **drainage gallery from 1030 m elevation to 1031.63 m elevation,** catching surface and underground water in the mine pit. Water enters a pipeline passing over a tunnel for crushed ore transporting, then it is flow in the processing plant as circulation water. The gallery, **660 m length, internal cross-section** 5.1 m^2, 3% of inclination, it was constructed during March – December 1995. The gallery was cut by drilling and blasting. Continuous concrete of 200 mm thickness supports the mouth and metal frames at 10m interval. The part to the fault section along the whole length is fixed by anchor bolts in the arch. Supported sector of 10 m in the vicinity of hydro facilities by concrete support and three-joint metal frames; metal frames behind the fault to the end. The part to the fault section, 15m - concrete and metal frames. After fault section to the end – metal frames.

Next project continues to solve ecological problems, by utmost catching polluted water from western stock piles, by diversion and pure water preservation of the Negrstitza River. The project has been designed and executed by "Geotechmin" OOD during September 2001 – October 2003. That is **a tunnel in Ellatzite Mine western stock piles range, 832 m length,** 1, 5% of inclination, internal cross-section 6, 10 m^2 to 7.9 m^2 according to applied type of support. Internal cross-section dimensions are: 2.40 m width, 2.65 m height. The tunnel was made after conventional technology: drilling with ПП-54 perforators and blasting with „Amonit-6" and „Elacit" explosives, manufactured at the explosive plant owned by "Ellatzite Med" AD. The whole tunnel bed is fixed with permanent gun-applied concrete lining 30 mm thickness, monolithic sulfate resisting concrete. Entire tunnel is passing close to the surface relief, through intrusive rock – granodiorite, within the interval from 0, 00 m to some 70 m grus(s) /fragile; but within the interval from 50 m to 70 m lays a tectonic fault. The cutting of that tunnel creates potentiality of additional quantity overburden piling; however by means of a subsystem construction projects to catch up to 200 l/sec polluted water from Negarstitsa River mostly, where over again it goes via the ore transportation tunnel to the processing plant.

Until 2004 Ellatzite Mine drainage was carried out by pumps, underground water was pumped up to enter the pipelines installed in above presented existing drainage gallery at 1030 m altitude. That kind of drainage is associated with a great number of problems during assembly or disassembly caused by frequent blasting. At the same time the pit slopes steepened notably. In order to maintain appropriate optimum slope angle, additional drying was a precondition. Open pit water collection area is equal to 0, 92 км2, average flow 25-30 l/sec; when high water flow reaches 202 l/sec. All that provoked further designing and completing of a project better than the previous projects' with the purpose Ellatzite open pit mine drainage and drying. In 2002 a designers team to "Geotechmin" OOD worked out a project, the subject of the project:" **Drainage gallery under the pit bottom at level 955"**, to carry out gravitational drainage and to dry the open pit mine from level 1030 to level 955, where water passes through two chimneys being cut from the gallery to crosscut the last level. Construction of the drainage gallery was complete by "Geotechmin" OOD, starting at the beginning of 2003 and finished end of 2004 година. The l**ength is 1920 m,** 0, 04% of inclination, 5.90 m^2 single center arch internal cross-sections. Internal cross-section dimensions: 2.40km width and 2.70 m height. It was the conventional technology of cutting the drainage gallery. Drilling and blasting involve ПП-54В perforators and an Elacit explosive, manufactured by explosive plant owned by Ellatzite Med AD. Removed rock was loaded by box loading equipment; haulage by electrical car transport. A provisional air compressor station supplied the air. Mine drainage is gravitational; ventilation involves a combined set up of airing: both suction and pressure line. The gallery is supported by sprayed concrete lining 2-3 cm thick and metal anchor support along the whole length, and continuous concrete of 0,30m thickness in the tectonic faulty areas. The provisional support was made by two-joint metal framework and advance support with steel ties.

At the beginning of 2005 the drainage gallery "955" was put into operation, after that the problems with drainage and drying of Ellatzite mine were fully solved. Also, in addition some engineering and mining solutions were completed allowing an overall slope angle increase by 3 – 5 degrees in different profiles.

Metal price increase at the London Metal Exchange until 2008 supposed a new vision on Ellatzite deposit reserves, and technical outlines applied for working out. As a consequence, at the beginning of 2007 the work on an actual technical contour, designed by "GEOPROJECT "EOOD, a subsidiary of "GEOTECHMIN" OOD, was finished with overall slope angle 48^{o} , the pit floor level 880, length 1900m and width 1400m. This contour of Ellatzite open pit mine ensures extraction of 120 Mt proven reserves (Category 111), as of 01.01.2009, thereby increasing lifetime of the pit until 2022, thus to satisfy renewed concession term completely. With the new Ellatzite enlargement mining operation goes to depth, while transport distance increases. Under existing circumstances it became necessary to optimize transporting of the crushed ore, resulting in construction of our new in-pit primary crushing plant (KET 3) and transportation of crushed ore by rubber conveyor belt through a tunnel that is the **underground link between the** new built in-pit primary crushing plant (KET 3) and the existing crusher out of the mine. The l**ength is 565 m,** 12.1 m^2 internal cross-section, single center arch. Internal cross-section dimensions: 3820mm width and 3100mm height. From the area "**Tunnel entry"** the intersection of the tunnel is at elevation 1034 m, horizontal inclination and 15 m length; then rising of 15.65% to the ex primary crusher at 1120.60 elevation. It was a curious fact during the time of construction of that tunnel; about 25% of the tunnel passed through a pile consisting of materials of different grain size and strength properties. That section was cut following a special method, by advance injection and support work.

The new primary crusher is located within the pit. Mine atmosphere there is very much polluted from dust and diesel engine exhaust gases. To serve the need of normalization of existing working conditions in the crushing department, a **tunnel, called "Exit"** was designed and constructed, connecting the one above described with a working area out of the pit, thereby to reach the targets to:

Feed the necessary air quantity of 8.5 m^3/sec by compression for ventilation of the recently build primary crusher plant.

Secure an easier provision of machines and equipment, both for assembling and repairs when commissioned and in service.

This additional **"Exit"** to the main tunnel, **163 m length,** internal cross-section of 11.45m^2 and single center arch, 3.28% of inclination and internal cross-section dimensions: 3800mm width and 3400mm height. This tunnel was driven following a conventional above described method, too, but yet using up-to-date equipment: BOOMER 281 drilling carriage, SCHOPF front loader – 3.7 m^3 shovel capacity, ANFO explosive once again manufactured by "Ellatzite Med" AD explosive plant and initiated by non electric detonators.

The pit bottom lowering provoked the need of modern pit drainage solutions. For that purpose, in 2006 started the construction of a new **tunnel, a drainage adit** under the pit bottom, at an elevation 840, **2085 m length.** This to carry out gravitational drainage of the open pit mine from level 950 to level 860 by discharging underground water through blast holes drilled from level 950 to a connecting hole with a chimney drilled in advance.

Internal cross-section of the tunnel is 12.40m^2 with single center arch. Dimensions of the internal cross-section, as follows: 3.4 m width and 3.9 m height. That is the cross-section of those intervals of hard host rock, where supported by ТФА metal anchors or 3 mm gun-sprayed concrete. To intervals where applying of provisional metal support of КОП-21, 5 two-joint metal arches was needed, the drill cross section is 14.60 m^2. Dimensions of the internal cross-section, as follows: 3.80 m width and 4.15 m height, sprayed concrete lining thickness 20 cm, so to set final cross-section of 12.4 m^2. Also here conventional drilling technology using modern equipment as it was introduced above.

Final working serves Negartstitsa River water protection over again. **"Additional buildup"** of already built facilities is needed to achieve better and more reliable water collection and to evacuate pure and polluted water, being in direct contact to North pile close to Negarstitsa River. That is drain-pipe extension ahead of actual tunnel for pure water and extension of above described tunnel for polluted water. This means **new-designed of 557.27 m length,** internal cross-section 10.30 m^2 single center arch and internal cross-section dimensions: 3400 mm width, 3300 mm height.

It is understood from the retrospective review made so far of tunnel construction at Ellatzite open pit mine area that **eight underground workings, total length over 13 125m serve for technological and ecological activity at the mine.** All workings, with the exception of the original one, have been designed and realized by "Geotechmin" OOD and/or subsidiary companies, moreover up to this moment all of them have fulfilled or fulfill today their own technological or ecological designation what is a guarantee for the experience and professionalism of this company when designing and implementing installations of similar type.

REFERENCE

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New Crushing Transport System at Copper Mine Ellazite-Med AD

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ABSTRACT The mine floatation complex Ellazite AD has been operational since1982. It processes copper porphyry ores with design capacity of 10 million tons. The complex is situated at two platforms. There is an open mine and the Primary Crushing Department for the ore in the north of the ridge of Balkan. In the south of the mountain there is a floatation plant, waste depot and the water supply system. They are connected with the help of a rubber transport conveyor, 7 km long, installed in a tunnel with a slope of 7%. In the process of exploitation the transport distance from the obtaining locations to the Primary Crushing Departments have increased.

In the last few years there is constant tendency for price increase of the petrol products. The construction of a new Crushing Transport system aims to reduce that negative impact by decreasing the share of the automobile transport and increasing the conveyor one. As a result new rubber transport conveyors are added to the existing one, all installed in tunnels and the crushed ore material is stored on the number 1 open storage. The connecting tunnel is driven in predominantly hard rock, but one part of the tunnel passes through inhomogeneous rock and solid materials, which require a special method. Also the problem with the ventilation system is specifically addressed in order to avoid the inhalation of gases by the staff, servicing the Crushing Department and the tunnels. The existing infrastructure of the site is considered so as not to affect the intense production schedule of the complex. The realization of the new Crushing Transport system will help reduce the production costs, the dust pollution, gas and vapor emission and to increase the reliability of the transport in the mine during the winter months.

1 INTRODUCTION

The mining and concentrating complex “Elatsite” has been put in normal operation in 1982 for processing of copper porphyry ore after broad geological exploration, investigation and engineering activities, performed by “Niproruda” and after 1992 by “Geotechmin” Ltd, Sofia. It has a design capacity of 10 million tons per year. Here are implemented the conceptions for development of mining and concentrating in 80s of last century, adapted to the potentiality of production in Bulgaria and for import of process equipment from abroad.

The complex is built on two platforms. To the North of the Balkan mountain ridge are located the open pit and the Primary Crushing Department (PCD) No.1, and to the South – the Concentrating plant, tailing pond and the facilities for the circulating water supply system.

In 1994 has been built and put into operation PCD No.2, which is equipped with a cone crusher “ALLIS” 54-74. Both plants can operate individually or together.

The connection between the two sites is made via a main rubber-belt (MRBC) of length about 7 km. It is installed in a tunnel.

The open mine is situated in central Balkan mountain at 1000÷1500m altitude. The region of the deposit is built mainly by three types of rocks: Paleozoic metamorphic complex, granodiorites and dike rocks. The most widely appeared granodiorites.

The deposit is of streaked injected type and it is connected to the contact zone of granodiorites with schist. The main ore minerals are chalcopyrite, pyrite, bornite and molybdenite, and the non-metallic minerals – quartz, potassium feldspar, biotite, sericite, chlorite, epidote and calcite.

During the period of operation of the mine about 250 million tons of ore have been produced. It is estimated that another 150 million tons will be extracted. In the course of operation of the complex have been made innovations in the mine, concentrator, tailing pond and the auxiliary plants. As a result of this its capacity reached 13 million tons ore per year. In 2008 in the mine were extracted 12.7 million tons of ore. Now is being implemented the third stage of mine expansion for ensuring its operation till 2019.

At present the mine is deep with the deepest board about 500 m while in future will reach about 650 m in its steepest part.

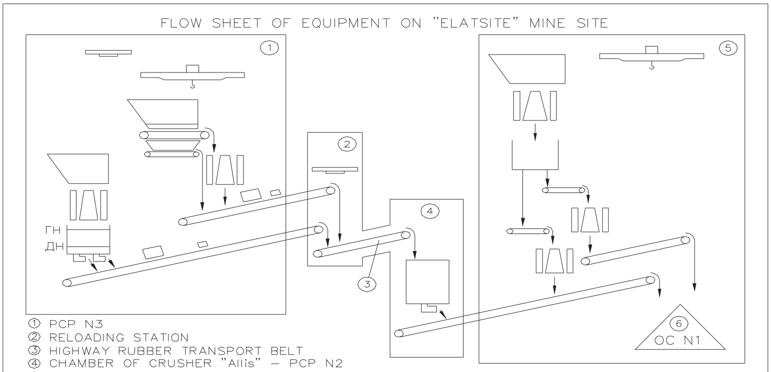

fig. N1

The steps are constructed with angle 65-70°. Special attention is placed on the protection the batters from the blast effect by contour explosion at board angle 46÷48°.

The blasted ore mass is loaded by diesel and electrical excavators with bucket volume 10 and 15 m^3 and after that it is transported with dump trucks with weight bearing capacity 100 and 130 tons.

In certain periods throughout the year an air inversion is observed and the harmful gas and dust emissions fall in the pit of the ore mine.

In result of the mining works gone in lower depth essential changes appeared in the organization of the ore haulage. The slopes of roads in the mine area contour reached limit values. The transportation distances from the mining horizons to the existing PCD No.1 increased and for the period of operation of the mine till 2019 will be changed within the limits of 1,9-3.5 km.

In the last few years is observed a constant tendency for price increase of dump trucks, spare parts and the petrol products.

In order to be minimilized the negative effect of these factors, as well as of the harmful effects of gas and dust emissions a new crushing and transportation system has been developed. It comprises reducing the share of car transport versus haulage organized with stationary rubber-belt conveyors (RBC), and the location of the Primary Crushing Department (PCD) in it. In

Fig.1 is shown the flow sheet diagram of the new system and the initially built PCD No.1 at the mine site.

It comprises:

- New Primary Crushing Department – PCD No.3 in the mine area – 90 m lower than the existing one;
- Trestle with two outgoing belts;
- Reloading station;
- Main conveyor belt, installed in the tunnel;
- Reloading station in PCD No.2;
- Existing RBC, which conveys crushed ore to storage yard No.1

The location of the PCD No.1 within the mine area is chosen in consideration with its development, minimum reserve of resources, the existing technological infrastructure, the relief, the geological conditions and the safe conduction of blasting works.

In PCD No.3 is applied one-stage scheme of crushing, where two independent circuits are formed.

The first section is equipment completed with machinery delivered by "Thyssen Krupp". It is equipped with a jaw-cone crusher "KRUPP" BK 160-190, which is fed by one side. The intake opening is 1600 mm wide and can passes-in pieces with maximum size of 1350 mm. The dozing in the feeding process is done by means of an apron feeder. The design capacity of the crusher 1800 t/h. with content of size +200mm – 5% in crushed ore. The control of this section is done by the charge of electrical motor and the level of filling of the crushing chamber with ore and the height of ore layer on the apron feeder. The data is processed and is used for control of the speed of feeding. Driving is achieved by two geared motors and one frequency regulator. A hydraulic hammer is installed for breaking the oversized pieces onto the apron of feeder and in the crushing chamber of the crusher "KRUPP" BK 160-190.

The second section is equipment completed with a cone crusher "Allis" 54-74, which is dismounted from PCD No.2. It is fed with ore with maximum size 1100 mm, which the dump trucks pour into its receiving bin. It has catalogue capacity 1650 t/h with content of size +250 mm – 5% in crushed ore. The chamber under the crusher has effective volume 300 tons. The latter is discharged via two vibratory feeders with frequency control of vibrators. Control of the section is carried out at lower and upper level of crushed ore in the chamber. A hydraulic hammer is installed for breaking the oversized pieces.

The design capacity of PCD No.3 is 2500 t/h when both sections operate. Mainly operates the first section with throughput 1800 t/h, and the second one has capacity of 700 t/h. The control of the two sections is carried out via belt scales.

Under each crusher there is an outgoing RBC with design capacity 2500 t/h. They are installed in a trestle, which connects PCD No.3 with the reloading station.

The reloading station is located close to PCD No3. In it are installed the two driving stations of the outgoing conveyor belts, tension station of RBC, which is installed in the tunnel. A crane is installed for carry out repair works.

RBC in the tunnel connects the reloading station with the existing chamber of PCD No.2, where is installed its driving station. It has two driving rubber-lined drums. The conveyor belt is 558 m long and it is installed in a tunnel with permanent slope 16%. Its structure is developed by "Thyssen Krupp".

The crushed ore, conveyed by RBC with a help of a vibration feeder with controllable frequency and amplitude, is unloaded onto the existing RBC. It conveys the crushed ore to a heap in an open storage yard No.1 with effective volume to store 30 thousand tons.

The tunnel is 540 m long and its clear cross section 10,60 m^2 with one-centered arc and angle of inclination 16%. The section dimensions are: width 3820mm and height 3100 mm. It passes through solid rocks and embankment with length of about 150 m. The rock embankment is non-homogeneous with various size and strength of materials.

The tunnel through the heap has been driven by a special method, with preliminary injection injection fortifying works. Injection anchors SYMETRIX types, a product of the company "Atlas Copco", are used. The technological sequence for driving the tunnel comprises:

- preparation works;
- test drill works;
- injection works
- Tunnel driving and installing a permanent support.

During the preparation works it is necessary to define the exact place of the technological chamber between two neighboring sections for conducting of the test drilling works. The driven section ends with a bigger safe clearance – 400 mm in each direction. This section and the heading's forepart are covered with shot Crete in order to prevent the infiltration of the injection solution.

The drill-holes have diameter 89 mm, depth 15,6 m, with an angle 3° as per tunnel axis and average distance between their axes -300 mm. The drill-holes in the upper cross section of the tunnel are arranged at 300÷400 mm from the tunnel's wall. Pipes are placed in the drills and every 1 m there are 4 openings with diameter 12 mm, placed in two diametric planes at distance 60 mm. Every opening has a reverse valve. The pipes should be placed so, that the injection openings to be arranged in chess board fashion. The treating of the heaped material with a solution starts from the bottom upwards and the parameters of the solution are decided in detail for each separate drilling. The final injection pressure is limited to 8 bars. Decreased pressure is also applied when necessary.

Excavation works are performed with maximum length of step 2m.In certain conditions drill and blasting works are also admissible, observing a specific design.

Four types of profiles are developed for the permanent support based on the geometrical characteristics of the cross section. The support-lining design is closed type with invert arc with static functions. The basic elements of the applied support and lining are lattice girders, shotcrete and steel nets. The steel lattice girders are 4 types designed for each separate profile.

Starting, management and stopping of the new crushing transport system is done by the operator in the central panel in Primary Crushing Department No.3. All the data for the parameters of the technological process and the technical condition of the equipment is concentrated in the operator station. The achieved technological results, the machines' condition and the production costs are presented in reports. The information is transmitted through the constructed optical net at certain work places in the open pit mine and the complex for analyses and making management decision. Also a system for overall television observation is created and the information is displayed on the control panel.

In the development of the project of the new crushing transport system a special attention is placed on healthy working conditions.

All technological openings of the levels for pouring the dump trucks are closed with lids. The zones of intense dust emissions are encapsulated and connected via pipelines with the bag filter. The separation of the captured dust is done in two stages. The large particles are settled in two cyclones, and the fine dust in the bag filter. The settled dust, after moistening with water, is unloaded onto the outgoing belts.

Compensating the evacuated dust is solved in non-traditional way. In the tunnel, the reloading station and PCD No.3 is created overpressure, which does not allow the polluted air from the mine to enter there. For that purpose on the North slope of the open pit there is located a station for blasting clean air in the tunnel of the main RBConveyor. At the entrance of the tunnel in the reloading station there is an installed pipeline, which sucks clean air with delivery capacity of 30 thousand m^3/h and transports it to the distribution facilities in PCD No.3. There also is included a steam heater for heating the air up to 8°C during winter when repair works are done.

The system is commissioned at the end of 2008, and in normal operation in May 2009.

The first circuit, equipped with a jaw-cone crusher "KRUPP" BK 160-190 operates with throughput 1800 t/h by the request of crusher manufacturer. The content of size +200 mm in crushed ore varies within the limits of 2÷10% depending on the condition of the linings and the specified technological regulations for operation of the circuit.

The second circuit, equipped with cone crusher "Allis" 54-74 operates with throughput 1600-1800 t/h depending on the grain-size characteristic of blasted ore, when operates individually.

When the two circuits operate together the system achieves its design capacity – 2500 t/h.

By putting in normal operation of the new crushing and conveying system of ELATSITE open pit mine will be reduced the share of the automobile transport in the remaining operation period of the complex till 2019 with 133 million kilometers. This will result in:

- reduction of production costs for haulage of ore by dump trucks;
- reduction of smoke, gas and dust emissions;
- reduction of the number of process and maintenance staff for servicing the dump trucks;
- reduction of the rods' length in the open pit mine, which should be maintained;
- increasing the reliability of ore conveyance and reducing the influence of the weather on it;
- creating pre-conditions for rehabilitation of the environment;
- creating healthier working conditions for the process personnel, who has only control functions.

After falling down the limitations imposed by Thyssen Krupp for the throughput of the crusher "KRUPP" BK 160-190, the first circuit will operate with capacity 2000÷2200 t/h. This satisfies the requirements of the technological regulations of the mine operation.

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Geo-Technological Production of Uranium In Situ – Prospects for Development in Bulgaria up to the Year 2040-50

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ABSTRACT The paper presents a brief summary of the history of the uranium production in Bulgaria. A summary of the existing uranuim deposits in Bulgaria and the environmentally friendly methods that could be used for their exploitation. An innovative strategy for the future development of the industry is presented.

1 OBJECTIVE PREMISES AND THE NATIONAL OBJECTIVES

In Bulgaria are known a large number of uranium deposits with industrial reserves of 21000 t of uranium and extractable quantities of attendant rhenium and vanadium. Of those about 11000 t are in sedimentary, exogenic (sand) seams with natural permeability ($K_ф$=2-5 m/24 h) in the Upper Trace lowlands at a depth of 150-250 m. The uranium content of the production horizons is on the average 0.02%, but uranium with 0,005% is also extracted – with sulphur acid technology for underground leaching "in situ" (Bozhkov, 2008).

The remaining 10000 t of uranium are contained in massive (non-permeable) deposits in the ore zones in the Balkan Mountain, Middle Mountain and Rila-Rodopa. The deposits are in the form of vein-type and lenticular with average content 0.12-0.08% uranium and depth of up to 500-600 m.

The forecasted reserves are 50000-60000 t uranium, based on extended radio-geological surveys and detailed metal-genesis mapping of 6 new ore regions (Bedrinov, 2008).

Uranium production in Bulgaria began in 1946 until 1992, when with a decision by the Government was closed SC "Rare Metals" and by 1995 were closed all 44 mines and sections and the factories in Buhovo (near Sofia and in the village of Eleshnitsa (near Razlog).

Over 18 million tons of ore (12000 t of uranium) were produced from conventional underground and open pit mine.

From 1970 began production of uranium "in situ" by underground borehole leaching in the Upper-Trace lowland (35 sections) and up to 1992-93 were produced 4500 t of uranium in resins "in situ". From 1985 in 3 underground mines began the application of the technology for underground leaching of shrinked ore and uranium production in resins.

Generally, in 1980 Bulgaria got at the first place in the World in the application of the high-technology – geo-technological production of uranium "in-situ" with 67-70% of the total production! In the other countries (USA, USSR and others) the share was around 25-30%.

The production of uranium by the new technology proved to be highly effective with a three to five times lower cost and as ecologically completely safe with minimized ecological tasks.

This major success was due to the complex research work performed by SC "Rare

Metals" from 1962-63 until the closing of the company.

During 2006-08 in the Ministry of Economics and energy were submitted 72 applications for uranium production by many Bulgarian, two Russian, one Canadian and one Australian company.

A National Scientific discussion at a round table [Reports] with a topic "Uranium Production in Bulgaria took place at the end of 2008. The discussion was organized by the Scientific Union of Mining, Geology and Metallurgy. Positive conclusions were achieved and recommendations were made to the Government to re-establish uranium production as a public-private corporation within the "Energetics" National Holding

1) "The main economic objective:" long term supply of NPGS "Kozloduy" and "Belene" with our own uranium concentrate for production and supply of nuclear fuel, independent from the world market.

2) "Other economic objective:" to ensure profits for the companies-concessionaires (with government participation and golden share) by applying mining methods at the highest contemporary level – physical-chemical geo-technology for uranium production "in-situ".

3) "Ecological objective:" safe mining of the deposits without long term damage, complete restoration of the environment and without damages to the health of the employees.

4) "Social objective:" utilization of highly qualified labor of engineers- geo-technologists (mining, drillers, geologist and hydro-geologist, ecologist, etc.); 10-15 times fewer employees (compared to the conventional underground production) with special high school education, with very high labor productivity.

5) "Scientific objective:" staged implementation during the next 30-50 years of improved and ever more effective solutions of innovative tasks, mentioned in the National Strategy (see below).

2 TECHNOLOGIES AND MINING METHODS FOR UNDERGROUND LEACHING OF DEPOSITS AND URANIUM PRODUCTION IN RESIN "IN SITU". PRODUCTION-TECHNOLOGY SCHEMES

2.1 Effective Technologies for Underground Leaching of Deposits

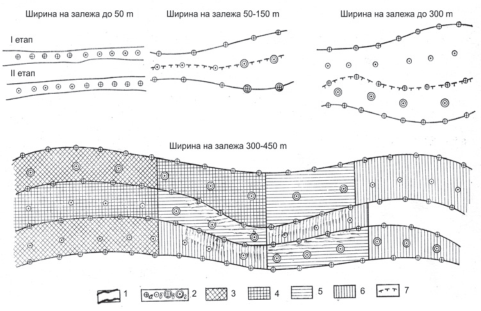
a) for deposits with mainly 6-valent uranium and low carbonate content (3-4%) the existing technology for sulphur-acid leaching should be kept without additional oxidizers at the beginning, but gradually it would be useful to implement the bacterial intensification, by building at every future geo-technological mine bio-reactor for production of solutions with highly active iron- and sulphur disassociate bacteria, which is proven to speed-up and reduce the cost of leaching.

b) with the ores with high content of 4-valent uranium and low content of carbonates the sulphur-acid technology is also the most effective, but in that case is mandatory to use additional oxidizers – bacterial mass plus chemical oxidizers and in the future more effective complex reagent plus intensifiers (see. 4).

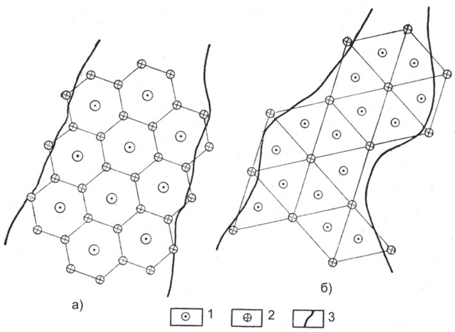
c) with the ores with more carbonates (above 3-4%) alkaline technology is applied.

2.2 Mining Methods with Underground Teaching of Deposits

a) For hydrogenised deposits in the Upper Trace Valley and in the Struma Valley (and in new areas) should continue the application of boreholes methods (which were proven in the past) - in variants with 3-4-5-6 point production sells, i.e. 2-3-4-5 pressurizing boreholes and one brine hole (Fig. 1 and 2), the principal characteristics are given according the coefficient of filtration. Respectively from 5-6 m/d to 0.5-1.0 m/d. and in the future mastering new variants at depths of 400-600 m.



Фиг.1. Технологични схеми при линейни сондажни системи. 1 – контур на залежа; а и б – нагнетателни и черпещи, в и г – нагнетателни и черпещи с висока производителност; 3 –блок отработен на 80 %; 4 – блок отработен на 50 %; 5 – блок отработен на 20 %; 6 – блок в начално закисляване; 7 – граница на изклиняване на зоната на пластово окисление



Фиг.2.Клеткови сондажни системи. а – шест-точкова; б – три-точкова клетка; 1 – черпещи сондажи; 2 – нагнетателни сондажи; 3 – контур на залежа

b) for the massive deposits (Eleshnitsa, Proboynitsa, Smolyan, Smolyanovtsi, Seslavtsi, etc.) could be effective only the methods (in situ leaching, from ecological and economic point of view) only the underground shrinkage methods, (Figures 3 and 4) with optimum size (fractions up to 50, 50-100, 100-150, 150-200 mm and minimal larger fractions); this is achieved by more effective blast-drilling technology for mass blasting and breaking the ore in the entire block. There are no limitations for the application of these methods from mining and geologic, hydro geologic, geo-mechanical conditions, surface landscape, climatic conditions etc.; for porphyroid-disseminated and streak type ores it is possible to leach uranium from poorer ore with 0.02%-0.005% and thus to increase the initial reserves.

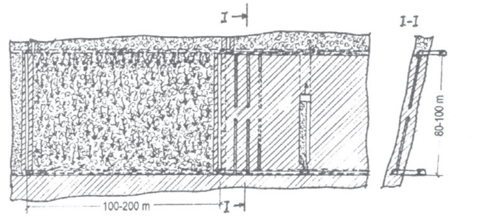


Figure 3. Shrinkage stoping for massive deposits.

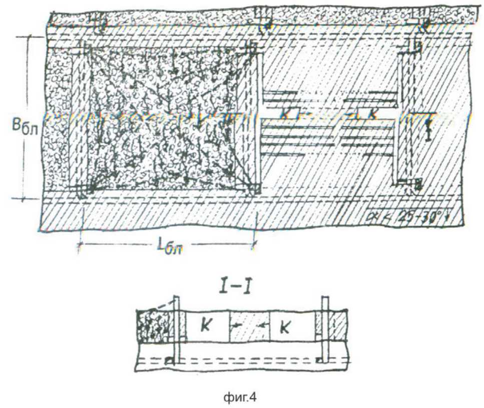


Figure 4. A variety of shrinkage stoping for massive depostis.

c) for the ore produced during the preparation of the ore blocks (including the compensation space) on the surface of each mine the most rational and ecologically safe are the methods for leaching in artificial facilities – trenches, basins, reactors – short term (up to a few months) and then placing the leached ore on a permanent ecologically safe waste pile.

Another variant is also applied: At a site with hydro-insulation is piled gradually the entire quantity of “byproduct” ore or several such sites are built and piles with a thickness of at least 20-30 m and than begins leaching by spraying, which could continue for many years even after the closing of the mine. This method has many disadvantages and is not recommended.

2.3 Technological Conditions

a) Naturally for the borehole methods in the hydro-genetic deposits we should continue with the well established pressurized method (continuous in time and space) – in closed cycle for the section and the cells.

b) For the shrinkage methods and incline of the deposit over 40-50^{o} the most effective method of operation is by continuous sprinkling of the ore – also closed cycle for the production blocks; when the dip is 40-50^{o} it is possible to apply the hydro-static method.

c) when the dip of the deposit is smaller than the application of hydro-static method could not be avoided (with complete overflow of the shrinked ore) cyclic mode for each block.

2.4 Production and Technological Schemes of the Uranium Geo-technological Mines

a) geo-technological mine for "borehole uranium production" – generalized schemes (Fig. 5 and 6).

b) geo-technological mine with underground leaching of blocks with shrinked ore – the principal technological scheme is similar and specific for the deposit.

c) geo-technological site for leaching of dumps – the scheme is principal - it is also analogue and specific according to the landscape and the method for leaching of the dump.

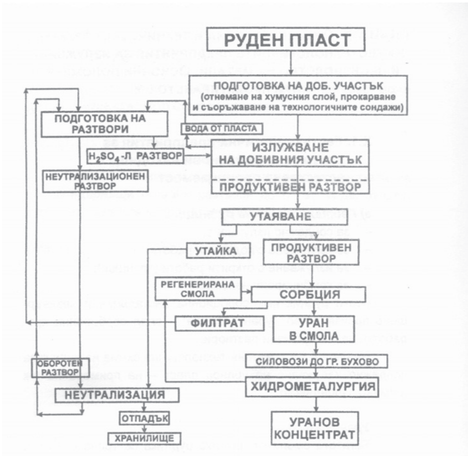
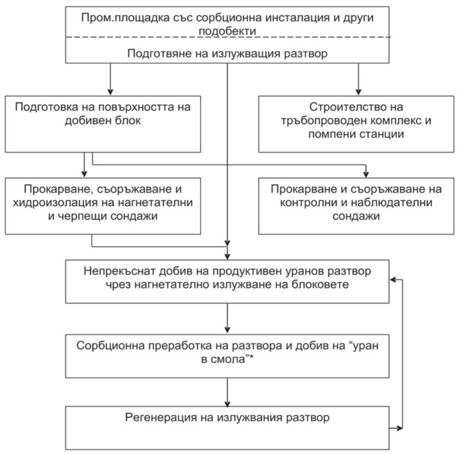


Fig. 5



*) Добитият "уран в смола" се десорбира и преработва до уранов концентрат U_3O_8 в завод "Металург" – гр.Бухово; регенерираната смола се връща в рудника и се използва многократно до закриване на обекта

Fig. 6

3 THE MAIN ECOLOGICAL TASKS, METHODS AND MEANS FOR SOLVING THEM IN GEO-TECHNOLOGICAL PRODUCTION OF URANIUM AND RARE METALS

Without any doubt every project for a concession and the respective solution within the range of the production of uranium and rare metals must ensure compliance with all requirements and regulations for the safety of the environment and the water in the areas of production activity.

A. Main ecological tasks in a mine for borehole production of uranium

1) Preparation of the surface of the production block or the entire uranium section by removing a soil layer with a thickness of 0.6-0.8 m., stockpiling and saving the entire quantity soil until the end of the exploitation of the block (section).

The reclamation of the area of the block (section) with the initial or improved soil is achieved by following method: a) during six months to one year is done liming with the objective to neutralized the area around the boreholes or the entire area of the block from eventual pollution with technological solutions; b) spreading of the stockpiled soil (technical recultivation) and c) biological recultivation by grassing, (afforesting) and fertilize during 1-2 years, chemical and radiation control in order to ensure meeting the ecological requirements for future utilization of the land.

In the past (until they were closed) in the Upper Trace Valley mines for borehole production of uranium were reclaimed approximately 4000 dka with improved quality of the soil.

2) Protection of the fresh underground water in the artesian basins of the Upper Trace Valley and in the Struma Valley when methods of borehole productions are applied: It is achieved by continuous hydro-insulation of the technological, control and monitoring boreholes in every production block (section) – it is performed by injecting behind the borehole casing of high quality fast drying concrete mixture with the highest strength; by high pressure water pumping the hydro-insulation is tested, in case of water loss the borehole is sealed and rejected.

By this method in the past was ensured the functioning of thousands of boreholes in all sections for borehole production of uranium and we did not allow contamination of the fresh water. The rejected boreholes were up to 2-5%.

3) Protection and cleaning of the residual geo-technological solutions after closing of the production block (section).

a) protection from flowing beyond the boundaries of the block (section) of the technological solutions, If there is such risk, usually it is achieved by building anti-filtration walls with the aid of a borehole net and pumping under high-pressure fast drying concrete mixture; the filtration wall could surround (insulate) the entire block (section) or only the permeable “windows’ and with the direction of movements of the water toward Maritsa, Tundzha, Struma and their feeders;

b) neutralization and cleaning of the remaining technological solutions after closing down of the production block (section): it is performed many times in closed cycle by sorption extraction of the uranium compounds (up to uranium in the resin), water dilution of the filtered solution and return to the block – to achieve the permissible levels for the remaining content of harmful metals and sulphates is required periodic selective sorption, sedimentation and safe storage – also until the acceptable ecological norms are achieved.

B. Main ecological tasks in a geo-technological mine with underground leaching of shrinked ore. They are similar and specific according to the landscape and the method for opening-out and development of the levels and the production blocks, according to the availability of water, the volume of the mine waters and their mixing with the productive solution, according to the tectonics and the presence of permeable faults and other factors.

The stage of the gradual slowdown of the leaching of the block (section and the entire level) and the cleaning of the remaining solutions to category II water runs the same way as for the borehole production of uranium - by selective sorption (liquid extraction).

C. Main ecological tasks in the sections with heap pile leaching and in the sections for leaching of by-product ore in artificial installations – trenches (basins). And here they are similar and specific according to the variants of the mining methods.

There are and are applied in the geo-technological mines abroad (and in the past – in Bulgaria until 1992-95) technical, physics-chemical and biologic methods and means for complete protection and reclamation in the initial natural quality and outlook and quality of land surface, the surface and underground water.

The experience in Bulgaria and abroad shows that the cost for protection of the environment and the water are completely bearable for the geo-technological companies, since they do not increase the production costs by more than 10-15%.

4 INNOVATIVE STRATEGY IN “PRODUCTION OF URANIUM AND RARE METALS IN BULGARIA” - PROSPECTS UP TO 2040-50

This strategy (generalized) aims at continuous increase of the complex technical and economic effectiveness and ensuring of ecologic safety during production of uranium and rare metals during the period of up to the years 2040-50 – at the highest innovative level.

The strategy includes the following main tasks:

a) to improve the existing and devising of new complex reagents for highly effective leaching of uranium and other metals;

b) to intensify the development of the mineral bio-technology with the objective of selection and industrial use of new types of tio bacteria and other micro-organisms in the technologies for leaching of uranium and rare metals.

c) the same eventually for new chemical oxidizers with acceptable toxicity and prices;

d) to improve the technological regimes for leaching of uranium and other metals with all types of mining methods;

e) to develop and implement new, including patented, variants of mining methods for leaching. And very important – borehole methods for depts. Up to 500-600 m;

f) to intensify the processes of leaching of uranium and other metals by physical methods: influence with electric field with direct and alternating current; periodic borehole-blasting and hydro-dynamic influence in the production blocks;

g) to improve the existing and development of new ion-exchange materials (resins and liquid extraction agents) according to the variety and multi-components of the produced solutions of uranium and rare metals with the objective of their selective extraction;

h) to improve the hydro-metallurgical processes and technologies (sorption, liquid extraction agents, etc.) for the production of uranium and rare metals;

i) to develop and implement automated systems for design, management and control of geo-technological enterprise for production of uranium and rare metals.

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The Correlation of the Technical and Technological Parameters During the Operation of the Bucket Whell Excavator $S_{ch}r_s$ 650

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ABSTRACT In the conditions when either on the production practice or on research there are numerous occasions when, the theoretical knowledges are insufficient, the only way with which we can have a satisfactory acknowledge of different technological phenomena or processes remains always the statistical planning of experiments. For a general experimental study of such a process we must take in consideration many indicators with the aim to overcome all the difficulties in the examination and optimization of such a processes according to the indicators. However, the problem solution that provides satisfactory reports of indicators can be achieved by implementing regressive analysis of experiment through comprehensive factor 2^k plans. Such plans can be applied with adequate success in correlation of the technical and technological parameters in the practice of superficial usage bucket wheel excavator. In this paper is presented correlation of technical and technological parameters during the work of excavator SchRs 650 for working conditions in overburden on open cast pit Kosova.

1 INTRODUCTION

Rational exploitation of the equipments with continual activity, depends on a great number of factors such as: constructive, technological, natural and climatic. The depending scale of these factors is defined in base of their correlation scale. The correlation of the technological and technical parameters in the surface exploitation represents a very complicated process, because it is depending from **case to case** and from the digging formation.Clay formations of Kosovo differ from the formations of the other deposits in terms of the physical-mechanical characteristics, if it considers the works' technology. That is why the main scope of this study is the establishment of the technical and technological scale of the correlations factors.

$$(Q_h,N,e,k_L) = f\,(S,h,v_k) \qquad (1)$$

According to orthogonal plans of the second order $\mathbf{2^k}$ through the mathematical experimentation theory with factors such as : slice thickness „S", cutting height „h" Fig .1 and the moving velocity of the working wing „v_k". Quite often, after the capital repairs or different modifications done with chain-bucket excavator excavators, is necessary to complete the capacity test or the technical promptness test for these equipments, in order to have a clear panorama of the repair argumentation. The study presented in this paper, presents a new method in this field aiming the study of the reports between the technical and technological parameters, enabling the enlargement of such parameters spectre. In this case it can be taken more information comparing with the other parameters carried out in different cases.

2 THE EXPERIMENT PLAN

The correlation of the technical and technological parameters, based on the data related to orthogonal coordinates of the second order 2^k,k aiming the increase of the exploitation scale of the capacity as well as decrease of the specific consumption of electric energy. In these cases are presented the mathematical models which express the dependence between these technical (Q_h,N,e,k_L), and technological parameters (S,h,v_k). For this reason the relation between the above mentioned parameters can be expressed through the formula

$$R=C\,S^{p_1}h^{p_2}v_k^{p_3} \qquad (2)$$

which represents the interlacement of the technological and technical parameters, enabling enabling the enlargement of such parameters spectre. In this case we can obtain more information comparing with the other parameters performed in different periods of time.

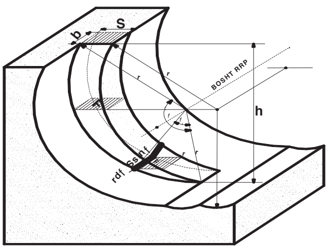

Figure1. Technological parameters.

In this case formula (1) can be transformed ina derivative form: R{Q_h,N,e,k_L} (2)The definition of the incognito parameters C,p1,p2 and p_3 and the formula (2) can be performed through the statistical elaboration of the results and in these cases we have in all :

$$N=2^k+n_o \quad \text{where:}$$

k-the number of the parameters , k=3 , $\boldsymbol{n_o}$-the number of the measurements in the centre of the experiment plan $\mathbf{n_o=4}$.After the calculations the result is:$N=2^3+4=12$[measurements].For more simple presentation of the parameters values in the measurement process of the hour capacity of the chain-bucket excavator "Q_h" and other parameters such as the power in the active axe "N", specific consumption of the electric energy „e" and the cutting force exercised „k_L", it must present the plan matrix of the experiment plan. Before presenting the matrix it must be performed the variables (S,h,v_k) codification through the values (-1,0,+1), presenting.

1 bottom limit value
0 average limit value
+1 upper limit value

The codification or transformation equation of the real values is performed through the formula:

$$x_i = 1 + 2\frac{\ln f_i - \ln f_{i\max}}{\ln f_{i\max} - \ln f_{i\min}} \qquad (3)$$

where:

f_i- presents the limit values of the variation interval of the parameters which means that if: $\mathbf{f_i=f_{imax}}$ then the coded value will be x_i= +1, meanwhile if $\mathbf{f_i=f_{imin}}$ the coded value will be x_i=-1. The plan matrix of the study is presented for real and coded values of the technological parameters considering as well some technical parameters as Q_h,N,e and k_L, with the proper values presented on the table 1. In order to avoid any eventual mistake, the choice of parameters is based on the random selection.

In the case of the capacity of the chain-bucket excavator measurement before the beginning of the study of the parameters relations we must define the initial values of the experiment plan. At the same time these values present the average value of the verification procedure parameters of the excavators ($S_{ch}R_s$650) capacity and in this case are totally completed 59 measurements The measurements are completed conform the technology projected from the constructer who guarantees a capacity 2000[m^3/h]. The measurement values of the above mentioned parameters are carried out considering the average values of the parameters

- Slice thickness **S=0.45[m]**
- Cutting height **h=4.5[m]**
- Wing speed $\mathbf{v_k=19[m]}$

Thus, from these parameters values can be asked the question if we can preliminary determine the research steps or they can be predetermined. In each case they must fulfil the condition:

$$f_{mes} = \sqrt{f_{max} f_{min}} \tag{4}$$

Because in this case is not given the variation step, the research step for each parameter can be elected according to the will, anyway in this case it must be taken into account that the parameters that are supposed to effect the process, must be undertaken complete the smallest step meanwhile for the factors that slightly effect the process must be undertaken the biggest step. According to the above principle the steps of the parameters are:

-Slice thickness h_1=0.15
-Cutting height h_2=1
-Wing speed h_3=1

In these conditions the parameters limits will be:

1: slice thickness
1 the bottom limit S=0.3[m]
2 the upper limit S=0.6[m]

2: Cutting height
1 upper limit h=5.5[m]
2 bottom limit h=3.5[m]

3: Wing's movement speed- limit
1 bottom limit v_k=18[m/min]
2 upper limit v_k=20[m/min]

Table 1. Plan Matrix of the 2^k experiment.

No	Real variables			Coded variables				y_i
	S	h	v_k	X_0	X_1	X_2	X_3	
1	0.3	3.5	18	+1	-1	-1	-1	y_1
2	0.45	4.5	19	+1	0	0	0	y_2
3	0.6	3.5	18	+1	+1	-1	-1	y_3
4	0.3	5.5	18	+1	-1	+1	-1	y_4
5	0.45	4.5	19	+1	0	0	0	y_5
6	0.6	5.5	18	+1	+1	+1	-1	y_6
7	0.45	4.5	19	+1	0	0	0	y_7
8	0.3	3.5	20	+1	-1	-1	+1	y_8
9	0.6	3.5	20	+1	+1	-1	+1	y_9
10	0.45	4.5	19	+1	0	0	0	y_{10}
11	0.3	5.5	20	+1	-1	+1	+1	y_{11}
12	0.6	5.5	20	+1	+1	+1	+1	y_{12}

3 MEASUREMENTS OF THE TECHNICAL PARAMETERS

For the accomplishment of the measurements faze of the hour capacity „Q_h" active power „N" with which is performed the digging process, specific electric energy consumption „e" and the cutting specific force „k_L" exercised of the working wheel in the digging rocks must be also defined. The hour capacity „Q_h" for these types of excavators has been made through photometric measurements and weighing equipments of the TT227N, UTEZ Celje, made in Slovenia . Figure below:

Figure 2. Weighing equipment type TT227N.

The active power working wheel RRRR is defined by multiple equipments METREL type (INSTALTEST 61557 MI 2087)

Figure 3. Multiple equipments METREL used for active power measurements.

Specific electric energy consumption„e" express the ration between active power of the working wheel and the hour capacity of the excavator as below:

$$e = \frac{N}{Q_{ef}} \quad [\text{kWh/m}^3] \qquad (6)$$

Specific cutting force „k_L" in this case has been calculated through the formula:

$$k_L = \frac{335.2\eta N - 0.913 Q_{ef} D\gamma g}{\sqrt{Q_{ef} D n_{zb} \varphi}} \quad [\text{daN/cm}] \quad (7)$$

Table 2. The technical parameters achieved by the measurements.

No	Technical parameters „y_i"			
	Q_h [m^3/h]	N [kW]	e [kWh/m^3]	k_L [daN/cm]
1	1160	467	0.40	44
2	2200	663	0.30	92
3	2300	679	0.30	103
4	1795	592	0.33	69
5	2200	663	0.30	92
6	3450	853	0.25	130
7	2200	663	0.30	92
8	1300	496	0.38	56
9	2500	712	0.28	112
10	2200	663	0.30	92
11	2000	629	0.31	78
12	3900	900	0.23	145

4 THE ANALYSIS OF THE MEASUREMENT RESULTS

The mathematical analyse of the technological parameters during the optimisation of the equipment capacity, must be done aiming the perform of the mathematical models, which express the nature relation of technical and technological working parameters of the bucket and chains excavators .Such mathematical models represent the possible values of technological parameters which have been reached during the verification capacity of bucket and chains excavators, SchRs 650 type.In this case the proper surface to have a linear shape as below:

$$y = b_o + \sum_{i=1}^{n=3} b_i x_i \qquad (8)$$

Based on the fact that in this case is asked to approach the regress surface, in which are included different factors as: slice thickness "**S**", cutting height **"h"**, working wing movement speed of the excavator "$\mathbf{v_k}$", expressing the regress surface parameters with : x_1=**S** , x_2=**h** and x_3= $\mathbf{v_k}$, its form will be:

$$y = b_o + b_1x_1 + b_2x_2 + b_3x_3$$

which must be transformed in logarithmic function as below:

$$\ln R = \ln C + p_1(nS + p_2(nh + p_3 \ln v_k)) \qquad (9)$$

Variables, which take part in this logarithmic function, are:

$y = \ln R \quad x_1 = \ln S$

$x_2 = \ln h \quad x_3 = \ln v_k$

Meanwhile the coefficients next to the variables are substituted with transformation equation as below:

$b_0 = \ln C \quad p_1 = b_1$

$p_2 = b_2 \quad p_3 = b_3$

So, as result the expression of the regress surface will be transformed as below:

$$R = C\, S^{p_1} h^{p_2} v_k^{p_3}$$

According to the formula (3), regarding to variables $\mathbf{x_i}$ is formed the matrix of the experimentation plan 2^k+n_o, in the case when k=3, the matrix of the experimentation plan takes this form meanwhile the order of the factors can be done randomly.In base of the orthogonal characteristic, which must complete the matrices, it is enable that the regress equation coefficient "$\mathbf{b_i}$", can be defined through the formula below:

$$b_i=\frac{1}{p}\sum_{n=1}^{p} x_{iu} y_u \qquad (10)$$

After the definition of the regress coefficients (mathematical model) and based on the pre-determinate conditions is done the transformation of the equation in its initial form, through the formula below:

$$p_i=\frac{2b_i}{\ln\left(\frac{f_{i\max}}{f_{i\min}}\right)}$$

$$p_o=\left|\left|\sum_{n=1}^{3} b_i\right|_{i=0,1,2,3}-\sum_{n=1}^{3} p_i \ln(f_{i\max})\right|_{i=1,2,3} \qquad (11)$$

Through the definition of these parameters can be go beyond to the equation (2) profiting the correlations between technical and technological parameters. Following this, according to the F criteria the evaluation of the statistic significance, comparing the distribution in the zero point, has been done.

$$F=\frac{s_i^2}{S_E^2}>F_t^* \qquad (12)$$

Model parameters distribution is calculated according the following ration:

$$S_i^2=\frac{s_i^2}{f_i} \qquad (13)$$

Meanwhile the sum of the second exponent is defined following the formula:

$$Sb_i=b_i\sum_{n=1}^{p} x_{iu} y_u = N_i b_i^2 \qquad (14)$$

where i=0,1,2,3.There are treated two cases for the parameters as follow:

i=0 N=12

i=1,2,3 N=8

For this case are secured the freedom degrees "$\mathbf{f^i}$", meanwhile the distribution around zero point is calculated as below:

$$S_E^2=\frac{S_E}{f_E}$$

The sum of the second exponent is defined following the formula:

$$S_E^2=\sum_{n=1}^{4}(y_{ou}-y_o^2)=\sum_{u=1}^{4} y_{ou}^2-\frac{1}{n_o}\left(\sum_{u=1}^{4} y_{ou}\right)^2 \qquad (15)$$

In this case the freedom degrees are calculated following the formula: $\mathbf{f_E=n_o-1}$. When the certain level is "α" and $F_l>F_t^*$,the parameters "$\mathbf{b_i}$" are significant, in the contrary cases these parameters must be excluded from the mathematical model. Each mathematical model, that describes a working process of the systems or the represents an event must define the confidence interval according to the criteria "**F**", for the certain level "**α**".

$$F_l<F_t^*, \qquad (16)$$

As the central point is at the same plan in which the experiment is repeated $\mathbf{n_0}$-times, then the F calculated value –Fi , necessary for the definition of the certain interval of the mathematical model is expressed by the following form:

$$F_{l_{LF}}=\frac{S_M^2}{S_E^2} \qquad (17)$$

The difference between the experiment values "$\mathbf{y_i}$" and the calculated values "$\hat{y}_i$" is expressed as follow:

$$S_M^2=\frac{S_{LE}}{f_{LE}}=\frac{S_R-S_E}{f_R-f_E} \qquad (18)$$

Or if it is expressed according to the disintegrated equation will have the following dispersion:

$$S_M^2=\frac{1}{f_R-f_E}\left\{\sum_{u=1}^{12}(y_u-\hat{y})^2-\left(\sum_{u=1}^{4} y_{ou}^2-\frac{1}{n_o}\left(\sum_{u=1}^{4} y_{ou}\right)^2\right)\right\}$$

Meanwhile the measurements results dispersi with a medium level is calculated following tl formula:

$$S_M^2=\frac{1}{f_E}\left\{\sum_{u=1}^{4} y_{ou}^2-\frac{1}{n_0}\left(\sum_{n=1}^{12} y_{0u}^2\right)^2\right\} \qquad (19)$$

For this reason the **"F"** criteria value, are preliminary selected from the proper tables with the solution with the level of the certain level.

$$f_{LE}=f_R-f_E \qquad (20)$$

and $f_E=n_0-1$

The regressive and dispersive analyse and the parameters correlation is done analyzing the technological parameters which have an important indication during the definition of the technical parameters reached during the capacity determination phase. In this stage can be found the correlation between these parameters and using the formula (2), can be given the correlation equations of different parameters such as:

Hourly capacity “Q_h”,

Active power of the working wheel , “N”,

Electric energy specific consumption “e”,

Specific cutting force “k”.

The graphic „3D” of the technological and technical parameters correlation for each case is performed using MATLAB software

- Hourly Capacity:

  $Q=117\ S^{0.287}\ h^{0.274}\ v_k^{0.934}$

- Active Power:

  $N=186.5S^{0.334}\ h^{0.36}\ v_k^{0.343}$

- Specific electric energy consumption:

  $e=1.59\ S^{0.04}\ h^{0.08}\ v_k^{-0.590}$

- Cutting specific force:

  $k_L=13.64S^{0.383}h^{0.411}v_k^{0.407}$

Below are presented the graphics interpretation of the profited correlation for the cutting height h= h=4.[m]

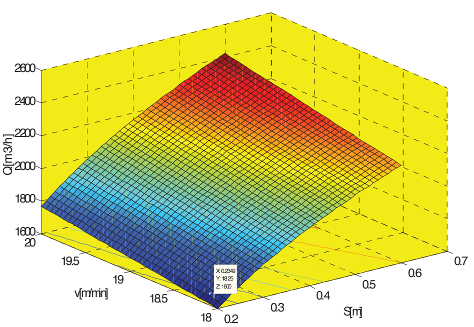

Figure 4. Hourly capacity correlation „Q_h” from slice thickness „S” and the working wing movement speed„v_k”.

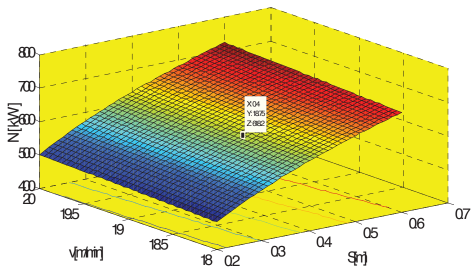

Figure 5. The correlation of the active power working wheel „N” from the slice thickness „S” and moving speed of the working wing„v_k.

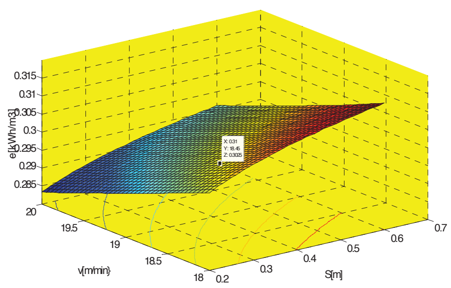

Figure 6. The correlation between the specific electric energy consumption „e” , slice thickness “S” and the working wing movement speed „v_k” .

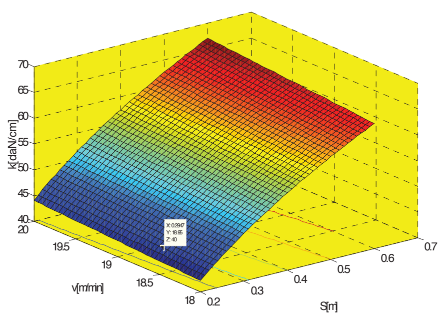

Figure 7. The correlation between the cutting force„k_L” slice thickness „S” and working wing movement speed„v_k”

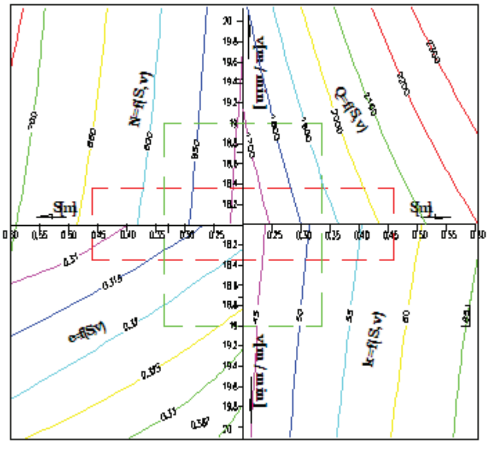

Figure 8. The nomogram of the technical and technological parameters correlation for the cutting height h=4[m].

6 CONCLUSIONS

In base of profited diagrams and as a result of the result elaboration through the regressive and dispersive analyze according to the orthogonal plans „2^k'' of experiment with technological parameters of the working process of the rotoric excavators, in the case of of their correlation with the technical parameters an come to this result: the profited values of the technical parameters according to the correlation equation are are almost the same with the values profited from the verification process of these excavators capacities if we own the tecnologaical parameters from the interval the presented variation in this study. The correlation of the technical and technological parameters based on such scientific studies presents a safe basement for the prediction of the hour capacity and other indicators for these excavators working conditions ($S_{ch}R_s$650) in MS.KOSOVA, based on the compatibility of mathematical model solution.

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Payload Estimation of a Walking Dragline - A Case Study

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ABSTRACT This paper presents a case study carried out on a 31 m^3 capacity walking dragline for determining the energy consumption standing for per ton of rock in the bucket. Digging cycles of sucessive full and empty buckets with identical trajectories, under identical operational and digging configuration, current and voltage armature feedback signals of drag, hoist and swing dc motors are recorded and processed. Armature signal-time curves of empty and full digging cases are drawn. From these plots effect of digging and load on the armature feedback signal magnitudes can be seen qualitatively. On the other hand, the tremendous increase in the consumption of energy compared to the empty bucket case can be witnessed quantitatively from the energy consumption tables calculated. The bucketful rock was dumped on the ground and transported to a weigh-bridge for correlating energy consumption per ton of rock in the bucket.

1 INTRODUCTION

The importance of knowing payload in the bucket is an invaluable information. It is the indicator of how dragline bucket performs. The target is being having the maximum payload in the shortest digging time with the minimum digging energy within the limits of rated suspended load. The higher payloads imply better bucket fill factors and optimum fragmentation of the rock in the bench. This in turn implies that drilling and blasting performance is satisfactory. In short it is a measure of overall dragline productivity, the aim being to move bigger amounts of (weight) broken rock from the bench to the spoil area to uncover more of the coal seam. The case was conducted on a 31m^3 capacity dragline in Tuncbilek surface mine.

2 PAYLOAD MEASUREMENT

The conventional method is weighing a bucketful of material on the nearest weighbridge. It is an expensive and time consuming method especially if the bucket size is large. The ideal system is monitoring payload by an onboard system by installing load cells on bucket rigging and/or hoist ropes and a data acquisition system. Another system is based on energy consumption levels of the motors utilizing voltage and current feedback signals of D.C. motion motors. If higher accuracy is required the load cell system is better. Loadcells with strain gauges are mounted on bucket's rigging at the connection point of hoist chain to hoist rope Deslandes, *et al*. (1990) & Anon b (2003).

2.1 Principle of Measurement

As device a laboratory type monitor was used. The dragline is equipped with control cabinets that house a series of control frames for each work-motion motors. Proper test points on each control frame provide a clean ± 15 VDC signal for both armature voltage and current.

It was assumed that by monitoring two successive cycles with empty and full bucket under exactly the same bench, swing and dump and bucket trajectory parameters would establish a base for a payload estimation. The first one was being with no load in the bucket pretended to dig and completed the cycle. The successive one was a real digging cycle with material filled in the bucket followed the same trajectory of the previous cycle.

3 PAYLOAD VERSUS FEEDBACK SIGNALS

When the recorded signals of the so called empty and full bucket digging cycles are plotted, impact of payload in bucket on energy consumption level is seen qualitatively through signal amplitutes. Upon the analysis of the energy consumption parameters of the two successive cases, it is possible to follow how much payload this extra electric energy consumption corresponding to.

3.1 Armature Signal –Time Curves of Empty and Full Digging Case

Armature signal-time plots of the empty and full digging cases are given in Figures 1-2 and Figures 3-4, to illustrate impact of payload in the bucket. The calculated energy parameters are given in Table 1. The differences are evident between energy consumption levels of the two cases from the amplitudes and raggedness of signals.

The reason why swing feedback voltage signals are plotted against time is to identify the segments of a cycle, and the times elapsed for the segments. The left hand side swing voltage curve indicates the segment of the cycle where bucket is back to the face to digging position, and the time elapsed for this segment. At start of digging position swing action stops and swing signal reaches zero value. At the digging segment, a very small or zero swinging action of the bucket is involved, and when it is filled swinging action starts (swing signal value rises again from zero) to the dumping area with full bucket. The digging segment of the cycle is identified on the plot where there is no or negligible swing action. The right hand side section of the curve indicates the cycle's segment where the filled bucket moved to the dumping area. At this segment of the cycle swing feedback signals are negative due to the fact that direction of the swing movement is changed resulting a change in the polarity of the swing signals.

Table 1. Energy consumption parameters of empty and full bucket cases of 31 m^3 capacity walking dragline, Özdoğan (2002).

Cycle Time (s)	Cycle Energy (kWh)	Normalized Energy (kWh/s)	Specific Energy (kWh/m^3)	Drag Energy (kWh)	Hoist Energy (kWh)
64.19	7.49	0.12	0.24	2.53	4.42
64.43	18.07	0.28	0.59	8.82	8.70

3.2 Discussion

The bucket weight of the equipment with rigging was 37 metric tons, and the weighed payload was 59 metric tons. In empty digging the tare weight of the bucket was responsible for the values of the dragging and hoisting signals, whereas in full digging the bucket load including the tare weight were responsible for the values of the signals.

Estimated gross load for this specific case was 11.11 metric tons per kWh of hoist energy consumed. In order to obtain payload figure the tare weight has to be deducted, of course. The payload can be estimated on the full cycle energy consumption, as well. Full cycle energy consumption's one kWh stands for 5.35 metric tons of gross bucket load. In both cycles, swing made towards the new high wall to dump the bucket load on an accessible ground surface for easier loading and transporting. Eventhough the cycle times are about the same 64.19 and 65.43 seconds, energy consumption parameters differ dramatically, Table 1.

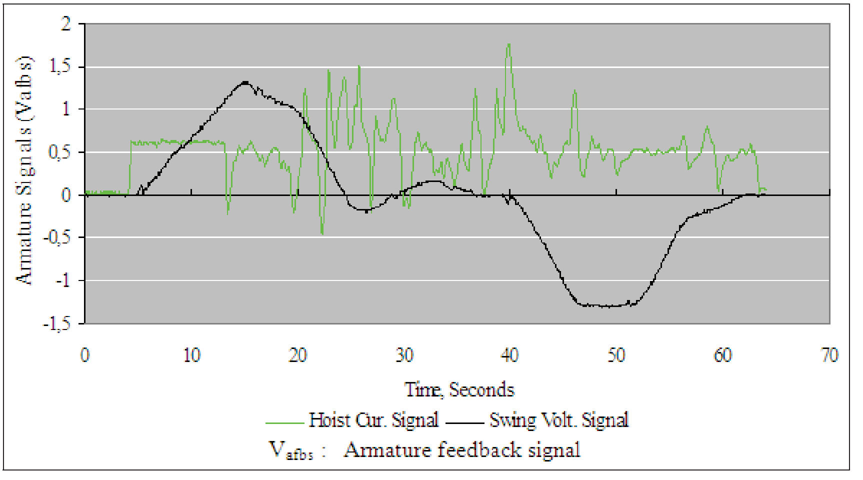

Figure 1. Time vs. hoist current - swing voltage signals of empty bucket digging cycle, Özdoğan (2002).

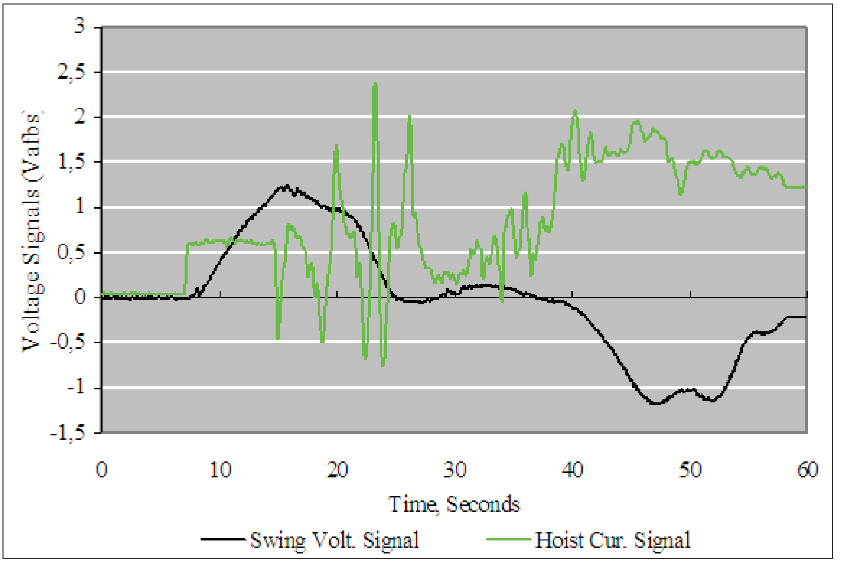

Figure 2. Time vs. hoist current - swing voltage signals of full bucket digging cycle, Özdoğan (2002).

Figure 3. Time vs. drag current - swing voltage signals of empty bucket digging cycle, Özdoğan (2002).

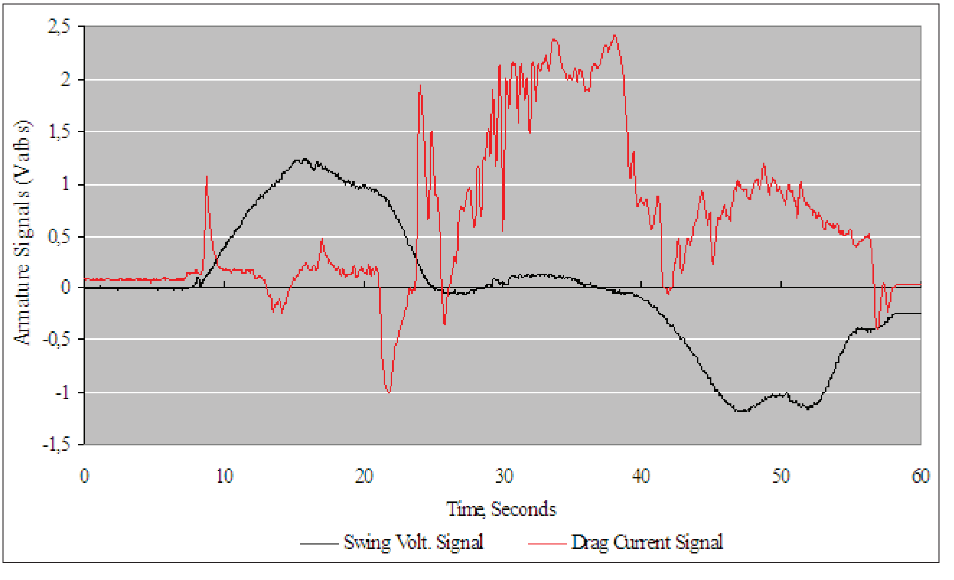

Figure 4. Time vs. drag current - swing voltage signals of full bucket digging cycle, Özdoğan (2002).

4 CONCLUSIONS

Importance of measuring the payload is obvious in order to assess and monitor the performance of the dragline and the operator and blasting and the fill factor quantitatively. As it is seen from the case that it is possible to monitor the payload indirectly by monitoring armature feedback voltage and current signals of the dc motion motors of the equipment and calculating the hoist motor energy consumption or total cycle energy consumption of dragline for every cycle. However, according to the literature the accuracy of weighed load by this system is not very accurate, it may vary ± 10 % - 20 % of the true weight, Anon a (2003). Therefore, the recent trend is to utilize load cells and strain gauges for more accurate direct load weighing. In load cell case the error is in the range of 5%, Anon a (2003) & Anon c (2006) & Anon d (2009).

New generation walking draglines are normally equipped with onboard load monitoring systems. Irrespective of having on board monitoring system or not, dragline bucket load should be checked and measured by any means available at the site. Bucket loads should be inspected in a periodic manner qualitatively and quantitatively. These inspections may give hints on bucket performance, fill factors, changing rock density, rock fragmentation, bench geometry, equilibrium of the equipment and allowable suspended load etc. especially when equipment is moved to another slice and/or panel.

The target is to increase the paylaod in the bucket in order to increase the performance of the walking draglines which can only be achieved and controlled by monitoring the weight of the load in the bucket frequently if possible continuously.

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Modernization of the Process "Flotation of Molybdenum" in "ELLATZITE –MED" AD, Bulgaria While Incorporating a System for Management Using Information

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ABSTRACT The high technological requirements toward the process flotation of molybdenum imposed the implantation of new system for management using information. We enlace the whole technological flow through modern machines for control and management. We describe the capacity and the possibilities of the central station for operators and we give photos of the main screens.

1 INTRODUCTION

Molybdenum as raw material has wide acceptance in the metallurgic industry and it makes it valuable for extraction.

Its low content in the ore extracted in "ELLATZITE" –MED imposed the introduction of a new contemporaneous management system that uses information to be able to follow precise and flexible technological process.

2 STRUCTURE OF THE SYSTEM

The company KASTIVA gained the order for constructing a system for management of the technological process. She offered a system that satisfy maximum requirements of the person that needs it.The system for management using information is decentralized and englobe totally the flotation process and the operations for preparing and transportation of flotation reagents.The management is done using programmable logic controllers –PLC. The visualization of the process, the input, the archivation and the processing of the information is done from operator station with installed software of company SIEMENS. One of the main screens is shown on Figure 1.

The connection with distant sub –objects is done using Profibus DP communication.

3 CONTROLLED PARAMETERS

3.1 In Flotation Process

3.1.1 Measure of level of pulp in flotation machines

This is one of the main parameters in the technological process. For this purpose are used ultrasound sensors with high accuracy.

3.1.2 Measuring of p-ha and ORP of the pulp in flotation machines

The combined measuring device with electrode constructed for very aggressive medium of company EMERSON showed a stable work with necessary maintenance.

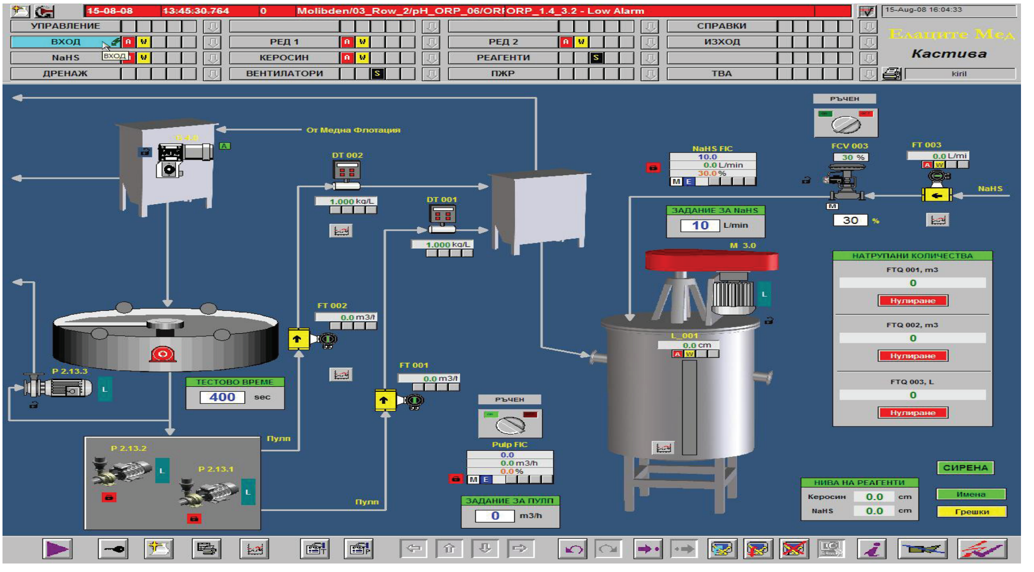

Figure 1.

3.1.3 Control of the state ON /OFF of all machines that take part in the technological process

Discrete signals follow the state of each mechanism and register on the operator station the situation of it and direct the operator to the causes or the conditions to execute the tasks. The equipment of the whole technological scheme start working one after other. The state of the machines and the values of the technological parameters is visualized on Figure 2.

3.1.4 Constant measuring of the density of material for flotation

Precisely calibrated radio –isotope device for density measuring Gammaplot MFMG of company Endress + Hauser measures the weight of the volume of the input pulp in the process. This is an important parameter for the setting of the technological regime of flotation of molybdenum.

3.1.5 Constant measure of use volume of the material for flotation

This is a classical method for such kind of products using device with induction for measure the use of company Endress+Hauser and these devices are equipped with sensor system, resistant to abrasion.

3.1.6 Stationary system for control and signalization when there are harmful emissions of gases

A system with sensors for harmful gases controls the micro climate in the main production. Depending on the registered values of the harmful emissions are activated automatic the necessary fans.

3.1.7 Control of the reagents added to the process

The level in the buffer and collector vessels for flotation reagents is controlled and also the quantity of these reagents in flotation process.

3.1.8 Control of the levels in the intermediate vessels between separate operations

Ultrasound sensors control the levels in the intermediate vessels for material between the different operations in the process.

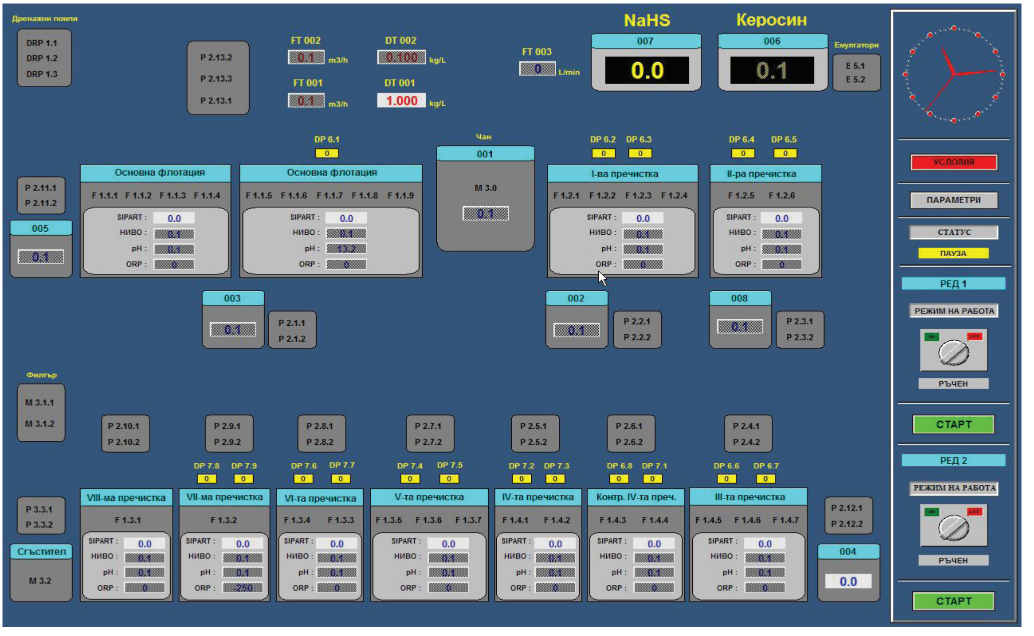

Figure 2.

While Preparing of the Reagent Na HS

Control of the state ON /OFF of all pumps

Control of the state OPEN /CLOSE of all automatic valves

Control of the level of three buffer tanks for solution of NaHs

3.2 During the Preparation of Kerosene Emulsion

3.2.1 Measuring of the quantity of water and kerosene

A device for measuring the use of water made from company Endress –Hauser controls the quantity of the water and corisole device of company Yokogawa measures the quantity of kerosene. A possibility for fulfill to prepare an emulsion with content of kerosene that corresponds to the technological requirements and guaranteed high accuracy.

4 EXECUTED CONTOURS FOR AUTOMATIC MANAGEMENT

4.1 Management of Flotation process

4.1.1 Automatic management of the levels of pulp in flotation cells

Using pinch –valves with positioner and programmable PID regulator is obtained a precise maintenance of the levels in flotation cells according preliminary determined tasks.

4.1.2 Automatic regulation of the used volume of the pulp for flotation

PID regulator that directs the pump for material using frequency inverter stabilizes the quantity of the pulp and tends to maintain it relatively constant. Operator station provides a lot of possibilities for setting the PID regulator and this is a premise to get precise while managing the flotation process. The screen for the setting of the regulator is shown in Figure 3.

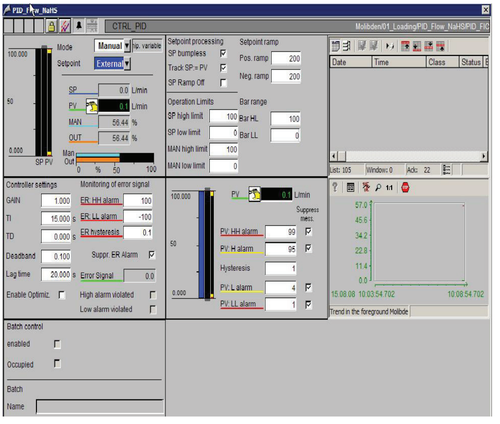

Figure 3.

4.1.3 Automatic dosage of flotation reagents

Membrane dosage pumps assure the necessary quantity of reagents in molybdenum flotation process. They can be managed from their own control panel and also from operator station. A possibility for stabilization of ORP is fulfilled when changes the quantity of the solution of NaHS.

4.1.4 Maintain of constant level in collector tanks for the reagents

Automatic filling of the collector tanks with preliminary prepared reagents and maintain of a constant level in them. It is guarantee that the vessels for material will not be overfilled and will not stay empty and this will not influence negatively the technological process.

4.1.5 Automatic maintenance of level in intermediate vessels

The purpose while doing the regulation of level in the intermediate vessels for material through PID regulator and frequency invertors for management of the speed of the corresponding pump is that the vessels will not be overfilled with material and will not stay empty. The change in the speed of the pumps is slowly to disturb minimally the process.

Constructed this way the management system for information is used regularly and execute its functions in the management of the process of molybdenum flotation.

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  ISBN 978-9944-89-782-2

The Evaluation of the Inyoite Mineral as a Neutron Shielding Material

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ABSTRACT Inyoite is a colorless and transparent borate mineral that occurs in the form massive granular or spherulitic aggregates in borate deposits. Inyoite is originally found in Turkey in the regions of Kırka and Bigadiç, Death Valley in U.S., and in the western side of the Kazakhstan.

In this study, depending on the ability of neutron adsorption of inyoite mineral, the structural and physical properties are examined. First of all, the sieve analysis of inyoite mineral is made then five different particle sizes are selected, prepared and X-Ray Diffraction of inyoite minerals is made for structural analysis. After that, boronoxide contents of inyoite minerals are determined experimentally. After these examinations X-Ray Florescence analysis is made for the determination of Ca, Zn, and Fe and Ar contents.

Thermal analysis of inyoite minerals is performed for obtaining the energy and weight changes with temperature. For this aim, the method of Differential Thermal Analysis and Thermal Gravimetry is used. Afterwards, inyoite mineral is tested in the Howitzer equipment for the nuclear absorption and confront with the colemanite mineral that is mostly used in the experiments at this field of study.

1 INTRODUCTION

1.1 Boron Minerals

Turkey has the 72.2% of boron minerals that are present in the World. Approximately World's total reserves are 1.2 billion tones. Turkish borate deposits contain largest reserves in the world, which can sufficiently meet world's needs for many years. Turkey has also the largest borax, ulexite and colemanite reserves in the world. All the countries are dependent upon colemanite and ulexite reserves of Turkey.

Many countries around the world, especially USA and France, use boron compounds as a shield material in nuclear reactor technologies. A lot of researches have been made in the area of nuclear shielding, both in Turkey and the world. (Helvacı 2004; Kılınç et al., 2001, Köklü et al. 2003, T.R. Prime Ministry Ninth development plan 2006, Uyanık 2006).

1.1.1 Application fields

The major usage areas of boron minerals make up a wide diversity, including glass, ceramic, nuclear, space-aviation, electricity-electronics-computers, metallurgy, energy, transportation, textile, travel, cosmetics and chemistry (Sarıhan 2006).

1.1.2 Inyoite mineral

Inyoite ($Ca_2B_6O_{11}13H_2O$) is a colorless and transparent borate mineral that occurs in the form of massive granular or spherulitic aggregates in borate deposits. The structure of the mineral consists of two $BO_2(OH)_2$

tetrahedrons and a BO_2OH triangle linked by calcium and hydroxyl-hydrogen bonds. Inyoite alters by partial dehydration to meyerhofferite and colemanite, both of which are used as raw materials for borax and other boron-related products.

Inyoite has a molecular weight of 177.60 g/mol and has a density of 1.87 g/ml.

Inyoite is originally found in Turkey in the regions of Kırka and Bigadiç, Death Valley in U.S., and in western Kazakhstan. (Helvacı 2004, Köklü 2003 et al., T.R. Prime Ministry Ninth development plan, 2006).

Figure 1. Structure of the Inyoite

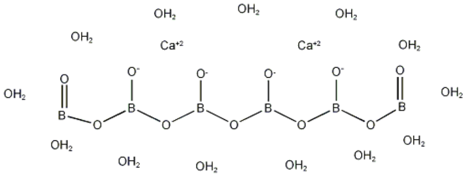

Figure 2. Shape of the Inyoite

1.2 Studies that are made using Boron Minerals

Many countries around the world, especially USA and France, use boron compounds as a shield material in nuclear reactor technologies. A lot of researches have been made in the area of nuclear shielding.

Materials like boron carbide, boron oxide, iron, lead and bismuth are used in shielding studies made in the world.

The some examples of the studies that are made in Turkey are given below:

Gündüz (1980), inserted vitrified and powdered colemanite ore in aggregate form to concrete.

Yarar (1994), provided a slimming of approximately 10 cm in the shield thickness, with an addition of 10% colemanite by concrete weight, in the design of the biological shield of a power reactor.

Elbeyli et al. (2003), stated that Hemihydrate borogypsum may also be useful for decreasing the radioactive permeability of concrete due to the boron content.

2 EXPERIMENTAL

In this study, the inyoite mineral is supplied from the Kırka region of Turkey.

2.1 Preparation and Determination of the Minerals using XRD

Inyoite Minerals, which are supplied from the Kırka region, are selected from the ore sector of the mine and used directly without making any pretreatment.

Firstly sieve analysis is made according to the sieve sizes of 140, 200, 500 and 560 microns. Then prepared minerals are identified by the equipment of Philips PANanalytical X-Ray Differactometer (XRD) (Kıpçak 2009).

2.2 Determination of the B_2O_3 Content of Inyoite

Due to the interaction principle of the boron minerals with the thermal neutrons, the B_2O_3 amounts of the boron minerals used should be known.

During the experiments carried out, 1 g of sample material taken from Inyoite minerals are dissolved in 3 ml of 37% HCl and then diluted to 100 ml. Pure boric acid (H_3BO_3) at the same conditions was used as the reference material. B_2O_3 amounts were determined through an acid-base titration with 0.1 N NaOH, in a METLER DL-25 titrator. (Kıpçak 2009)

2.3 Determination of the Fe, Zn and Ar Contents of Inyoite by X-Ray Fluorescence Spectroscopy (XRF)

To determine the other elements present in the mineral with the aim of observing the source of the radioactivity formed in boron minerals under neutron flow, 150 mg of powdered sample with a particle size less than 140 mm was mixed with 50 mg pure wax® ($C_3H_8O_7N_7$) that is used as an adhesive, in an agate mortar for five minutes. Then, pellets are formed in a 37 mm mould set with a 40 MPa hydraulic press.

For the measurements, a silicon drift detector in a Minipal4 model instrument of PANalytical brand is used. The separation power of the silicon detector is between 4 kV-30kV. In this instrument, analysis is made for elements with an atomic number between Na and U, making use of the characteristic X-rays (Kıpçak 2009).

2.4 Thermal Analyses of the Inyoite

With the Differential Thermal Analysis and Thermal Gravimetry (DTA-TG) experiments, the investigation of the energy and weight differences with the temperature change present in the boron minerals are aimed.

The instrument used is Perkin Elmer brand Diamond TG-DTA Termogravimetric / Differential Thermal Analyzer Model.

The analyses are made under inert Nitrogen atmosphere, the temperature change is set to 10^{o}C per minute and the temperature range is between 30^{o}C and 900 ^{o}C. (Kıpçak 2009)

2.5 Howitzer Neutron Permeability Experiments

The neutron permeability experiments in Howitzer were carried out with a thermal flux of approximately 10^4 n/cm^{2}s. In these experiments, the source detector space was kept constant at 5 cm and no space was left between the pellets and the detector. The schema of experiments is shown in Figure 3.

The Howitzer Neutron Permeability Experiments are carried out only at the particle size of -500, +200 microns because of the small amounts of other particle sizes. Two experiments are conducted with different pellet thicknesses of 0.880 cm and 1.307 cm.

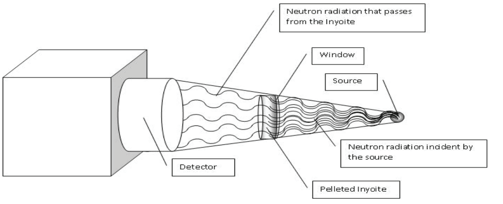

Figure 3. Schematic shown of the neutron radiation through the pelleted Inyoite

3 RESULTS

3.1 XRD Results of the Inyoite

The identification results of the Inyoite minerals of different particle sizes are shown in Table 1 and Figure 4 through 8.

Table 1. pdf card numbers and names of the Inyoite minerals.

Particle Size (micron)	Pdf Number	Mineral Name
-140	01-073-0413	Inyoite, syn
+ 140, -200	01-073-0413 01-076-0669	Inyoite, syn Calcium Borate Hyd.
+200, -500	01-073-0413 01-075-0640	Inyoite, syn Calcium Borate Hyd.
+500, -560	01-073-0413 01-076-0669	Inyoite, syn Calcium Borate Hyd.
+560	01-073-0413 01-076-0669	Inyoite, syn Calcium Borate Hyd.

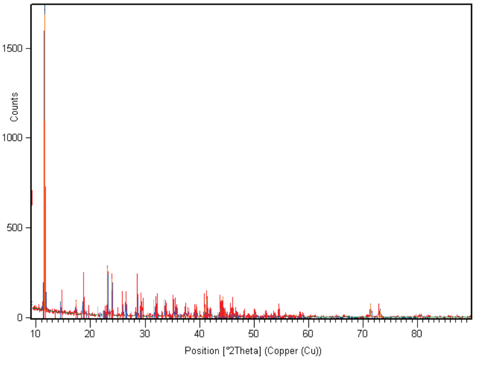

Figure 4. XRD pattern of -140 micron Inyoite .

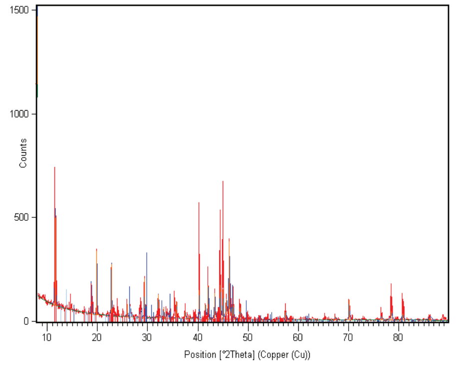

Figure 5. XRD pattern of +140,-200 micron Iyonite.

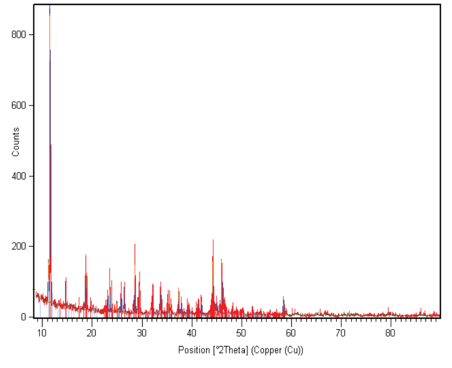

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Figure 6. XRD pattern of +200, -500 micron Inyoite.

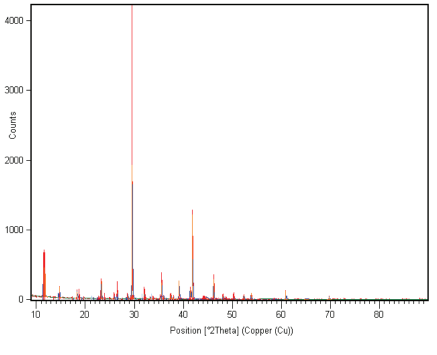

Figure 7. XRD pattern of +500, -560 micron Inyoite.

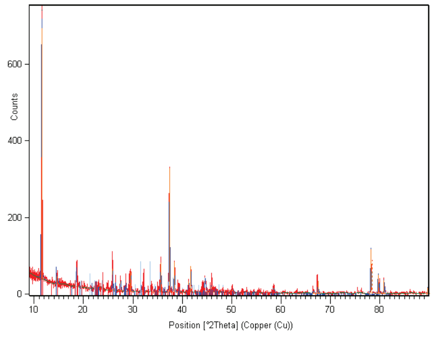

Figure 8. XRD pattern of +560 micron Inyoite.

3.2 B_2O_3 Contents of the Inyoite

The B_2O_3 contents of Inyoite minerals with different particle sizes are given in the Figure 9.

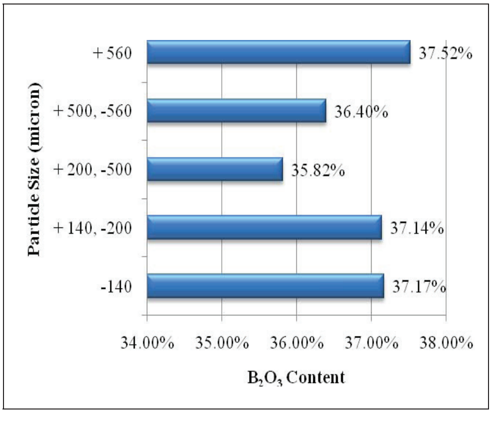

Figure 9. B_2O_3 Contents of the Inyoite minerals.

3.3 XRF Results of the Inyoite

The XRF results of the Inyoite, which has a particle size of -140 microns, showed no content of any Fe and As but it had Zn content of 154.1 ppm.

3.4 DTA/TG Results of the Inyoite

DTA/TG results of the Inyoite minerals are shown at the Figure 10 through 14 and calculated temperature peaks and weight losses are represented at the Table 2.

Table 2. Calculated temperature peaks and weight losses of Inyoite.

		Peak 1	Peak 2	Peak 3	Peak 4	Peak 5
-140	T_i (°C)	68.84		396.74	747.28	820.65
	T_p (°C)	133.15		406.70	776.27	841.49
	T_f (°C)	202.90		417.57	793.48	860.51
	ΔH (J/g)	289.20		1.19	-14.79	-9.03
	Δm (%)	36.564		5.057	**Total**	**41.621**
-200, + 140	T_i (°C)	50.72		399.46	753.62	874.09
	T_p (°C)	134.06		406.70	778.08	889.49
	T_f (°C)	203.80		414.86	797.10	898.55
	ΔH (J/g)	369.54		1.26	-14.88	-2.86
	Δm (%)	37.438		6.209	**Total**	**43.647**
-500, +200	T_i (°C)	85.14	121.38	401.27	761.78	
	T_p (°C)	111.41	132.25	413.95	786.23	
	T_f (°C)	121.38	256.34	429.35	807.97	
	ΔH (J/g)	-48.30	-23.00	3.57	-13.45	
	Δm (%)	35.526		14.839	**Total**	**50.365**
-560, +500	T_i (°C)	74.28	124.09	403.08	762.68	802.54
	T_p (°C)	114.13	135.87	412.14	789.86	825.18
	T_f (°C)	124.09	205.62	430.25	802.54	846.01
	ΔH (J/g)	-4.98	52.86	2.85	-6.61	-6.10
	Δm (%)	34.825		22.936	**Total**	**57.761**
+560	T_i (°C)	34.52	127.38	401.19	770.24	
	T_p (°C)	117.86	136.90	414.29	791.67	
	T_f (°C)	127.38	195.24	433.33	811.90	
	ΔH (J/g)	-2.12	-91.01	1.44	-4.21	
	Δm (%)	35.197		21.74	**Total**	**56.937**

In DTA/TG graphs primary y-axis represents the weight lost of minerals (%), secondary y-axis represents the microvolt values of which the DTA line is graphed and x-axis represents the Temperature values (°C).

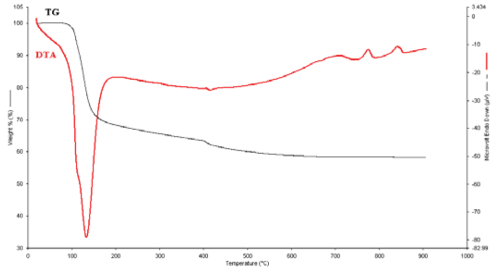

Figure 10. DTA/TG of -140 micron Inyoite.

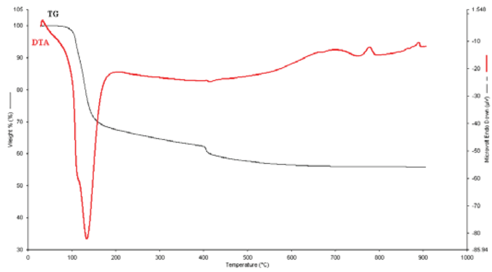

Figure 11. DTA/TG of +140, -200 micron Inyoite.

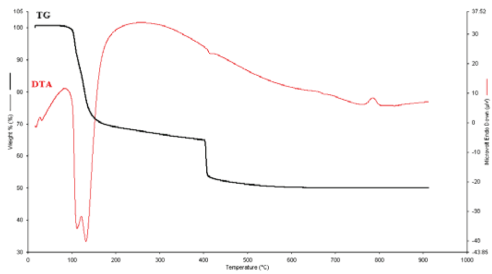

Figure 12. DTA/TG of +200, -500 micron Inyoite.

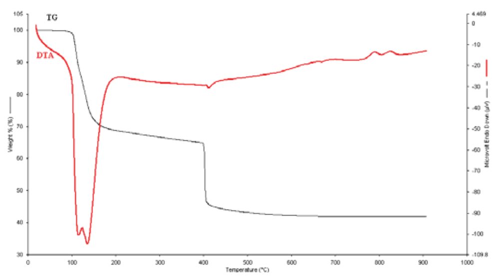

Figure 13. DTA/TG of +500, -560 micron Inyoite.

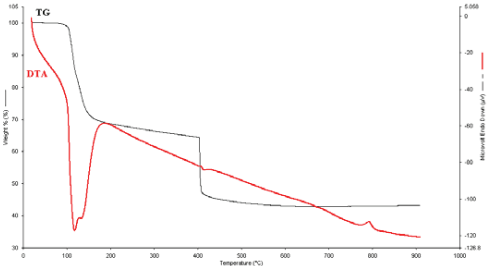

Figure 14. DTA/TG of +560 micron Inyoite

When the different particle sizes of the Inyoite minerals are investigated for DTA/TG analysis, all minerals had both endothermic and exothermic peaks and these peak values are given at the table 2.

At the end of the thermal gravimetry analysis -560, +500 microns particle sized Inyoite had the highest weight loss of 57.761% and - 140 microns particle sized Inyoite had the lowest weight loss of 41.621%.

3.5 Howitzer Neutron Permeability Results of Inyoite

At each thickness 10 parallel experiments are conducted and the final permeability values are calculated from the average of this 10 experiment.

The average neutron permeability values are given at the Table 3 and neutron absorption values are given at the Figure 15.

Table 3. Average neutron permeability values of the -500, + 200 particle size Inyoite.

Thickness (cm)	Neutron permeability (%)	Neutron absorption (%)
0.880	93.11	6.89
1.307	84.45	15.55

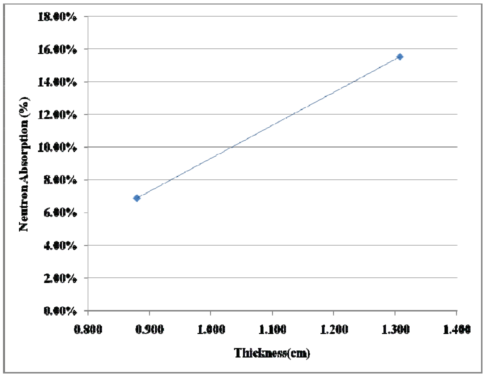

Figure 15. Average neutron absorption values of the -500, + 200 particle size Inyoite.

4 CONCLUSION

From the XRD results of the study different particle sized Inyoite minerals have the same pdf number of 01-073-0413.

Due to the interaction principle of the boron minerals with the thermal neutrons, the B_2O_3 amounts of the boron minerals are analyzed and Inyoite mineral that has a particle size +200, - 500 microns had the lowest content of 35.82 % and +560 micron had the highest content of 37.52%.

Inyoite mineral with particle size of -140 microns had no content of any Fe and As but it had Zn content of 154.1 ppm.

From the investigation of the energy and weight differences with the temperature change present in the Inyoite minerals, -200, + 140 microns particle sized Inyoite had highest peak point of 889.49^{o}C.

Experiments that are carried out only at the particle size of -500, +200 microns with pellet thicknesses of 0.880 cm resulted that only 6.89 % of neutrons absorbed and with pellet thicknesses of 1.307 cm resulted that 15.55 % of neutrons absorbed. As it is seen neutron absorption value is increased due to the increasing thickness.

Also it can be stated that at the other particle sizes of the Inyoite mineral would be better neutron absorption results due to their higher contents of B_2O_3 according to the particle size of -500, +200 microns.

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The Investigation of Comminution Properties of Different Coals on Impact Crusher and Numerical Analysis

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ABSTRACT In this study, breakage behavior of three different coals in a laboratory impact crusher was investigated. It is known fact that there are many difficulties and problems in pendulum and drop weight test method. A new model is developed by *t-family* value evaluation and Bond work index approach, and this model is tested. Obtaining *t-family* values from drop weight apparatus is time consuming and difficult. The mathematical model impact crusher results, is found to be more appropriate in determining *t-family* values than pendulum device and drop weight apparatus.

1 INTRODUCTION

Coal is a heterogeneous substance, which is consisted of combustible (organic matter) and non-combustible (moisture and mineral matter) materials. Comminution behavior or grindability of coal, which is a measure of its resistance to crushing and grinding, is related to its physical properties as well as its rank, chemical and petrographical compositions. Examination of the grinding behavior of coal is important, because the comminution of coal is essential for any kind of its utilization such as combustion, carbonization, gasification and beneficiation. Enormous quantities of coal are presently ground to produce power plant feed. The energy cost of grinding is significant at 5 kWh /ton - 15 kWh/ton (Vutharlu et al., 2003; Chelgani et al., 2008; Özbayoğlu, et al., 2008).

Nowadays, impact crushers are widely used for comminution operations because of their high size reduction ratio, easy modification of the product and a relatively simple design. On the other hand, the prediction of the behavior of mineral processing plants through modeling and simulations is the more and more employed as a reliable, time and cost-saving approach for development, analysis and optimization of crushing circuits. In this context, the availability of relevant mathematical models for impact crushers is important for a successful simulation of such plants (Nikolov, 2002).

Bond testing has been in use since the late 1920's; laboratories and operations around world use the procedure as a component of comminution circuit design and to evaluate plant performance. In spite of such long-standing use, the topic of accuracy and precision of Bond work index determinations recurs with great frequency (Mosher&Tague, 2001).

Single particle tests to determine the comminution behavior of ore can be separated into pendulum device (Figure 1) and drop weight apparatus (Figure 2) based tests.

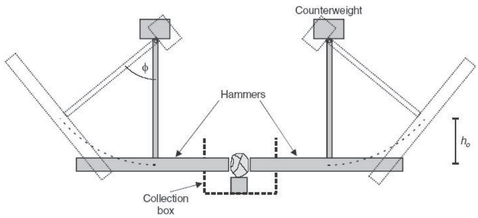

Figure 1. Pendulum device.

Figure 2. Drop-weight test apparatus.

Twin pendulum test relies on the particle being broken between an input pendulum released from a known height and a rebound pendulum. The drop weight test differs in that the particles are placed on a hard surface and struck by a falling weight. Both these approaches have been used extensively in field of comminution. In recent years, the drop weight apparatus are being replaced by twin pendulum. The standard drop weight device is fitted with a 20 kg mass, which can be extended to 50 kg. The effective range of drop heights is 0.05 to 1.0 m, which represents a wide energy range from 0.01 to 50 kWh/t (based 10-50 mm particles). Following sample preparation the mean mass of each set of particles to be broken is calculated. The results from the drop weight tests provide an energy/input size/product size relationship. This relationship is analyzed using a set of curves to describe the size distribution produced from breakage events of increasing size reduction or energy input (Bearman et al., 1997).

Several test methods are improved in the field Narayanan and Whiten (1998) have widely used a novel procedure for estimation of breakage distribution functions of ores from the so-called *t-family* of curves. In this method, the product size distribution can be represented by a family of curves using marker points on the size distribution defined as the percentage passing(t) at a fraction of the parent particle size. Thus, t_2 is the percentage passing an aperture of half the size of the parent particle size, t_4 is one quarter and t_{10} is one-tenth of parent particle size. They have proposed empirical equations for relating the reference curve data t_{10} with the impact energy.

The t_n versus t_{10} relationships can then be used to predict the product size distributions at different grind times (Sand & Subasinghe, 2004).

The t_{10} value is related to the specific comminution energy by the equation:

$$t_{10} = A\left(1 - e^{-bEcs}\right) \qquad (1)$$

where,

t_{10}= percentage passing 1/10th of the initial mean size,

Ecs= specific comminution energy (kWh/t)

A,b= ore impact breakage parameters

In the drop weight test, a known mass falls through a given height onto a single particle providing an event that characterization of the ore under impact breakage. Although, the drop weight test has advantages in terms of statistical reliability and the potential use of the data from the analysis, it has a number of disadvantages, particularly the length of time taken to carry out a test. For each drop weight test, 15 samples are tested in 5 size fractions at 3 levels of energy input (Kingman et al., 2004).

In this study, breakage behavior of three different coals in a laboratory impact crusher was investigated. It is known fact that there are many difficulties and problems in drop weight test method. A new model is developed by *t-family* value evaluation and Bond work index approach, and this model is tested.

2 MATERIALS AND METHOD

2.1 Material

Three different coal samples taken from different region of Turkey were used as the experimental materials. The chemical properties of the coal samples are presented in Table 1.

Table 1. Proximate analysis of coal samples used in experiments.

	Soma	Aydın	Kale
Moisture(%)	15.09	17.92	21.13
Ash (%)	6.75	8.69	15.56
Fixed Carbon (%)	40.68	38.62	32.43
Volatile Matter (%)	37.48	34.78	30.88
Net Calorific Value (kCal/kg)	5000	4400	3600

2.2 The Test of Standard Ball Mill Bond Grindability

The Bond grindability tests were conducted with -3.35 mm dry feed materials in a standard ball mill (30.5x30.5 cm) following a standard procedural outline described in the literature (Yap et al., 1982; Deister, 1987; Deniz et al., 1996; Deniz, 2004). The work indices were determined at a test sieve size of 106 μm. It has no lifters and all the inside comers are rounded. The Bond ball mill (Figure 3) is operated at 70 rpm and is equipped with a revolution counter. The grinding charge consists of 285 iron balls weighing 20.125 grams. The ball size distribution is given in Table 2.

Figure 3. Bond mill.

Table 2. Ball size distribution.

Ball diameter (mm)	Number of balls
38.10	43
31.75	67
25.40	10
19.05	71
15.87	94
Total	285

The standard Bond grindability test is a closed-cycle dry grinding and screening process, which is carried out until steady state conditions are obtained. This test was proposed by Bond and Maxson (1943) and used by different researcher (Yap et al., 1982, Austin & Brame, 1983; Magdalinovic, 1989). The material is packed to 700 cm^3 volume using a vibrating table. This is the volumetric weight of the material to be used for grinding tests. For the first grinding cycle, the mill is started with an arbitrarily chosen number of mill revolutions. At the end of each grinding cycle, the entire product is discharged from the mill and is screened on a test sieve (P_i). Standard choice for P_i is 106 micron. The oversize fraction is returned to the mill for the second run together with fresh feed to make up the original weight corresponding to 700 cm^3. The weight of product per unit of mill revolution, called the ore grindability of the cycle, is then calculated and is used to estimate the number of revolutions required for the second run to be equivalent to a circulating load of 250%. The process is continued until a constant value of the grindability is achieved, which is the equilibrium condition. This equilibrium condition may be reached in 6 to 12 grinding cycles. After reaching equilibrium, the grindabilities for the last three cycles are averaged as a Bond grindability index (G_{bg}).

$$G_{bg} = \frac{P}{n}$$
(2)

G_{bg}= Grindability index of sample (g/rev)
P = The net amount of grinding products, (g)
n= Revolution of Bond ball mill

3 EXPERIMENTS

Firstly, Standard Bond's grindability tests were made for three coal samples, namely are Soma, Aydın and Kale. Results of tests, Bond grindability values of coal samples were appeared 0.53 g/rev, 0.82 g/rev and 0.92 g/rev respectively. Then, the laboratory horizontal shaft impact mill, rotating at 2840 rpm, driven by 1.1 kW motor, carry three rows of hammers, were used in the experiments. One kilogram sample of six mono-size fractions (-6.7+4.75, -4.75+2.8, -2.8+1.7, -1.7+1.18, -1.18+0.600, -0.600+0.355 mm) were prepared and crushed in a laboratory-scale impact mill for determination of the *t-family* curves. Each sample was taken out of the impact mill and sieved product size analysis.

Results of *t-family* curves versus mean size fraction for their different coals are shown in Figure 4-6.

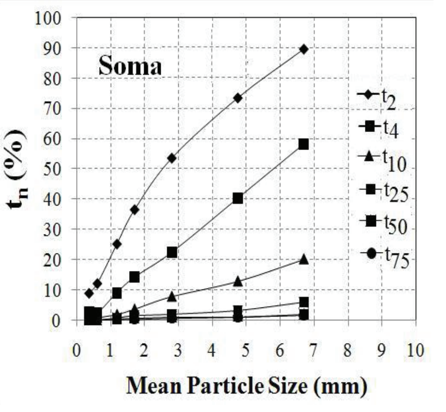

Figure 4. t_n versus mean size fraction for Soma.

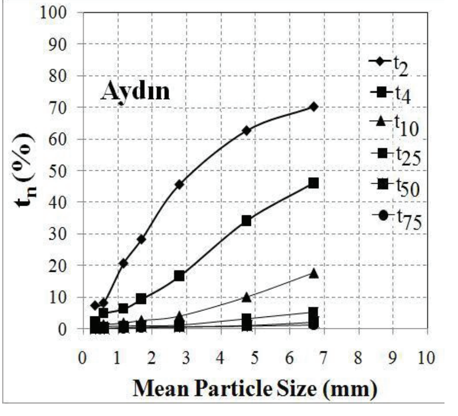

Figure 5. t_n versus mean size fraction for Aydın.

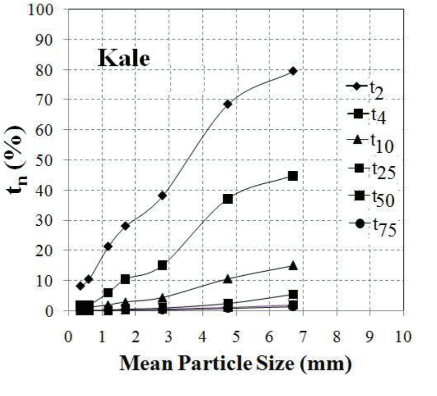

Figure 6. t_n versus mean size fraction for Kale.

4 PROPOSED MODEL

Narayanan&Whiten (1998) that the cumulative fraction of products passing $1/n^{th}$ of the mean size was denoted by t_n. It was also reported that this relationship was applicable to different ore types tested under different impact loading conditions. A similar relationship has been observed t_n values for crushing products in the direct laboratory impact mill and forms the basis of the proposed model.

$$t_n = \left(18.80 * n^{0.730} * X\right) + \left(0.87 * G_{bg} + 1.84\right)$$
$$r^2 = 0.945 \qquad (3)$$

where,

t_n: the cumulative percentage passing($1/n^{th}$) of the mean particle size (%)

G_{bg}: Grindability of coal (g/rev)

X: Mean particle size (mm)

The experimental values and the calculated results obtained by Eq.(3) were compared in Figure 7.

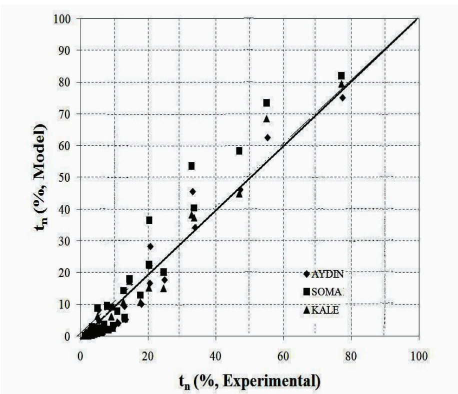

Figure 7. Comparison of experimental and calculated t_n value for coal.

Eq.(3) mostly satisfies the experimental values in a wide range of feed size, and Eq.(3) is useful especially when evaluating the particle size proportion in the actual operation by a high regression value (r^2=0.945).

In this partial, the relationship between the cumulative percentage passing (t_n) with Bond Grindability (G) and mean feed size (X) was empirically described by Eq. (3).

5 CONCLUSIONS

In this study, tests of three different coals with an impact crusher mill were carried out in laboratory crushing. The effects of mean particle size and grindability of coals were investigated on product size distribution. A set of *t-curves* were calculated from the laboratory impact mill, and a new model was developed.

The results showed that Soma lignite sample was a more friable than other two samples.

Obtaining *t-family* values from pendulum device and drop weight apparatus is time consuming and difficult. The mathematical model impact crusher results, is found to be more appropriate in determining *t-family* values than drop weight apparatus.

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Investigation by Simple and Multivariable Linear Analysis Methods of the Interrelationships between the Bond Grindability with Physicomechanical and Chemical Properties of Coals

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ABSTRACT Although there are several formulas available for predicting Bond grindability of coal, most of them are linear and do not simultaneously take into consideration most of the relevant factors. Several test methods are improved in the field of coal workability determination by mechanics of excavation. They consider Impact strength index (*ISI*), Point load index (*Is*) and Friability index (F_D) can be mentioned of easy examples. In Bond grindability test, several difficulties are found in Standard processes. Therefore, some researchers have investigated the prediction of grindability values of coals based on proximate analysis, petrography, and vitrinite maximum reflectance with using linear regression. In this research, different techniques for the estimation of Bond grindability (G_{bg}) values of coals are studied. Data from ten sub-bituminous coals from Turkey are used by featuring physicomechanical (*ISI*, *Is* and F_D) and chemical eight coal parameters, which include proximate analysis (Moisture, Ash, Volatile Matter, Fixed Carbon and Calorie). Linear and multivariable linear regression techniques are used for predicting the G_{bg} values for the specified coal parameters. Results indicate that a multivariable linear regression gave the most accurate G_{bg} prediction than simple regression in the estimation process.

1 INTRODUCTION

In terms of global quantities of material reduced in size, it has been estimated that the annual tonnage is of the order of several thousand millions, and in terms of the energy expended, the yearly megawatt hours amount to several hundred millions. Many expressions of grindability have been proposed over the years, but of these two of them have come into prominence because they have become the recognized basis for design of certain types of mill. One of them is the Hardgrove grindability index, associated mainly with vertical spindle mills and the other is Bond's grindability index, associated with tumbling mills (McIntryre&Plitt, 1980; Prasher, 1987).

The grindability of coal is an important practical and economic property to coal handling and utilisation aspects, particularly for pulverised coal fired utilities. In general, coal grindability characteristics reflect the coal hardness, tenacity, and fracture which are influenced by coal rank, petrography, and the distribution and the types of minerals (Vutharlu et al., 2003; Chelgani et al, 2008).

Coal is a heterogeneous substance which is consisted of combustible (organic matter) and non-combustible (moisture and mineral matter) materials. Therefore, the prediction of grindability index on the basis of the proximate analysis may not give accurate result if the age, rank and petrographic composition of coals are not similar (Özbayoğlu, et al., 2008).

Although the HGI testing device is not costly, the measuring procedure to get a HGI value is time consuming. Therefore, some researchers have investigated the prediction of HGI based on proximate analysis, petrography, and vitrinite maximum reflectance with using regression (Trimble&Hower, 2003; Ural&Akyıldız, 2004; Jorjani, et al, 2008; Chelgani et al., 2008).

Other several researchers used nonlinear multivariable regression and generalized regression neural network to find the correlation between HGI and the proximate analysis, petrography, and vitrinite maximum reflectance of coals (Li et al., 2005; Özbayoğlu, 2008).

In the design of grinding circuits, the Bond grindability method is widely used for a particular material in dimensioning mills, power needs and the evaluation of performance. Its use as an industrial standard is very common provides satisfactory result in the all industrial applications. Despite having many advantages, this method has some drawbacks such as being tiring and requiring long test time also it needs a special mill (Deniz&Özdağ, 2003). Therefore, some researchers have investigated the prediction of G_{bg} based on mechanical, physical and chemical properties of minerals and coals, with using regression (Berry and Bruce, 1966; Smith and Lee, 1968; Horst and Bassarear, 1977; Yap et al., 1982; Magdalinovic, 1989; Nematollahi, 1994; Deniz et al., 1996, Deniz &Özdağ, 2003).

Several test methods are improved in the field of coal workability determination by mechanics of excavation. They consider Impact strength index (*I.S.I.*), Point load index (*Is*) and Friability index (F_D) can be mentioned of easy examples.

Deniz et al. (1996) found an inverse relation between Bond grindability index and point load index for coals. Su et al. (2004) found some correlations between mechanical properties and Hardgrove grindability index for Zonguldak Coals. Similarly, Tiryaki (2005) derived correlations between mechanical properties and Hardgrove grindability index for Çayırhan Coals.

There are several formulas available for predicting the Bond grindability of coal, most of them are linear and do not simultaneously take into consideration most of the relevant factors.

In this research, simple and multivariable linear regression techniques for the estimation of Bond grindability (G_{bg}) values of coals are studied. Samples, which are taken ten different lignite coals from Turkey, are used by ten coal of different mechanical (*ISI*, I_S and F_D) and chemical properties, which include proximate analysis (Moisture, Ash, Volatile Matter, Fixed Carbon and Calorific Value).

Linear and Multivariable regression techniques, SPSS software package are used for predicting the G_{bg} values for the specified coal parameters. Results indicate that a multivariable linear regression gave the most accurate G_{bg} prediction than linear regression in the estimation process.

2 METHOD

2.1 The Bond Ball Mill Grinding Test

The standard Bond grindability test is a closed-cycle dry grinding in a standard ball mill (30.5x30.5 cm) and screening process, which is carried out until steady state condition is obtained. This test was described as follow (Karra, 1981; Yap et al., 1982; Deister, 1987; Deniz, 2004):

The material is packed to 700 cc volume using a vibrating table. This is the volumetric weight of the material to be used for grinding tests. For the first grinding cycle, the mill is started with an arbitrarily chosen number of mill revolutions. At the end of each grinding cycle, the entire product is discharged from the mill and is screened on a test sieve (P_i). Standard choice for P_i is 106 micron. The oversize fraction is returned to the mill for the second run together with fresh feed to make up the original weight corresponding to 700 cc. The weight of product per unit of mill revolution, called the ore grindability of the cycle, is then calculated and used to estimate the number of revolutions required for the second run to be equivalent to a circulating load of 250%. The process is continued until a constant value of the grindability is achieved, which is the equilibrium condition. This equilibrium condition may be reached in 6 to 12 grinding cycles. After reaching equilibrium, the grindabilities for the last three cycles are averaged. The average value is taken as the standard Bond grindability index (G_{bg}).

The products of the total final three cycles are combined to form the equilibrium rest product. Sieve analysis is carried out on the material and the results are plotted, to find the 80% passing size of the product (P_i).

2.2 The Impact Strength Index Test (*ISI*)

The test consists of basically the crushing of coal fragments, contained in a hollow cylinder closed at the bottom, by five successive drop hummer blows. The apparatus consists of a vertical steel cylinder of 4.45 cm internal diameter, closed at lower end by a screwed cap. A steel plunger, which weights 1.8 kg and 4.29 cm in diameter at the bottom, is fitted loosely inside the hollow cylinder. This eight 30.48 cm, measured from bottom of plunger to the inside surface of the base of cylinder. Basically, the test consists of placing 100 g of coal in the 9.51-0.318 cm size range in the cylinder and subjecting the assembly of fragments to 20 blows from plunger, raised to and dropped from the maximum height each time. The amount of coal is remaining in the initial size range after the test as defined as the *ISI* (Evans&Pomeroy, 1973; Akçın&Baş, 1990).

2.3 The Friability Index Test (F_D)

In this test, coal samples are fallen on to iron plate of which thickness is 12 mm from 186 cm height. Coal samples are sieved and 77-49 mm sieve fraction is separated. From this sieve fraction approximately 4500 g part is tested. The coal samples prepared is fallen twice after words, sieved with 49, 37, 25, 20 and 13 mm sieves. For each sieve size, size reduction rate is determined and later these rates are added and given as total percentage in size reduction. The percentage value is called as Friability index and it is showed as F_D (Eskikaya, 1976).

2.4 The Point Load Index Test (I_s)

Irregular lumps of coal samples can be used in point load test. In this test, coal samples are loaded between the two hardened steel points (plates) of the apparatus. The force (*P*), at which the coal breaks is determined from the peak pressure recorded a pressure gauge (Hoek&Bray, 1981; Akçın&Baş, 1990).

3 EXPERIMENTS

3.1 Materials

The test programme is conducted on of ten different lignite coal samples. Table 1 identifies these coals according to their chemical properties. In the experiments, coals are prepared and tested in the order of coal Friability test, point load test and impact strength test, respectively. After that, -3.35 mm materials are prepared for Bond grindability test. The results of experiments are given in Table 2.

Table 1. Chemical analysis values of samples used experiments.

	Moisture	Ash	Fixed Carbon	Volatile Matter	NCV*
1	13.58	9.71	39.72	36.99	5330
2	17.91	8.69	38.62	34.78	4400
3	15.09	6.75	40.68	37.48	5000
4	20.54	12.68	34.78	32.00	4150
5	21.13	15.56	32.43	30.88	3600
6	23.28	14.04	32.42	30.26	3390
7	30.53	9.63	29.99	29.85	3400
8	20.89	18.62	29.79	30.78	3450
9	25.56	16.05	27.45	30.94	3210
10	26.39	15.64	20.60	37.37	3200

*NCV: Net Calorific Value

Table 2. Averages of some test results for coals used experiments.

	G_{bg}	Is	F_D	ISI
1	0.92	7.72	26.82	61.02
2	0.84	7.55	23.41	60.58
3	0.82	11.65	20.20	66.88
4	0.81	11.32	17.07	70.15
5	0.73	11.94	16.83	70.56
6	0.69	10.81	16.03	72.72
7	0.64	12.93	17.27	74.15
8	0.54	21.15	16.22	74.99
9	0.44	38.02	11.53	74.08
10	0.45	41.03	10.10	79.89

4 RELATION OF TESTS

Firstly, using the simple regression analysis, linear, exponential and logarithmic functions were analyzed and it was found that the values of the correlation coefficient were very low. Secondly, with polynomial functions, it was shown that the values of correlation coefficient were high. Therefore, it was decided to use the multivariable regression in this study. Samples of lignite coals in four reflectance ranges, 60-70%, 70–80%, 80–90% and 90– 100%, , were selected for combined chemical and physicomechanical properties. Statistical analyses were conducted using the SPSS software package.

4.1 Linear Relations

The test results were analyzed using the method of least squares regression. The Bond grindability values were correlated with corresponding physicomechanical properties (*ISI*, I_s and F_D) and proximate analysis values (moisture, ash, volatile matter, fixed carbon and net calorific value). As shown in Figs. 1-8, no significant correlation was found between G_{bg} values with physico-mechanical properties and proximate analysis values. The reason for the lack of correlations is probably due to the fact that grinding is a complex phenomenon and depends on more than a coal property. For this reason, multivariable analysis is necessary for the evaluation of G_{bg} .

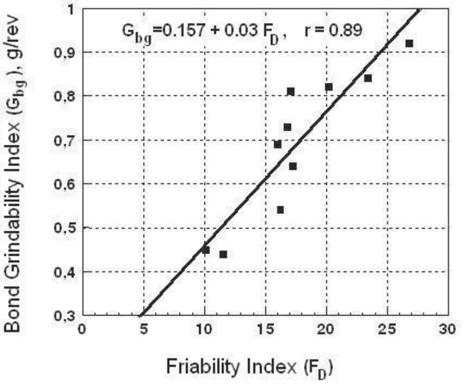

Figure 1. The relation between G_{bg} and F_D

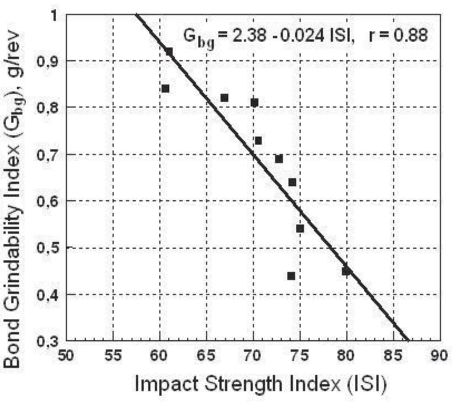

Figure 2. The relation between G_{bg} and *ISI*.

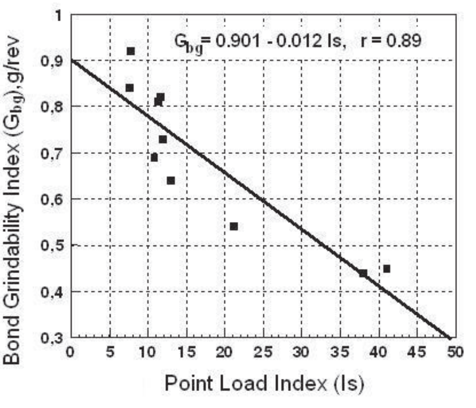

Figure 3. The relation between G_{bg} and *Is* .

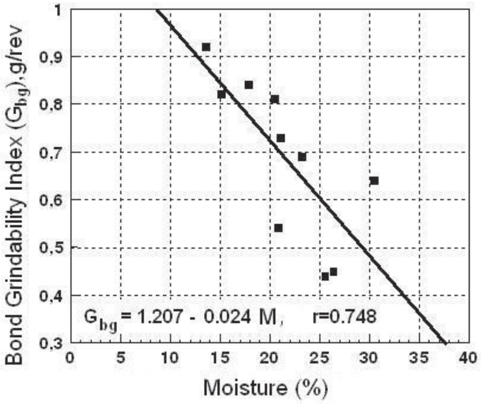

Figure 4. The relation between G_{bg} and Moisture.

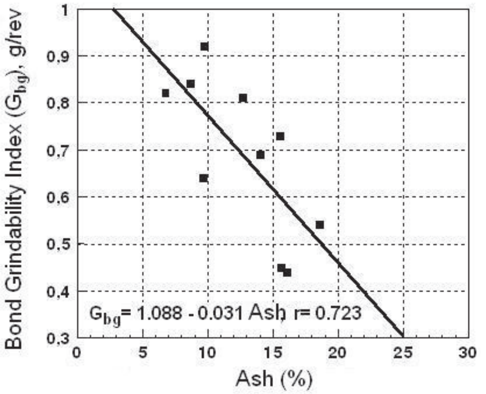

Figure 5. The relation between G_{bg} and Ash.

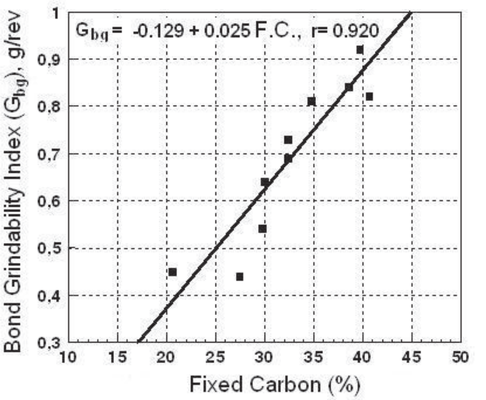

Figure 6. The relation between G_{bg} and Fixed Carbon.

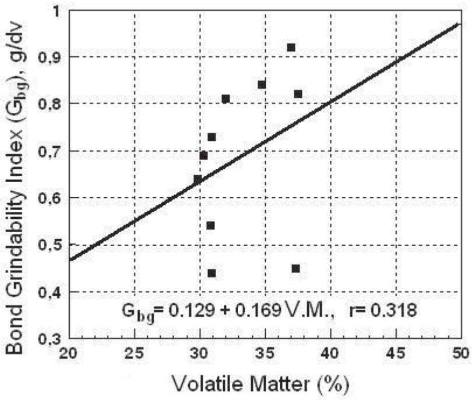

Figure 7. The relation between G_{bg} and Volatile Matter.

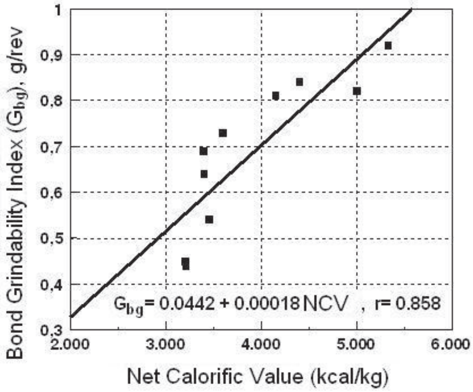

Figure 8. The relation between G_{bg} and Net Calorific Value.

4.2 Multivariable Relations

Multivariable linear regression analysis was applied to describe the relationships between dependent and independent variables. The purpose of multivariable linear regression analysis is to simultaneously identify two or more independent variables which explain variations of a dependent variable.

To develop the prediction equations having strong correlation coefficients for the Bond grindability index, the multiple regression analysis including three and four variables was performed. The correlation matrix for Bond grindability index (G_{bg}) and volatilization parameters are presented in Table 3.

The alternative model for physicomechanical properties produced is given as:

$$G_{bg} = 1.671 + 0.01(F_D) - 0.007(I_s) - 0.012(ISI) \quad r^2 = 0.96 \tag{1}$$

The simple regression analysis showed that there were no correlations between the G_{bg} and volatile matter. Therefore, in this study, volatile matter was decided to not use with the multivariable regression analysis. For proximate analysis, it was found that use of moisture (M), ash (A), fixed carbon (FC), and net calorific value (NCV) can achieve the best results to predict the G_{bg}.

The following equation resulted between G_{bg} and proximate analysis:

$$G_{bg} = -0.659 + 0.009(M) + 0.004(A) + 0.022(FC) + 0.001(NCV) \quad r^2 = 0.93 \tag{2}$$

Table 3. Full correlation matrix for the regression data set.

Properties	G_{bg}	F_D	*Is*	*ISI*	Moisture	Ash	F.C.	V.M.	N.C.V
G_{bg}	1	0.90	0.895	0.881	0.748	0.724	0.920	0.318	0.859
F_D		1	0.817	0.931	0.754	0.692	0.892	0.356	0.870
Is			1	0.727	0.526	0.592	0.845	0.580	0.610
ISI				1	0.795	0.690	0.910	0.373	0.859
Moisture					1	0.410	0.810	0.567	0.879
Ash						1	0.724	0.481	0.741
F.C.							1	0.265	0.873
V.M.								1	0.649
N.C.V.									1

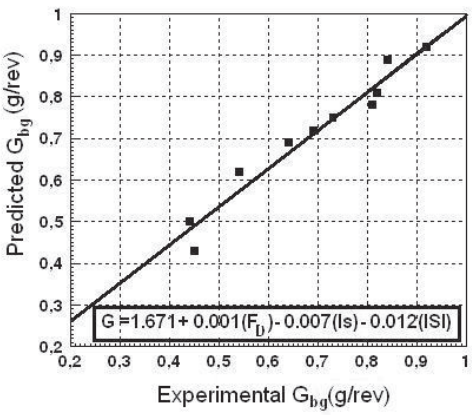

Figure 9. Scatterplot of G_{bg} values predicted by multivariable regression analysis (SPSS) versus those experimental for mechanical properties.

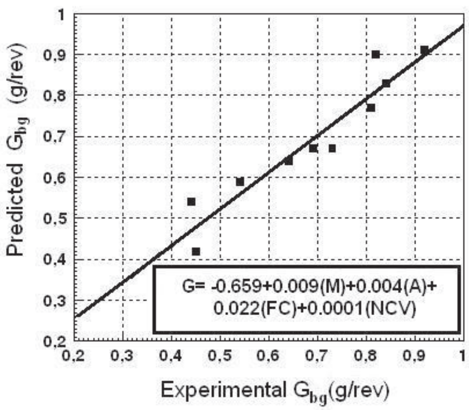

Figure 10. Scatterplot of G_{bg} values predicted by multivariable linear regression analysis (SPSS) versus those experimental for chemical properties.

5 CONCLUSIONS

Ten different lignite coals were tested in the laboratory and the relations between the Bond grindability index with proximate analyses values and physicomechanical properties were investigated. The simple regression analysis showed that there were no significant correlations between the G_{bg} with proximate analysis value and physicomechanical properties However, as a result of multiple regression analysis, the prediction equations having strong correlation coefficients (r=0.96 and r=0.93) were developed for the Bond grindability index.

Concluding remark is that the G_{bg} of lignites can be predicted from the equations developed from the multiple regression analysis especially for preliminary investigations. In addition, the derived equations including index test values are important for the practical consideration.

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Improvement of the Scheme for Flotation of Molybdenum Factory "ELLATZITE"

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ABSTRACT Copper –porphyry deposit "ELLATZITE" is one of the biggest deposit for copper ,gold and molybdenum extraction in Bulgaria .The main industrial element is copper ,the second industrial elements are gold and molybdenum . In 2008 the production of molybdenum was recovered which was stopped in 1990 due to economical reasons .A new technological scheme was developed (Kairaykov) where the flotation is done in inert –gas medium .During commisioning of the molybdenum section it was found that some changes in the scheme were necessary and these changes which were related to reagent regime were done.

1 CHANGES IN THE SCHEME

Molybdenum concentrate that was produced in the enrichment factory is supplied through two conditioners and each of them is with volume 33m^3. The use of two condtioners assures constant density which is necessary for the process and these conditioners play the role of buffer when there are short –time disruptions and deviations from extraction of collected concentrate. The copper – molybdenum concentrate that goes to molybdenum flotation is with copper content 23-24% and molybdenum content 0.18 - 0.20%. NaHS was used as the depressant for copper minerals. During the first exploitation of the scheme it was found that pH of the system was very high and it was due to NaHS, it imposed the use of sulphur acid in the first conditioner and of NaHS in the second conditioner. After the conditioning the copper –molybdenum concentrate is with pulp density 35% and it goes firmly in main molybdenum flotation and this flotation is done in 9 flotation machines "Vemco", situated in a cascade (5+4).

The tail product that is produced from molybdenum flotation is copper concentrate. The result of the change in reagents regime increased the extraction in the main flotation with 3-4 %.

The low content of molybdenum in the collected concentrate and the high degree of enrichment which have to reach 200 times are in the base of two important specific characteristics of the technology of copper - molybdenum selection (Kovachev) –a lot of flotation operations with big number of points of supply of reagents and very unusual little flow of foam and middling products in cleaning operations .

After the second cleaning the result foam product is with density 5-7% and it becomes thick in thickener f6 with central actuation .The resulting thickener product is already with density 30-35% and it goes to a conditioner apparatus on the next I I I flotation.

After the sixth cleaning it is necessary to do an intermediate thickening of the product and it is done in cone thickener with volume 0.3m^3 and next 2 cleaning operations.

Two machines IG with volume of the cell $0.130m^3$ were added to the designed machines to achieve a conditioning quality of molybdenum concentrate 45-47%.

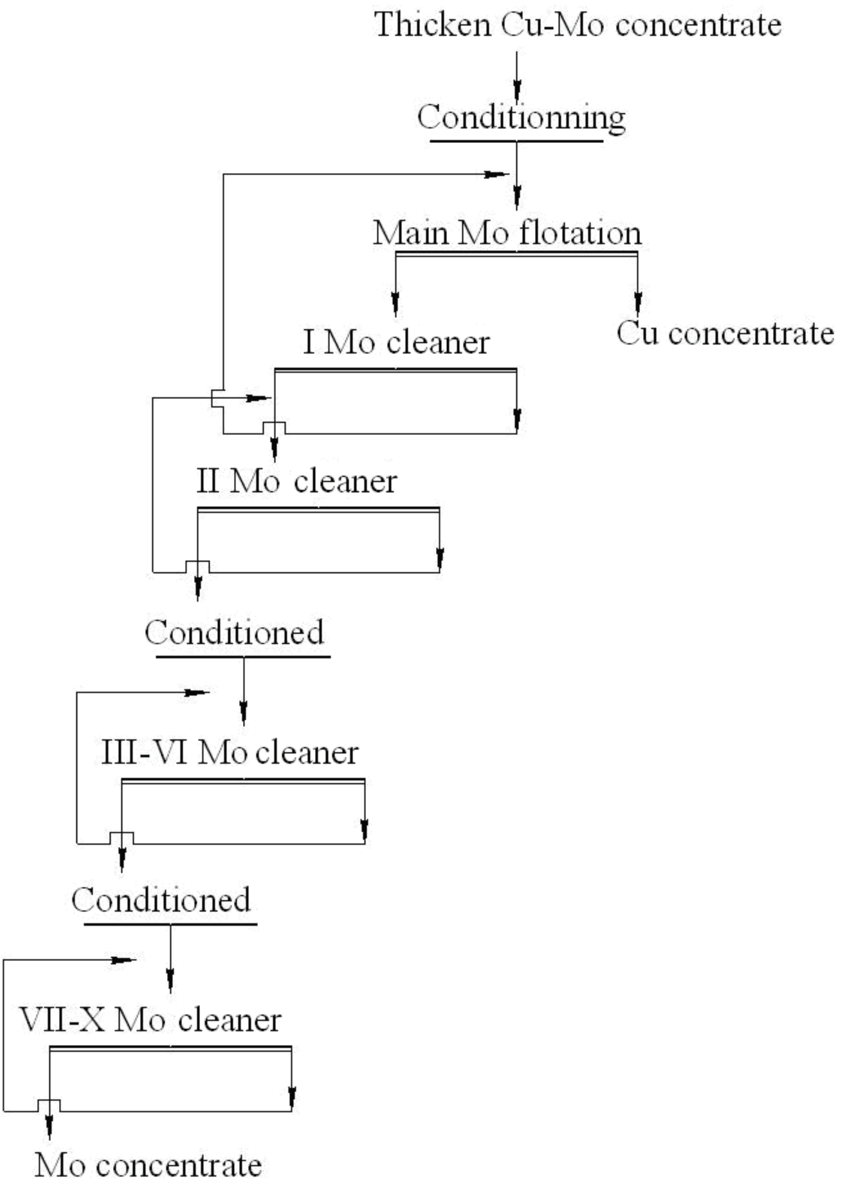

Figure 1. Flowsheet of Mo flotation of enrichement complex "Ellazite".

2 OPTIMIZATION OF REAGENTS REGIME

The regulation of pH in the main flotation is done through supply of sulphur acid in the first agitation apparatus.

NaHS is supplied in fractions with dozed pumps and 78% from the total quantity is introduced in the second conditioner -33m^3 and the quantity that remains is used in the next cleaning and controlled operations.

To collect the molybdenum we use kerosene and it is supplied in the process as 0.5% emulsion (Mehandjiski) and this emulsion is with long shelf –life .

3 AUTOMATIZATION OF THE PROCESS

All data for the technological process are in operator station (Fig.2). From this station we can observe the work of all machines, equipment and the technical parameters of the process. The concentrate is packed in big bags -1 tone and it is ready to be delivered.

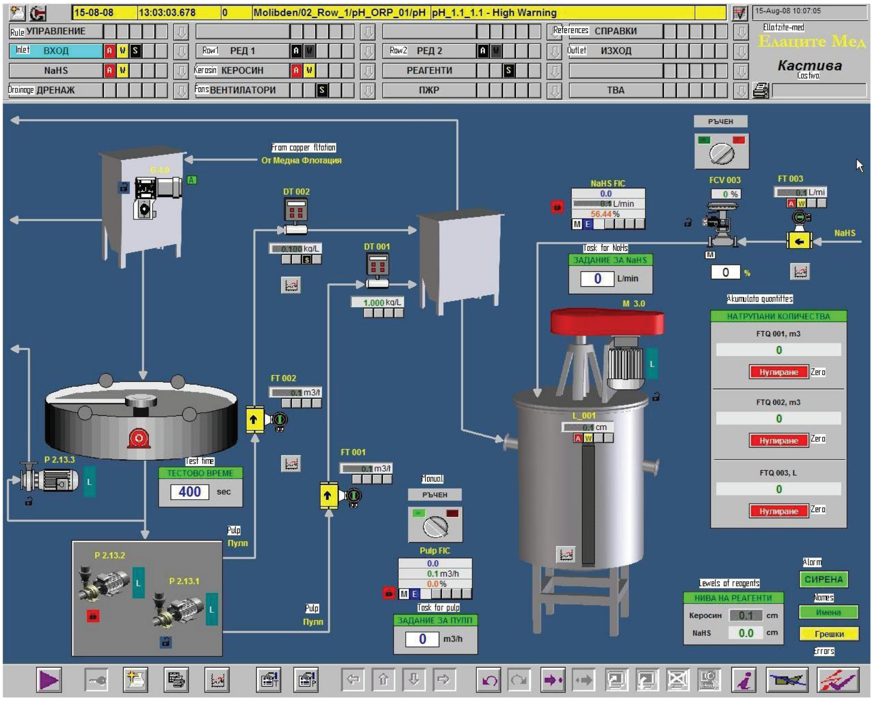

Figure 2. Operator station.

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Application of Belt Filter Press LAROX PF 50 in the Conditions of Enrichment Factory 'ELLATZITE'

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ABSTRACT Dewatering of the fine milled products from the flotation plant that contain 70-75% moisture is done in two stages: thickening and filtration. The importance of the dewatering process can be understood from its expenses and they are 4-10 % from the total sum of the expenses for ore enrichment.

The application of filter –press "LAROX PF -50" allowed to replace the drums that are morally and physically aged and the vacuum filters BOW -40 and to guarantee the moisture of the extracted concentrate in the limit 6-8%.

1 GENERAL DATA FOR THE FACTORY

The enrichment plant "Mirkovo" started to work in February 1983. The factory processes copper–porphyry ores that are extracted from mine 'ELLATZITE" that is situated on the northern part of Stara planina. The enterprise is unique that the ore passing through a tunnel with length 6700 m to a rubber–conveyor belt and reach the factory.

The productivity of the factory is 36000 t/day. The main ore forming minerals are chalcopyrite, pyrite, bornite, molibdenite and quartzite they are formed in the incorporating rocks like pro–veins. The size of the ore grains vary from several microns to millimeters -more often is 0.1 to 0.3 mm. The gold and the silver are associated mainly with the copper minerals.

2 SCHEME OF FLOTATION

The scheme of flotation from deposit "Ellatzite" consists of two stages and they reveal best the technological particularities of the copper –porphyry ores. In the first stage a final waste is obtained and a coarse collected concentrate goes to milling and flotation in the second stage. On Fig. 1, a linear scheme for collective flotation is shown.

The grinding of the coarse concentrate is done in ball mill with volume $32m^3$ –diameter 3.2m and length 4.5m and the content of the class is 0.008 mm in the ready product is approximately 90%.

The flotation reagents that are used in the factory are:

- Collector –ethyl and amyl xanthate
- Frother – methyl-isobutyl carbinol
- Regulator and depressor –hydrate lime

The supply of the flotation reagents is on many points with purpose to create an intense regime over the whole front of main flotation without super saturation of the initial chambers and reduce the efficiency of the refined operations.

While doing the enrichment is obtained a collective concentrate with copper content 23.50 – 24.50%, molybdenum 0.18 – 0.20 % and copper extraction 92%.

Depend on the specific conditions, dehydration or selection of molybdenum and next dehydration is done.

The thickening is done in radial thickener with diameter 30m with peripherical activation to 60 -65% density of solid substance. Then the filtration is done using a vacuum filters BOW -40, made in Russia.

Using as base technological task from 'ELLATZITE", GEOPROJECT LTD, Sofia was elaborated an investment project for incorporating a filter –press "LAROX PF -50 "GEOPROJECT. The chain scheme of the machines and apparatus is shown on Fig.2. The construction of the press includes fllter elements –plates that are placed horizontally between two pressing plates –"LAROX". During the filtration the plates in the package are pressed one to another and the package is opened to pull out the sediment. The package can be open and close using hydraulic cylinders. The filter cloth is dimensionless and goes zig–zag between the filter plates and this leads to filtration of the formed sediment from the sides of the cloth. The cloth directs the sediment aside of the filter and in the same time cleans both sides using water sprays with high pressure.

The work of the press is controlled automatically using an operator mechanism that is with programmed logistics and indications of all the operations. Since the principle of filtration under pressure is new for our factory we will arrange the operations that are done:

-Filtration – after the package with plates is closed the slime goes in every chamber for filtration. The filtrate flows through the cloth toward the chamber for filtration and it was discharged through the collectors to the tube for filtrate

- First press – the diaphragm full of water squeezes the sediment and presses it to the surface of the cloth and this way pull out the filtrate from the sediment

- Auxiliary wash – the washing liquid is pumped toward the chambers for filtration like the slime

When this liquid fills the chamber for filtration the diaphragm rises up and the water goes out till the upper part of the diaphragm. The washing liquid leaks to the tubes for filtrate after passing the filtrated sediment and the cloth.

-Auxiliary press – the washing liquid that stay in the chamber after the washing is forced under pressure out of the sediment.

-Drainage using air – with compressed air we can achieve a complete drainage. The air fills the filter chamber rise the diaphragm and forces the water over the diaphragm out of the filter. The use of air through the sediment reduces its humidity till an optimal level and in the same time empty the chamber for filter.

- Pull out of the sediment –after the drainage with air is over the plates package is open and the mechanism for activation of the cloth is started. The sediment from the filter falls on two rubber –conveyor belts B =100 and is discharged in the stock for concentrate.

The control and the observation of the work of the filter, the selection of the necessary program for filtration, the change of the parameters and the testing of the auxiliary mechanisms are done from operator interface. It gives clear text information for the signalizations –Fig. 3.

The right adjustment of the press, the regulation of the process according the different situations provides opportunity to achieve good results.

The factors that influence the results of filtration are the output material, the condition of the cloth and the channels for filtration.

The quantity of the extracted sediment depends on the density of the supplied material for filtration, the pressure of filtration and the time for filtration.

The humidity of the obtained sediment depends on the grain size of the particles of the pulp , the pressure of pressing , the pressure of the drainage air and also the pressure of the drainage with air.

The use of filter –press "LAROX PF -50 " guarantees the humidity of the extracted copper concentrate in the limit 6-8% and this allowed to be put in the market without any problems. The starting of the filter press will lead to reduce of the consumption of energy in times and will alleviate the technological regime of flotation.

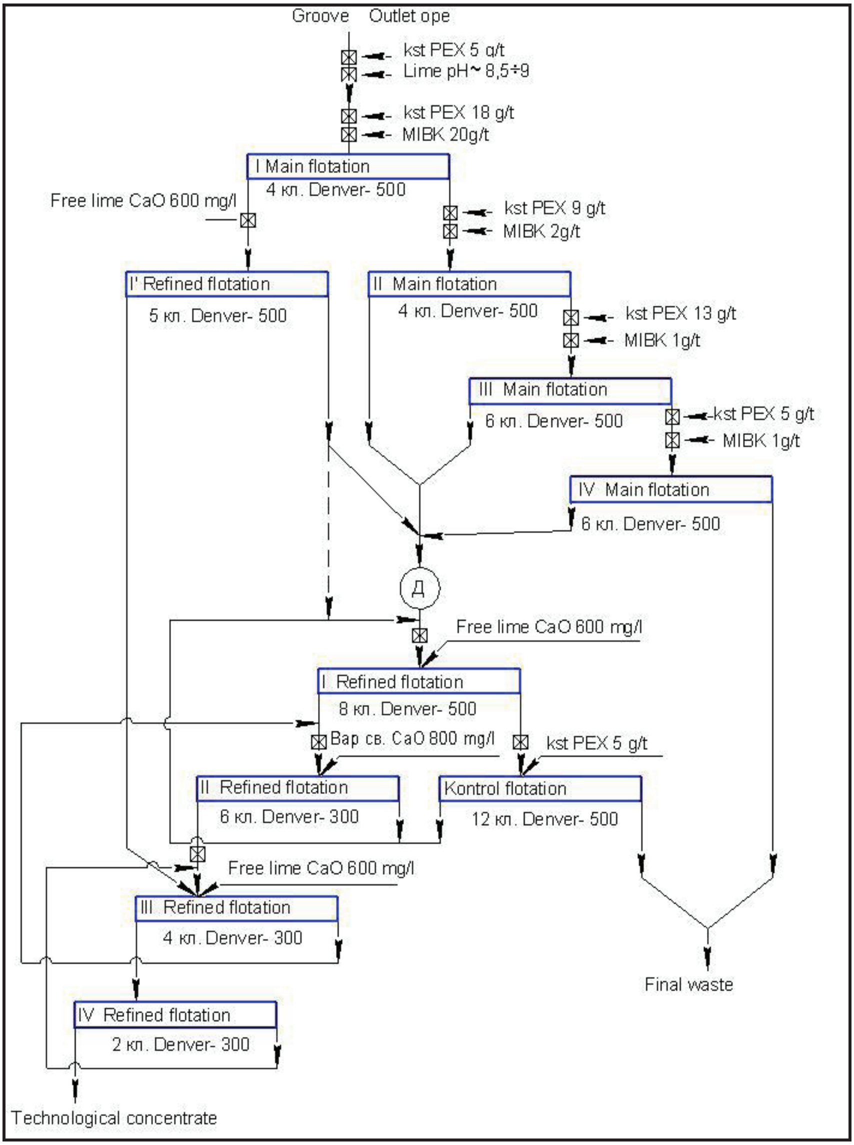

Figure 1. Linear scheme of flotation in enrichment complex “Ellatzite”.

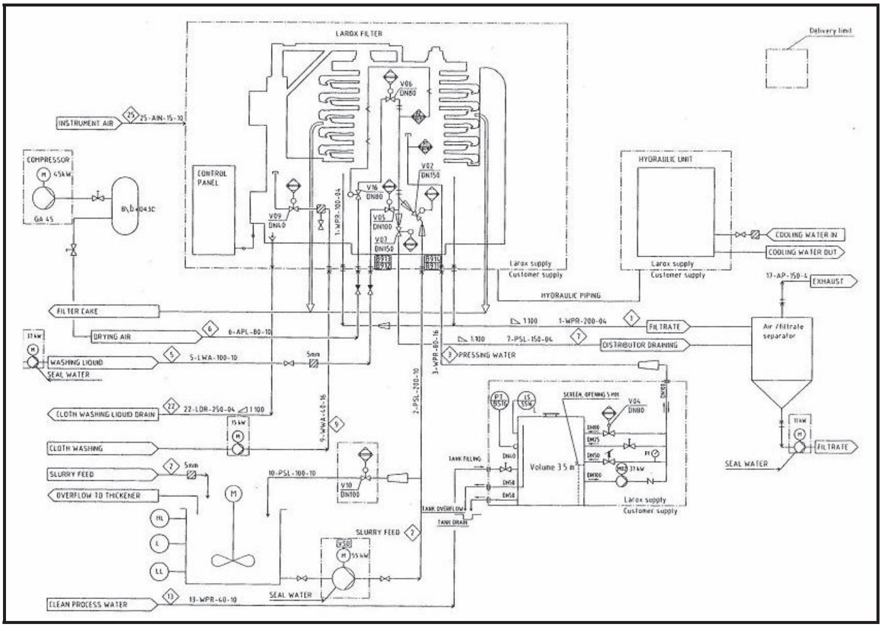

Figure 2. Chain scheme of the apparatus.

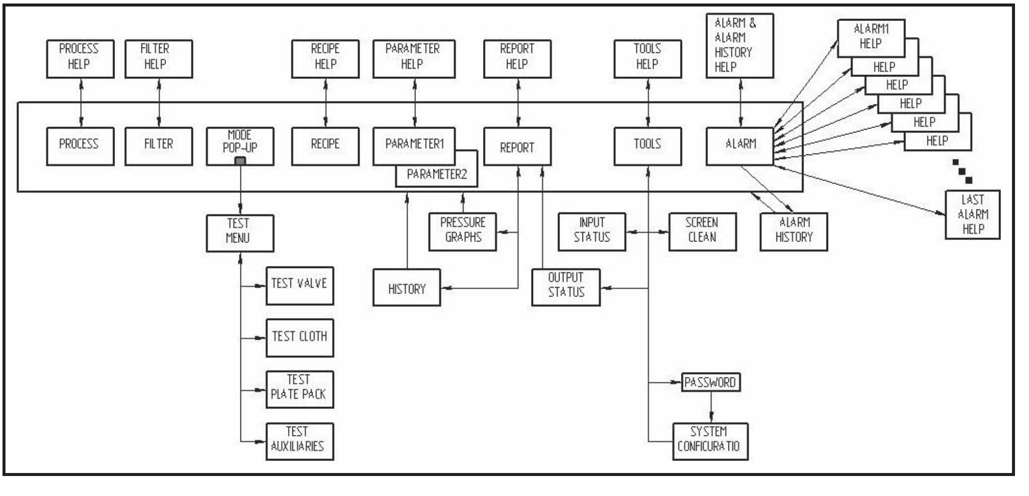

Figure 3. Structure of operator interface.

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Use of the Waste During the Grinding of Copper Ore in the Conditions of the Company "Ellatzite –Med", AD

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ABSTRACT In this article we will consider an auxiliary technology for crushing the ore waste through grinding of copper ore in the enrichment complex of the company "Ellatzite – Med", AD .We have analyzed the conditions for formation of the ore waste and are given its basic technological characteristics .We have presented also a chain scheme of the apparatus in the installation for processing the ore waste using cone crushers with inertia .We have done a technological –economical evaluation of the results of incorporation of this installation in the conditions of the enrichment complex ,the company "Ellatzite –Med", AD .

1 INTRODUCTION

The copper –porphyry deposit "Ellatzite", developed from company "Ellatzite –Med", AD is situated in the central part of Stara planina in the area of town Etropole .The mineralization is in form of veins –they are sprayed type and this mineralization is localized in both sides between granodyorit and schists .The ore is mixed type ; sulphide and sulphide –oxide .The incorporating rocks are presented by schists and granodyorit .

Mineral content of the ore;

- Primary sulphides –pyrite, chalcopyrite, bornite, molibdenite
- Secondary sulphides –chalcocite and covelitte
- Secondary carbonates –malachite and azurite
- Primary oxides –magnetite and hematite
- Secondary copper and iron silicates – chrysocole

The minerals that do not contain ore form rocks .They are as follows;

- For the schists; andaluzite, biotite, chloride, amphibole, epidote, quartz
- For granodiorite –quartz, plagioclase, feldspate, clay materials, carbonates and zeolites.

The main minerals that form the ore are sprayed in the incorporating rocks as pro –veins .The size of the ore grains vary from several micrometers to millimeters and it is mainly in the range 0.1 -0.3 mm .

The content of the types of ores that are processed in the mine complex is as follows;

-granodiorite -81.81%
- dykes -2.08 %
- Schists -16.11%

The average content of Cu in the ore is approximately 0.36 %.

The last stage of the reveal of the grain with minerals –the grinding of the ore is done in the enrichment complex in Mirkovo .The technological regime of the grinding requires the content of the class -0.08mm to be approximately 55-60 % and this of the class +0.2mm to be till 10-12%.

The scheme of grinding of the ore includes only one stage and it is classified in hydrocyclones that work in a closed cycle.

On the territory of the enrichment complex in Mirkovo are done also the technological processes flotation and dehydration of the concentrate –thickening and filtration.

2 TECHNOLOGICAL FULFILL

The grinding of the ore is done in ball mills with central discharge of the milled product; 10 mills 4500x 6000 and 1 mill 3600 x 5500 .The discharge of the milled product is done in a cone with spiral channels in the inner part .The fraction that goes under the sieves through the cone must be classified using hydrocyclones .The overflowed product from the hydrocyclones goes to a flotation and the sands form a circulating stream that must be milled again .The product from the cone that is over the sieves makes a technological stream and it can be identified as ‘ore waste " and it includes more than 95 % ore parts with grain size in the rage 5-25mm .To be able to use the waste the company "Ellatzite –Med " ,AD developed and fulfilled a project for industrial waste crushing .For this purpose is built an installation for intermediate crushing of the ore waste in cone crushers with inertia KID 600 .

The technical and economical advisability from the use of all important mineral components in the ore and also the mechanical stability of the milled ore parts determine the quantity and the grain size of the ore waste .The time that the ore parts stay in the mill influences greatly the physical parameters of the waste .The fragility of the useful minerals in the ore especially of the ores that contains molybdenum menace with excessive grinding of this material and it can lead to big consumption of energy and to big loss of useful component .

These mineralogical and physic mechanical characteristics of the processed ore determine the technological process of grinding and in this process is formed approximately 400 000 tonnes ore waste per year and the six of the particles is in the range 5-25mm and the copper content is approximately 0.25 % .This circumstance imposes the construction of an installation for intermediate crushing of the ore waste in cone crushers with inertia .The installation does the crushing of the ore waste in one stage and the size of the ore particles are in the range 5-25mm and the class is 95% -5.

The technological scheme of the installation is shown on Figure 1.

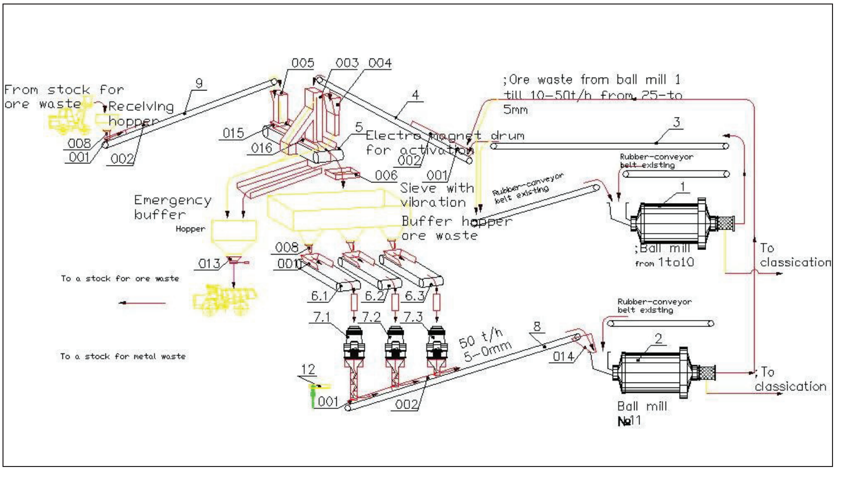


Figure 1. Technological scheme of the installation.

The ore waste, that is separated from 10 ball mills 4500x6000 (Pos.1) is transported through the summed rubber –conveyor belt that exists –(Pos.3) and it is supplied through a mobile groove of the inclined under angle 21 C rubber –conveyor belt (Pos.4). The transportation of the material on this inclined belt is done through chevron rifle furrow of the rubber –conveyor cloth .The inclined rubber –conveyor belt transports the ore waste till a overloaded node situated on level +17.50. If it is necessary the ore waste can be supplied in the feeding of the ball mill N 10 through the inclined rubber conveyor belt that exists.

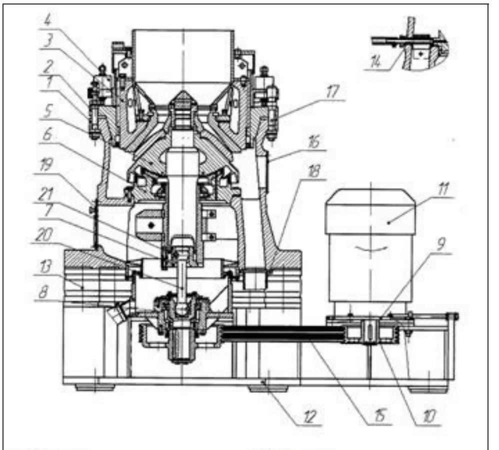
In the overloaded node the crushed material is poured from rubber –conveyor belt N 1 (Pos.4) on a groove (Pos.003) on a horizontal rubber –conveyor belt N 2 (Pos.5), situated on level +15.00. When stopping the rubber –conveyor belt N 2 in an emergency or when a buffer hopper for waste is filled the waste is supplied from an inclined belt N 1 through a groove (Pos.004) in an emergency buffer hopper from where is discharged through a tap (Pos.013) and it is transported to a stock for waste ore .

When the technological line works normally the rubber –conveyor belt N 2 (Pos.5) supplies material in a buffer hopper for ore waste.

The Ferro (metal) objects that are in the waste ore are removed through electromagnet catcher that is mounted on the activating drum of the rubber –conveyor belt N 2 (Pos.5) and on the groove (Pos.007) are poured in the emergency buffer hopper from where they are transported periodically to a stock for metal waste .with the remove of the metal objects is guarantee the normal work of the cone crushers with inertia .

The buffer hopper for waste ore takes the irregularity of the quantity of the material supplied in it and assure a volume for one hour work of the crushing department .When the hopper is fill till upper emergency level is blocked rubber -conveyor belt N 2 (Pos.5) .When the emergency buffer hopper is filled till upper level is blocked rubber –conveyor belt N 1 (Pos.4).

When the installation functions normally the waste flows from the buffer hopper through the grooves –the feeders (Pos.008) over three rubber –conveyor belts N 3 (Pos.6.1, 2 and 3) and they using special grooves (Pos.009) supply the material in crushing department and on level +9.70 are situated three cone crushers with inertia KID 600(Pos.7. 1, 2 and 3) two of them work and the third stay as a reserve .The scheme of the construction of the crusher KID 600 is given on Figure 2.



1- Corps	12- Frame
2- Supporting ring	13- Insulation for vibration
3- Regulating ring	14- Connector
4- Fixing mechanism	15- Gear cam belt
5- Mobile cone	16- Cap
6- Spherical bearing	17- Fixing
7- Unbalanced node	18- Inlet for material
8- Activation	19- Cap
9- Motor plate	20- Spindle
10- Washer for activation	21- Ball
11- Electric motor	

Figure 1. The scheme of the crusher.

The crushed material from cone crushers with inertia is discharged through the grooves (Pos .010 ,011 and 012) on a rubber –conveyor belt n 4 (Pos.8) that transports it through an inclined under angle 14 c estakade till ball mill N 11 (Pos.20 on Fig.1).

The crushers KID 600 (Fig.2) is cone crusher with inertia and its working chamber is formed between the revetment cone surfaces of the mobile cone 5 and the stationary cone 3 .The last is moved vertically through a pair of screws that connects it with corps 1 of the machine .This way can be regulated the width of the exhausting gap of the crusher .The mobile cone 5 is supported by a spherical bearing and it acts as a press around the center of this bearing under the action of the disturbing centrifugal force that is created from an unbalanced mass and it rotates around the bottom lose end of the shaft of the mobile cone .The material in the chamber of the crusher is second mobile “technological support “ of the mobile cone .The reaction due to the influence of the mobile cone fulfills the crushing of the waste .The motor 11 using the gear cam with belt 15 activate the spindle 20 and it execute the rotation of the unbalanced mass .

The size of the material that comes in the crusher is D 80 =40mm and the maximal size of the crushed product is –dmax =5mm. The power of the motor is N motor =75 kW, the nominal productivity of the crusher is Q $=10m^3/h$.

In table 1 is given a representative grain size characteristic of the waste that enters in the crushers KID 600 and in table 2 is given a characteristic of the crushed product.

The degree of crushing I in the crushers KID 600 is relatively stable and it changes in the range –I=4.5-7 depends on the productivity of the machine and the regime of work. For the representative extract shown in tables 1and 2 it is; I=D80; d80=17.62; 3.35=5.26.

3 CONCLUSION

The relatively short period of exploitation of the technological line for crushing the waste shows results that are as follows;

- The grain size and the physic – mechanical parameters of the crushed material are suitable for its processing in crushers KID 600
- The productivity of the machines assure the crushing of the whole volume to the obtained waste when both crushers work simultaneously
- The quantity of the fraction under the sieves is with grain size d, 5mm in the product after the crusher and it varies in the range B =92-96most often is 93-95. A product with such grain size is suitable for grinding in a ball mill N 11 (Pos.2) on Figure 1
- in the processing of the whole annual volume of the waste Q =400 000 tonnes with approximately copper (Cu) content 0.25% and
- Coefficient of extraction k =0.92 the total loss of the extraction of metal for the enterprise will be reduced and the extracted supplementary quantity copper will be approximately 880 tonnes per year.

Table 1. DT=d80=17.62mm.

Classes (mm)	**Extraction (%)**	**Class (+) %**	**Class (-) %**
+25	0.39	0.39	100
-25+20	5.47	5.86	99.61
-20+15	29.69	35.55	94.14
-15+12.5	51.04	86.59	64.45
12.5+8	12.76	99.35	13.41
-8+4	0.13	99.48	0.65
-4+0	0.52	100.0	0.52
Total	**100**		

Before the introducing in exploitation of the technological line for processing the waste using crushers KID 600 and mill N 11 (Fig.1) the waste was supplied for grinding in mill N 10. This reduced greatly the productivity of fresh ore in this mill.

The deviation of the ore that can be milled hard from the feeding of mill N 10 allowed to double the productivity of fresh ore and together with this to increase the productivity of the enrichment complex in village Mirkovo with 5%.

Table 2. DT=d80=3.35mm.

Classes (mm)	Extraction (%)	Class (+) %	Class (-) %
-12.5+8	0.03	0.03	100
-8+4	7.01	7.04	99.97
-4+2.5	28.75	35.79	92.96
-2.5+1.25	64.21	100	64.21
Total	**100**		

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Installation for Crushing and Sorting to Get Crushed Fractions for Rail and Road Construction and for the Production of Concrete

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ABSTRACT The present report has for task to analyze the possibility for constructing a crushing–sieving installation on the ground of mine 'ELLATZITE " to obtain crushed fractions for rail–road construction and road construction and for the production of concrete. The installation will process the raw from the piles with wastes for stock the part that does not contain ore. All the piles for wastes are analyzed to be able to determine physico –mechanical, chemical and petrographic properties of the rocks. Having in mind these data are determined the perspective areas that correspond to the requirements of the standards. With the execution of this project will be used the part that does not contain ore and the ores that are out of balance and will assure possibility to use better the mine in depth.

1 INTRODUCTION

In the process of revealing and extraction of ore from deposit "ELLATZITE" are found million tones part that does not contain ore from rocks with high physico–mechanical parameters. The works with the part that does not contain ore started in 1976 and till now are formed two piles with wastes – Eastern and Western and there are approximately 100 million tonnes from the rock mass.

In connection with the clarification of the possibilities to use the material for crushed fractions for road, concrete and rail pavement ore samples were taken to analyze the rock mass of "Eastern" and "Western" waste piles.

The analyzes are done in accredited laboratories of the "Construction Executive Center" – TPA Ltd ,"Baugraund institute" – KNIRIM Ltd, Direction of "Central Laboratory for Roads and Bridges" to the Scientific Institute for Investigation of Building Materials", The University of Mining and Geology "St. Ivan Rilski", "Roads and Equipments", and the Dutch laboratory "OOMS".

While doing the works with the part that do not contain ore of deposit "ELLATZITE" was distributed in two waste piles: Eastern and Western. The eastern is situated north–east from the cockpit of the mine. The mass consists of granodiorite, schists, hornfelze, porfpyre and other magma rocks. The rock parts are with big physical and mechanical qualities. The quantity of the stock rock mass on the waste piles is approximately 100 million m^3 -15 million m^3 of surface 530 decares in the eastern and 70 million m^3 of surface 1220 decares in the western waste pile respectively and approximately 15 mil.m^3 sterile mass with oxides.

2 CHARACTERIZATION OF THE DEPOSIT

2.1 Mineral Characterization of the Deposit

"ELLATZITE" copper–porphyry deposit is situated in the central part of Stara Planina

Mountain in the vicinity of town Etropole. The sample taking is with veins that are sprayed and it is localized from both sides of the contact between granodiorite and schists. The deposit is crossed from several powerful dykes of granodiorite and quartz diorite profile.

The ore is mixed type: sulphide, sulphide–oxide. The incorporating rocks are present by schists and granodiorite. Mineral composition of the ore:

- Primary sulphides – pyrite, chalcopyrite, bornite, molybdenite
- Secondary sulphides – halcozyn, covelin
- Secondary carbonates – malachite and azurite
- Primary oxides – magnetite, hematite
- Secondary copper and iron silicates chrysocol .

The non-metallic minerals are all rock-formating:

- For schists – andaluzit, biotite, chloride, amphybol, epidot, quartz
- For granodiorite – quartz, plagyoclas, feldshpar, clay minerals, carbonates, zeolites .

The main minerals that form the rocks – chalcopyrite, pyrite, bornite, molybdenite and quartz are sprayed in the incorporating rocks as pro–veins and it is from. The grain size varies from several microns to millimeters and most often it is 0,1 - 0,3mm.

The ores in deposit "ELLATZITE" are divided in three types:

- sulphide ores – with oxide copper to 10 %
- mixed ores –with oxide copper from 10 to 30 %
- oxide ores –with oxide copper more than 30 %

The major source of copper in the sulphide ores is chalcopyrite - about 78%, in the second place is bornite with about 22%. In the mixed ores the main sources of copper are chalcopyrite, bornite, chalcosine, covelline.

Table 1 shows summarized indicators of the physical and mechanical characteristics of the petrographic types of rocks in "Ellatzite" deposit.

Table 1. Physical and mechanical characteristics of the petrographic types of rocks in "ELLATZITE" deposit.

Indicators / Petrograpic type	ρ_s, g/cm^3	ρ_n, g/cm^3	n, %	RH_C, MPa	Ron_c, MPa
Granodiorite	2.55-2.97 2.72	2.53-2.77 2.64	0.38-10.8 3.39	22.7-168.9 82,4	3.4-12.7 9.0
Dyke	2.66-2.78 2.74	2.60-2.74 2.66	26.9-134.5 77.8	4.7-14.7 8.9	4.7-14.7 8,9
Schist	2.68-3.20 2.85	2.50-2.83 2.76	0.73-13.92 3.81	11.6-154.6 79.7	6.9-10.4 9.0

Legend:
ρ_s, - specific thickness; RH_C – power of press in dry condition; n – volume of the pores
ρ_n, - volume thickness; Ron_c – power of tension in dry condition

2.2 Geological Characterization of the Rocks from the Rock Massive of the Deposit

In the limits of the cockpit of the mine "ELLATZITE" in the geological structure and in the part that does not contain ore and in the eastern and western waste pile take part two main groups of rocks :

- Paleosoic metamorphite and magma
- Mezozoi upper creda rocks.

The main rock material in the waste pile consists of: granodiorite, diorite, porphyre, microdiorite, gabbrodiorite, that are contact changed phylite – hornfelse and schists with stains. The quantity of each lythological type in the different waste pile is different and it depends on the revealing in front of the bank rocks where it was developed in the moment.

Phylites – they are revealed in the most southern and south –eastern horizons -1390-1480 next to the surface. They are greydark with fine schists and the texture is also schists.

The contact changed rocks – hornfelse and construct a relatively big body with irregular contours and it is elongated in direction northeast – southeast. The south and the northeast pit edges are revealed in the cockpit–horizons 1090-1440.The complex of dyabase and phylite is composed of clay psamit rocks and in contact change it provokes the formation of typical hornfelse – metasands and metaaleurolite from the contact effects of the imposed thermal pre – crystallization.

Hornfelses are thick and fragile and their texture is with well visible bands, locally with processes of scarn related to the incorporation of granodiorite. In the northeastern part of the cockpit near the granodiorite the hornfelses are contact changed and they are influenced by the intense hydrothermal changes and they appeared like a thick net of cracks filled with quartz – feldspar veins.

Vezhen granodiorites – they are widely spread in the north – eastern parts of the deposit. There is gabbro–diorite on many places among them. In the central and the north–eastern parts of the cockpit the granodiorites are influenced by intense hydrothermal changes.

On some places in the northeastern part of the mine you can found some veins –dyke corpus from microdiorite, gabbrodiorite and diorite porphyre. The upper – creda magma corps are present by quartz – monodiorite and granodiorite porphyre with contours like veins and lens. The first are revealed in the northeast and south parts of the cockpit and are incorporated among the dyabase – phylite. complex, and the second are situated in the north–eastern pit edge of the mine.

The rock mass in the east waste pile is present by parts of granodiorite and hornfelse and in minor quantity granodyorite porphyre and magma veins included in them microdiorite and gabbrodiorite porphyre.

The West waste pile is mainly composed of hornfelse and contact-altered rocks and metasands and metaaleurolites, spotted schists, quartz - monzonite porphyrites and granodiorite.

2.3 Quality Estimation of the Rock Material from the Waste Pile

Having in mind the obtained results from the probes, analyzed in different chemical accredited laboratories is done a characterization of the quality of the rock material from the waste piles in “Ellatzite”.

Using as base the total obtained results compared with the results from the analyzed parameters, required by the Bulgarian State Standard (BDS) 1260 EN 132-42 with the respective Bulgarian standards 2282-83 - crushed stone for road base and asphalt cover, BDS 169-81 – materials for common concrete, BDS 635-96 – crushed stone for rail road, ballast for rail roads and Technical specification of IA “Roads“ is done a characterization of the rock materials from Eastern and Western waste pile, where the different lithological types are mixed from the part of the mine that do not contain ore.

The quantity of both lithological types in both heaps is not constant. To determine the quality of the rock mass, samples from both heaps are tested and analyzed.

The obtained results show that according the main physical – mechanical parameters the materials from both heaps do not differ substantially /Table 2/, that is why the determination of the quality is done for both heaps. The data show that based on the obtained results the materials for both heaps satisfy the requirements of the standards for road pavements, rail pavements and concrete.

Table 2. Indicators for the analyze and the results of them and the permitted values according the different standard documents.

Indicator	Obtained results			Values until the start of the indicator
	Total for the waste pile	Eastern waste pile	Western waste pile	
1	2	3	4	5
1. index of particle type according the mass	14-16, medium value 15	15,3	14	≤15 EN 1260:2002
2. Resistance to fraction in the drums of Los Angeles ,% loss mass	15-21, ,medium value 17.5	19	14	≤ 20 for category LA_{20} EN 1260:2002 From 30 to 50 BDS 169-81 BDS 2282-83 ≤24 to ≤30 BDS 635-96
3. Resistance to wear –Micro deval –mass loss	8-10,medium value 9	9	9	≤ 10 categorization $M_{DE\ 10}$ BDS 635-96
4. Fraction /mass loss	Fraction 5-10мм. – 11.2 10-20мм. – 8.7 20-40мм. – 9.3	11.2 8.7 10.1	8.7 8.6	BDS 169 – 81 ≤ 12
5. Water absorption /mass loss in %	0.3 – 1.1 medium value 0.65	0.85	0.50	BDS 635- 96 - < 0.5
6. Coefficient of polishment in %	54 – 58, medium value 56	55.30	56.75	≥ 50 –Technical specification of IA “Roads ”
7. Cohesion with bitum for fraction 5-10mm ,% conserved surface	33-49, Medium value 44 With add of an activator NTERLENEIN 400 R 0.5- 100 %	42	49	≥ 50 BDS 4132-90
8. .Resistance to ice% mass loss	0.3	0,3	15/25cycles 5-10мм.1/1,1 10-20мм.0.6/0.7 20-40мм.1.6/2.5	≤1for category F_1 EN 126 20: 2002 ≤ 10 BDS 2292 - 83
9. Resistance to fade /mass loss	2.7-6.6 –medium value -4.5	5	2.72	≤ 18 for category MS_{18} EN 1260: 2002
10. Index of the type of the particles ,index of the form ,% according the mass	17-21, medium value 19.5	19	20	≤ 20 for category SI_{20} EN 1260: 2002
11. Oblong and flat grains according the mass	5-10мм. / 8-14.3 10-20мм. / 3.5-10.3 20-40мм. / 2.8-14.8	2.72	2.73	≤15 BDS 169-81 ≤15≤35 BDS 2282-83 ≤ 30 BDS 635-96
12. Content of disolved				BDS EN12620 BDS169 – 81
- SiO_2 mmol/dm^3	0.29 – 43.68, medium value 17.8	22.75	15.31	No more than 50
- SO_3 %		0.28	1.23	BDS 169 – 81 ≤ 1
- nuisible organic additives	0.07 – 2.07, medium value 0.91	- none	- none	BDS 169 - 81
- water soluble chloride ions	- none	0.043	0.043	Declared from the manufacturer BDS En 12620
13. Content of the particles that can be washed	- 0.043 – 0.044 % 0.4 -0.7 Medium 0.55		0.4 – 0.7 Medium 0.55	BDS 169 – 81 ≤1 BDS 2282 – 83 ≤ 3 ≤ 5

3 INSTALLATION FRO CRUSHING AND SORTING

The installation will produce fractions for rail road construction, for roads and for concrete. The laboratory analyzes that were done show that the obtained fractions correspond to the requirements of the standards. A chain scheme of the apparatus is given on Figure 1.

This installation will produce 12 sizes crushed fractions. The biggest fraction will be 80 -40 mm, the smallest fraction 4-0 mm. To get such fractions is done e preliminary sieving of the inlet raw material and it is crushed in three stages with control sieving after each stage.

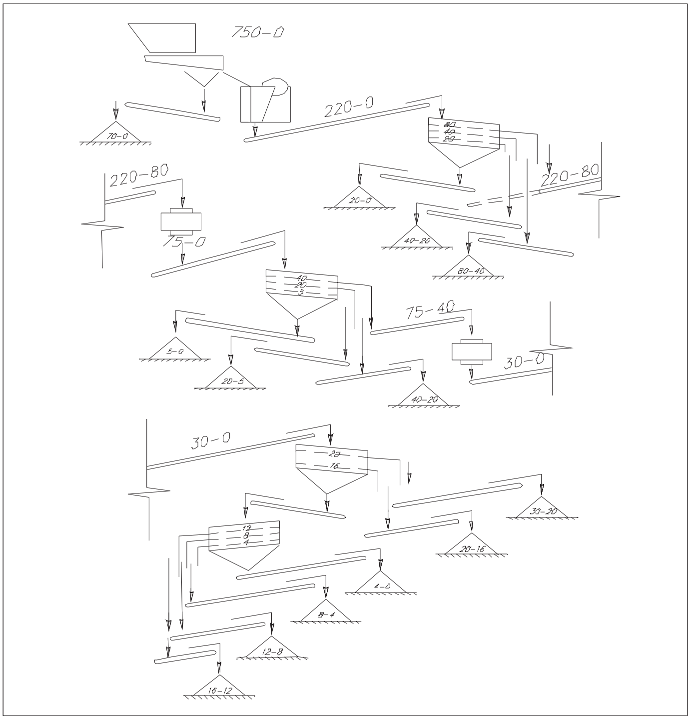

Figure 1. Chain scheme of the machines and the apparatus.

The transportation of the raw material with size 750 – 0 mm is done in auto lorry with load 40 tones. They pour in the receiving hopper of the corps for big crushing of the ore. Under the hopper is mounted a feeder with vibration for dosing the raw material in the installation. A part of the bottom of the feeder is formed like a sieving surface with distance between the beams 70 mm. A fraction is separated from the feeder 70-0 mm and in it is concentrated little class that are faded, the restrictors and other polluters. This way is guarantee ready fractions with determine physico – mechanical properties that correspond to the standards.

The first stage of crushing is done in crusher S 125 with jaw with receiving hole 1250 x 950 mm and it accepts parts with maximum size 750 mm .Under the crusher is mounted a rubber–conveyor belt with width of the cloth 1200 mm. The crushed ore mass goes to be sieved on a sieve with vibration FS 203 with three surfaces for sieving for separating the fractions 80-40 and 40-20 mm. For railroad pavements and base for roads. The fraction 20-0 mm does not correspond to the requirements of the standards concerning the grain size and if it is necessary can be processed supplementary.

For dust removal in the cycle for big crushing and the next sieving is planned to be mounted a sleeve filter with capacity 15000 m /h.

The fraction 250-80 mm and if it necessary the fraction 80-40 mm goes for medium crushing in cone crusher GP 300 with capacity 230 t/h and a receiving hole 250 mm. The crushed rock mass with the aid of rubber–conveyor belt with width of the cloth 1000 mm goes for classification on a sieve with vibration Fs 203 with three sieving surfaces to get fractions for concrete - 40-20, 20-5 and 5-0,5 mm.

The product with size 65-40 mm and if it is necessary 20-40 mm goes for crushing in a cone crusher for little crushing HP 300 with receiving hole 100 mm and capacity 139 t/h. The crushed product goes for sieving on a third sieve and it is done on a sieve with vibration with two sieving surfaces and second with three sieving surfaces. From it the obtained fractions for road construction are 30-20, 20-16, 16-12, 12-8, 8-4 and 4-0. Both crushers are situated in a common hall and there is mounted a crane with loading 5 t for there service.

For dust removal of the crushers and for little crushing and the sieves with vibrations for road fractions is planned a sleeve filter with capacity 30000 m^3/h.

The installation is developed without intermediate buffer volumes .This organization allows easier execution of the building construction and it can be fulfill more rationally on an inclined terrain.

A quality–quantitative scheme of the installation is shown on Figure 2. Based on it and on the working time is composed the production program. The installation will produce 500000 t annually ready fractions and it will work on 1 shift. The production has an oriented character. It depends on the requirements and the needs of the merchant for the corresponding fractions.

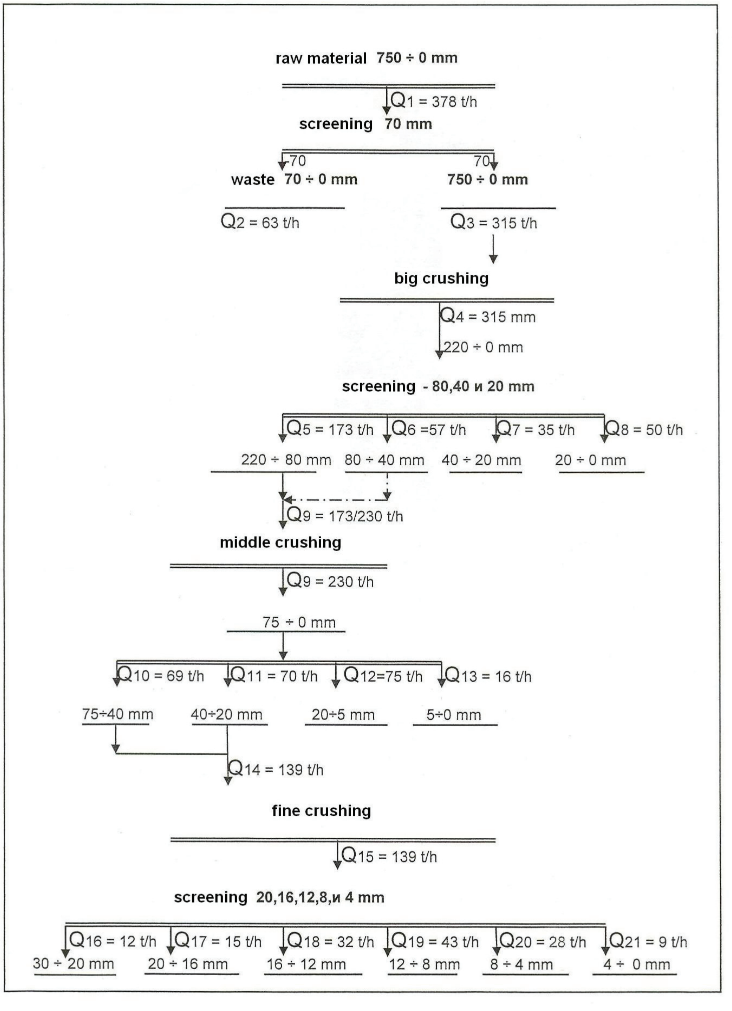

Figure 2. Quality–quantitative scheme of the installation.

4 CONCLUSION

The rock material from the waste piles "Eastern" and "Western" is present by rock parts from different litological varieties in some parts of the waste piles. In the rock mass of the waste piles take part granodiorite, diorite – porphyre, microdiorite, contact changed phylites –hornfelse and schists with stains, macrodiorites and gabbrodiorites.

The investigations that were done according the requirements and the methodology of the European standard and Bulgarian state standards show that as a whole the material from the waste piles is with big physico–mechanical indicators. The quality of it is determined according the Bulgarian state standard 2282 - 83, 169 -81, 635-96, EN 12620:2000 and the Technical specification of IA "Roads". Based on the obtained results we can make the following conclusions.

1. The rock materials from waste piles correspond to the Bulgarian State Standard 2282-83 for road base that are not processed with kinking agents for upper and bottom layer with a very light, light, medium and big category of moving, for asphalt mixtures for mark base 1 and 2, for asphalt mixtures that are used for cover 1, 2 and 3 for upper layer and for bottom layer mark 1 and 2.

According the requirements of the Technical specification 2000 of Executive agency "Roads" and the investigations that were made the fractions with materials can be used for asphalt mixtures for the wearing layer mastic asphalt, type A and wear layer type B with add of an activator INTERLENEIN - 400 r.

2. The gravel from the crushed rock material according all physical and mechanical indicators correspond to the requirements of the Bulgarian State Standard 169-81 for concrete 1, 2, 3 and 4 for mark 300MPa and more .

The content of SO_2 is higher on some places due to the fact that the concrete must be sorted.

3. The crushed and fractioned material from the waste piles corresponds to the requirements of Bulgarian state standard 635-96 for ballast of rail lines from 1 to 6 category with load more than mil tones annually.

4. Having in mind the changes in the content of some lithological varieties in the waste piles is necessary a permanent control of the extraction and it must be done on broader limits and control samples must be taken periodically.

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A New Approach in the Wet Separation of Quartz-Kaolin Raw Material

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ABSTRACT The deposits of quartz-kaolin raw materials are localized in the NE Bulgaria. This raw material is a mixture of quartz sand and kaolin in a ratio approximately 4:1 and is mined in open pits. After wet separation, the raw material yields two products – washed kaolin (different grades) and quartz sand (different grades). The kaolin-silica raw materials from Vetovo open mine and the technological tailings from Vetovo processing plant were characterized. Particle size distribution and chemical composition were determined. It was established that there are a lot of losses of quartz in final tailings. New approach on kaolin-silica raw material separation was tested in laboratory scale.

The test work was performed on the Derrick Stacksizer screening machines – Model 2SG48-60R-1STK single deck Repulp Stacksizer equipped with 0.43 mm opening Derrick Urethane Panels to make a 500 micron size separation and Model 2SG48-60R-1STK single deck Repulp Stacksizer equipped with 0.15 mm opening Derrick Urethane Panels to make a 100 micron size separation. The results of laboratory experiment show that there is a possibility to increase the production of quartz sand grade PK016 and to start the production of a new grade PK010 and finally to increase the quartz sand recovery.

1 INTRODUCTION

The main raw material mined in the open pits of KAOLIN AD and processed in the company's plants are quartz-kaolin sands. They are mined from several deposits situated in the eastern part of the Danube plain. This raw material is a mixture of quartz sand and kaolin in a ratio approximately 4:1.

The geological structure of quartz-kaolin sand consists of carbonate sediments from the early Cretaceous period. In the subsequent raising, the erosion revealed Barramian and Aptian rocks on the earth's surface. These limestones formed a large pleokarstics complex with vast drops. As a result of some adjacent areas erosion, the lowerings were gradually with quartz and feldspar grains. The chemical weathering transformed this material into the now-existing weakly cemented quartz-kaolin sands. A layer of Quaternary red clay and loess covers all of these materials.

KAOLIN AD is a company with the main line of business as the mining of non-metallic minerals. The company is a leader in industrial minerals production in Bulgaria. The main manufactured products are: different grades of kaolin, silica sand, chamotte, feldspar, carbonate fillers and ceramic read body. KAOLIN AD is the first sand producer in Eastern Europe and the fourth largest producer of sand in Europe. The company controls 1 % of the world's kaolin reserves and is the first kaolin producer in South-Eastern Europe and the second chamotte producer in Europe.

For a many years, the quartz-kaolin sand separation has been performed by water washing technology. After wet separation the raw material yields two products – washed kaolin (different grades) and quartz sand (different grades). The manufacturing process includes disintegration; three stages of spiral classification; two stages of hydrocyclone classification and dewatering of kaolin grades; hydroclassification; several stages of sizing using drum screens and dewatering of quartz sand grades.

Research and development program has started to improve the quartz-kaolin sand separation in processing plants. The program has undertaken an increasing of processing capacity, to decrease the losses in the tailings and to provide production of new quartz sand grades. This paper describes some of the results on an experimental work in the frame of the research program.

The purpose of this study was to develop a new approach on the wet separation of quartz-kaolin raw material.

2 EXPERIMENTAL

2.1 Sample Characterization

A number of samplings in two months duration in Vetovo processing plant were performed. Sampling and performance evaluation was applied at two points: quartz-kaolin raw materials from Vetovo open pit (processing feed) and final tails. The samples were collected from each stream every 30 minutes and were combined at the end of the shift. The following characteristics of the sampling products were determined: particle size distribution; chemical composition and mineralogical composition.

Particle size distribution of samples was determined by wet sieving: fraction "+ 45 μm" using vibratory sieve shaker Analysete 3, FRITSCH and fraction "- 45 μm" using SediGraph 5100. The chemical and the mineralogical compositions were determined using AES-ICP and Diffractometer Siemens D 500 respectively.

2.2 Wet Separation Laboratory Tests

The wet screening tests using DERRICK Corporation experimental facilities were conducted. The representative sample of quartz-kaolin raw material from Vetovo open pit was tested. The laboratory flow sheet on wet screening of quartz-kaolin sands is shown in Figure 1.

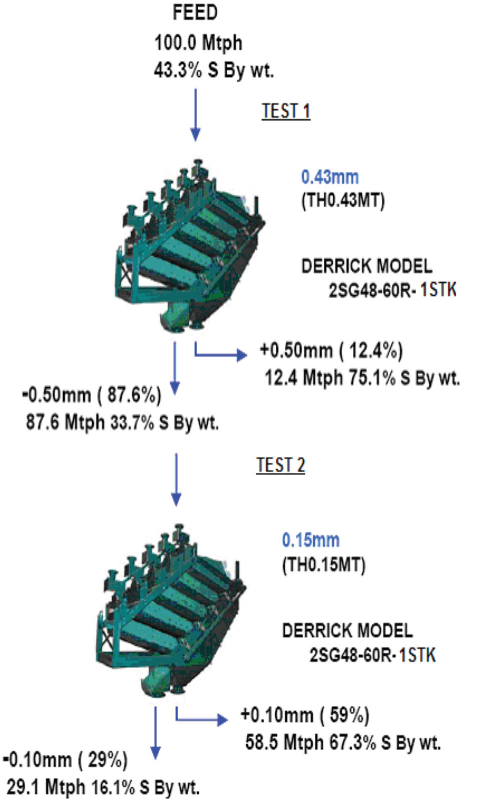

Figure 1. Wet screening laboratory flow sheet.

The test work was performed in two stages using Derrick Stack Sizer screening machines. In the primary stage Model 2SG48-60R-1STK single deck Repulp Stacksizer equipped with 0.43 mm opening Derrick Urethane Panels was used to make a 500 micron size separation at 100.0 Mtph. In the secondary stage Model 2SG48-60R-

1STK single deck Repulp Stacksizer equipped with 0.15 mm opening Derrick Urethane Panels was used to make a 100 micron size separation.

The Derrick Stack Sizer has redefined the concept of efficient, fine particle wet screening. The patented Derrick Stack Sizer consists of up to five individual screen decks positioned one above the other and operating in parallel. The Derrick Stack Sizer unit utilizes two Derrick's "High Speed-low Amplitude"1500 RPM, 2.5 HP, Model "SG" sealed bearing vibrators that do not require external lubrication. The vibrators, running in opposite directions, create a linear motion over the entire screen traverse. This type of action creates positive and providing maximum fluid throughput while solids continuously convey off the machine. The concept of the Derrick Stack Sizer Screen is that the width of a screen is more critical than the screen length or screen area in wet screening operations. The key factor limiting the capacity is the feed slurry volume that could be handled per increment of screen width. For added efficiency, the Repulp Stack Sizer unit incorporates specially designed a repulping trough and spray bar located on the screen surface. As the oversize fraction enters rubber lined wash trough, brine addition via the spray bar effectively wash and re-slurry feed solids. High frequency repulping action tumbles the particles thoroughly to release fines from the coarse particles. Following the repulping action, the slurry repots to the urethane screen surface where fines and water are removed. The High Speed-Low Amplitude, linear motion Repulp screening design, conveys the oversize material away quickly, thinning the load, and increasing screening efficiency and capacity.

3 RESULTS AND DISCUSSION

The data from particle size distribution and chemical analyses of the quartz-kaolin raw material from Vetovo open mine are shown in Tables 1 and 2.

Table 1. Particle size distribution of processing feed.

Particle size fraction microns	Yield wt. %
+ 800	1.3
- 800 + 630	1.2
- 630 + 400	7.3
- 400 + 315	9.5
- 315 + 250	14.6
- 250 + 160	32.8
- 160 + 100	10.1
- 100 + 71	2.7
- 71 + 63	0.7
- 63 + 56	0.6
- 56 + 45	0.5
- 45 + 20	1.5
- 20 + 10	2.0
- 10 + 5	2.7
- 5 + 2	3.4
- 2 + 1	2.4
- 1 + 0.5	2.3
- 0.5	4.4
Total	100.00

Table 2. Chemical composition of processing feed.

Grades	Content, %
SiO_2	91.80
Al_2O_3	5.4
Fe_2O_3	0.16
TiO_2	0.14
CaO	0.049
MgO	0.046
K_2O	0.21
Na_2O	0.029
L.o.i.	2.16

The performed analyses clearly demonstrate that the fraction "- 250 + 63 μm "is approximately half of the quartz-kaolin raw material, i.e. there are possibility to produce quartz sand grades PK016 and PK010.

The particle size distribution and mineralogical data of the technological tails are presented in Tables 3 and 4. The results show that the fraction "-250 + 63 μm" is over 50 % from the technological tails. Practically this fraction passes into the final tails. Therefore the separation of quartz-kaolin raw material is not running effective regarding this fraction.

Table 3. Particle size distribution of technological tails.

Particle size fraction microns	Yield wt. %
+ 250	0.18
- 250 + 160	5.8
- 160 + 100	22.35
- 100 + 71	17.6
- 71 + 63	5.99
- 63 + 56	4.29
- 56 + 45	4.77
- 45 + 25	5.86
- 25 + 10	19.59
- 10 + 5	4.84
- 5 + 2	2.76
- 2	5.97
Total	100.00

Table 4. Mineralogical composition of technological tails.

Minerals	Content, %
Quartz	88
Kaolinite	7
Mica	4
Other	1

The data from sample characterization clearly illustrate that the separation technology is effective for quartz particles over 0.250 mm.

The content of the fraction “-250+100 μm” decreases only 1.7 times, but the content of the fraction “-160 +63 μm” increases 3 times in the final tails.

Table 5. Wet screening test data from primary stage - 0.500 mm size separation

Opening	**Cumulative Percent Retained**			**Weight Recovery**		**Efficiency**		
Microns	**Feed**	**O' size**	**Un' size**	**O' size**	**Un' size**	**O' size**	**Un' size**	**Over all**
1180	0.7	5.3						
1000								
850	1.8	14.4						
710	3.0	23.8						
600	4.7	37.9	0.00					
500	7.6	58.8	0.11	12.5	87.5	97.6	94.4	94.6
425	10.9	78.5	0.4	12.2	87.8	87.9	97.1	96.1
355	16.8	90.5	2.6					
300	24.7	95.0	9.5					
250	36.0	97.1	21.2					
212	49.2	98.3	37.9					
180			54.6					
150	69.0	99.2	61.3					
125			68.7					
75	79.8	99.4	74.5					
25	83.4	99.4	78.9					
Mean				12.4	87.4			

There are a lot of losses of this fraction in the final tails.

Therefore there are a lot of possibilities to increase the production of quartz sand grade PK016 and to start the production a new grade PK010 and finally to increase the quartz sand recovery.

The wet screening test data of quartz-kaolin raw material are presented in Tables 5, 6 and 7.

In the first stage it an undersize product with 0.11 % “+0.50 mm” fraction at a 100.0 tph solid feed rate with a solids slurry of 43.3 % solids by weight was achieved.

Table 6. Wet screening test data from secondary stage - 0.100 mm size separation.

Opening Microns	Cumulative Percent Retained Feed	O' size	Un' size	Weight Recovery O' size	Un' size	Efficiency O' size	Un' size	Over all
500	0.1	0.2						
425	0.4	0.6						
355	2.6	3.9						
300	9.5	13.4	0.32					
250	21.2	31.3	0.64					
212	37.9	57.4	1.29					
180		74.5	2.57					
150	61.3	87.1	8.52	67.1	32.9	95.4	77.7	88.6
125		92.2	17.20					
106	70.4	93.7	23.95	66.6	33.4	88.6	85.8	87.8
90								
75	74.5	94.6	32.80					
25	78.9	95.3	44.5					
Mean				66.8	33.2			

Table 7. Wet screening test data summary of quartz-kaolin raw material from Vetovo open pit

TEST	1	2
SCREENING UNIT	2SG48-60R-1STK	2SG48-60R-1STK
RPM	1500	1500
ANGLE (DEGREE)	22.5	22.5
SCREEN PANEL	TH0.43MT	TH0.15MT
OPENING	0.43	0.15
FEED (M^3/HR)	170.3	109.0
FEED (Mtph)	100.4	44.5
SPRAY WATER (M^3/HR)	45.4	0.0
O'SIZE % Wt	12.4	66.8
U'SIZE % Wt	87.6	33.2
FEED % S by Wt	43.3	32.7
U'SIZE % S by Wt	33.7	16.1
O'SIZE % S by Wt	75.1	67.3
+0.50 mm % IN FEED	7.6	0.1
+0.10 mm % IN FEED	-	70.4
+50 mm % IN U'SIZE	0.11	-
+0.10 mm % IN U'SIZE	-	24.0
-0.10 mm % IN O'SIZE	-	6.3
OVERSIZE % EFF.	97.6	88.6
UNDERSIZE % EFF.	94.4	85.6
OVERALL % EFF.	94.6	87.8

The wet screening efficiency of 500 micron size separation was 94.6 %. In the second stage an oversize product with 6.3 % "-0.100 mm" fraction was achieved with wet screening efficiency of 100 micron size separation being 87.8 %.

The results clearly demonstrate that the wet sizing with Derrick Stacksizer screening machines is a new approach on the wet separation of Bulgarian quartz-kaolin sands.

4 CONCLUSIONS

The separation of quartz-kaolin raw material is not running effective regarding the fraction "-250+63 μm". Practically the half of this fraction passes into the final tails. It appears that wet sizing with Derrick Stacksizer screening machines can be an approach on the wet separation of Bulgarian quartz-kaolin sands. Therefore there is a possibility to increase the production of quartz sand grade PK016 and to start the production of a new grade PK010 and finally to increase the quartz sand recovery.

ACKNOWLEDGEMENTS

Financial support for this study and permission to publish this paper from KAOLIN AD is gratefully acknowledged

REFERENCE

http://www.derrickcorp.com/

Determination of the Optimum Conditions of Roasting and Dissolution of Chalcopyrite in SO_2 Saturated Water

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ABSTRACT The optimum conditions were searched for the extraction of copper from roasted concentrate of chalcopyrite, supplied from the region of Rize (Çayeli), in the aqueous medium saturated by SO_2 gas, by using Taguchi method. In result, because SO_2 forming in roasting was used in the dissolution, an environmentally friendly process protecting the environment from sulphur dioxide was obtained for high copper recovery. Experimental parameters and their ranges were chosen as 20-60^{o}C for reaction temperature; 0,025-0,15 g/mL for solid-to-liquid ratio; 30-90 min for roasting time; 500-700^{o}C for roasting temperature; 400-800 rpm for stirring speed; 10-60 min for reaction time. The particle size and gas flow rate were 230 mesh and 10 cm^3/min, respectively. The optimum conditions of dissolution were determined as 45^{o}C for reaction temperature, 0,025 gm L^{-1} for solid-to liquid ratio, 75 min for roasting time, 500^{o}C for roasting temperature, 400 rpm for stirring speed and 30 min for reaction time. Under these conditions, the yield for the dissolution of copper was found to be 91%.

1 INTRODUCTION

Chalcopyrite ($CuFeS_2$), is an industrially important mineral which forms an essential component of high quality copper ores (Rowson, 2003) which accounting for approximately 70% of copper in the reserves in the world (Cordoba, 2008) Copper has been produced in hydrometallurgical and pyrometallurgical applications. Chalcopyrite is mainly subject to pyrometallurgical treatment after concentration by a flotation process (Whittington, 2008). About 80-85% of the world's copper production is carried out using pyrometallurgical processes (Sokic, 2008) with SO_2 off-gases captured and converted to sulphuric acid (Antonijevic, 2004; Gbor, 2006). Treatments by pyrometallurgical processes are not attractive because copper ores present usually is low grade. Hydrometallurgical methods as an alternative to pyrometallurgical methods for low grade-ores presents important advantages such as effective recycling valuable metals, easy operating, low depletion, environmentally friendly and low cost (Ting-sheng; 2006). At present about 15 - 20 % of the total world copper production is treated by hydrometallurgy processes (Antonijevic, 2004). However, leaching chalcopyrite for copper recovery is currently uneconomic due to inert nature of chalcopyrite during oxidation in aqua solutions (Al-Harahsheh, 2006).

Although hydrometallurgy is put forward as a preferred alternative to pyrometallurgy due to economical and environmental factors chalcopyrite, has a crystalline structure in which iron and copper ions are in tetrahedral coordination with sulfur is, highly refractory under hydrometallurgical conditions, due to surface transformations which render products very stable under oxidizing conditions (Dreisinger, 2002).

SO_2 is a pollutant in the copper production from chalcopyrite concentrates by smelting/roasting. When SO_2 concentration exceeds a maximum value in atmosphere, it is harmful for human beings, animals and plants. On the other hand, when sulfur dioxide combines with water, it forms sulfuric acid, which is main component of acid rain. When acid rain falls it can cause deforestation, acidify waterways to the detriment of aquatic life and corrode building materials and paints. EPA has recommanded that 1-year average concentration of sulfur dioxide should not exceed 0.03 ppm and 24-hour average concentration should not exceed 0.14 ppm. In spite of the significant advances in the conversion of SO_2 to sulphuric acid to comply with the environmental regulations (Padilla, 2007), at present, SO_2 emitted from prometallurgical processes pollutes the environment in especially developing countries. For this reason, efforts have been made to develop process for treating various ores, minerals, or secondary materials (Das, 2000).

The optimization of leaching conditions of the ores is important in industrial processes and in some researches has been interested with these topics by using various techniques. As a technique, Taquchi's orthogonal array (OA) analysis is used to produce the best parameters for the optimum design processes with the least number of experiments. In recent years the main advantages of these methods are that the parameters affecting an experiment can be investigated as controlling and not-controlling and that the method can be applied to the experimental design involving a large number of design factors (Kücük, 2005).

Sokic et al, have studied dissolution kinetics of chalcopyrite by sodium nitrate in sulphuric acid medium. As result, it has been determined that copper leaching increases with increase of temperature, sulphuric and sodium nitrate concentrations, and decrease of stirring speed and particle size. Aydoğan et al. (2006) have determined dissolution kinetics of chalcopyrite in acidic bichromate solutions and found that more chalcopyrite may be dissolved increasing temperature, particle size, the concentrations of sulphuric acid and potassium bichromate. Padilla et al. have investigated dissolution kinetics of copper in sulphited chalcopyrite in presence of H_2SO_4-O_2 under higher pressure and it has been seen that dissolution percent of copper increases with increase of pressure and temperature. Devi et al. have examined chalcopyrite oxidation in medium of mangan dioxide plus hydrochloric acid. In result, they found that chalcopyrite dissolved more in mangan dioxide medium with activation energy of 70.6 kJ/mol and activation energy lowered to 24.8 kJ/mol in the medium of manganese dioxide plus hydrochloric acid. Antonijevic et al. inspected dissolution kinetics of chalcopyrite by using hydrogen peroxide in sulphuric acid solutions and the effects of various parameters on the dissolution. Authors found that reaction rate fitted to $1-(1-X)^{1/3} = k_s t$. Akcil et al. subjected chalcopyrite and pyrite to roasting and then acidic dissolution, and they resulted that because of shortening of leaching time with combination of roasting and leaching operations, this dual application was more preferable than application individually of each operation. Lu et al. studied effect of chloride ions on dissolution of chalcopyrite in acidic solutions and they determined that chalcopyrite dissolved less in free- chloride solutions than chloride-containing solutions. Temur et al. used Taguchi method in determination of optimum process conditions to dissolve chalcopyrite in aqueous solutions saturated with chlorine gas and the optimum conditions designated as 45^{o}C for temperature, 0.05 g/mL for solid-to-liquid ratio, 0.2 g/L for Fe^{3+} concentration, 0.025 g/L for Cu^{2+} concentration and 120 min for reaction time. Havlik et al. used $FeCl_3$+ CCl_4 in dissolution of chalcopyrite concentrate. Sulphur released during the reaction dissolved with CCl_4 and more copper was provided to pass to the solution.

In this study chalcopyrite are was dissolved in SO_2 saturated water by taking into consideration the experimental parameters of reaction temperature solid to liquid ratio, reaction temperature stirring, speed and reaction time, roasting time, roasting temperature and the Taguchi experimental design method was employed to determine optimal dissolution conditions.

It is proposed an environmentally friendly process for copper recovery from roasted chalcopyrite by making use roasting waste gas which is environmentally harmful in this study. In this way, it is contribute to waste gas minimization, occupational safety and health, environmental impact abatement and sustainable development.

2 MATERIALS AND METHODS

2.1 The preparation of materials

The concentrated chalcopyrite used in the experiments was provided from Rize Çayeli region in Turkey. The concentrate was sieved by using a 230 mesh ASTM standard sieve. Chemical analysis of original sample, determined by volumetrically and gravimetrical methods, is given in Table 1. and also XRD of original sample in Figure 1.

Table 1. The chemical analysis of original concentrated chalcopyrite.

Compounds	Cu	Fe	S	Zn	Pb	Al_2O_3	Moisture	Others
Percent	24.88	29.46	34.23	2.19	0.19	0.10	0.90	6.69

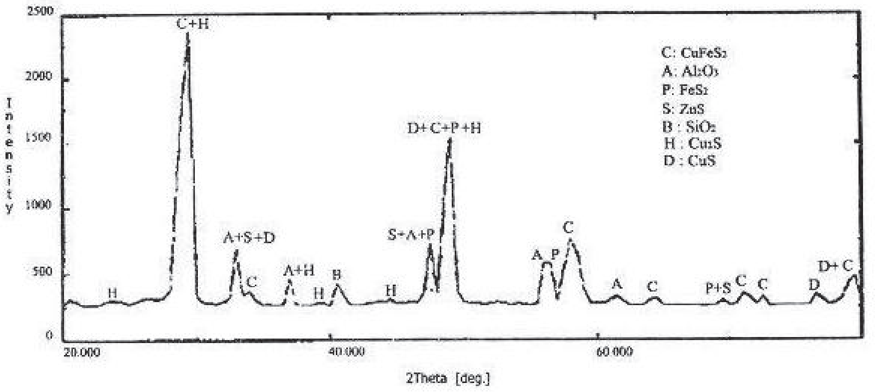

Figure 1. XR-diffractogram of original concentrated chalcopyrite

2.2 Taguchi method

Taguchi method is a systematic application of design and analysis of experiments fort he purpose of designing and improving product quality. The orthogonal array (OA) experimental design was chosen as the most suitable method to determine the experimental plan, L_{25} (5^6), for six, parameters each with five values given Table 3. In order to observe the effects of uncontrollable factors (noise sources) on this process, each experiment was repeated twice under the same conditions at different times (Copur, 1997) Performance value was chosen as the optimization criteria are divided into three categories the larger –the better, the smaller the better- and the nominal the best (Copur, 2007) The performance characteristics were implement by using Eq.1 the larger-the better.

$$SN_L = -10\text{Log}\left[\frac{1}{n}\sum_{i=1}^{n}\frac{1}{Y_1^2}\right] \qquad (1)$$

SN_L is performance characteristic, n the number of repetition done for an experimental combination and Y_i the performance value of the i.th experiment. In Taquchi method, the experiment corresponding to optimum working conditions might not be found in experimental plan table. In such cases, the performance value to for optimum working conditions can be predicted by using the balanced characteristic of OA (Copur, 1997, 2007). For this purpose an additive model can be used Eq.2.

$$Yi = \mu + Xi + ei \qquad (2)$$

Where μ is the overall mean of performance value, X_i the fixed effect of the parameter level combination used in i.th experiment and e_i the random error in i.th experiment.

Because Eq.2 is a point estimation, which is calculated by using experimental data in order to determine whether results of the conformation experiments are meaningful or not, the confidence interval must be evaluated. The prediction error is the difference between the observed Y_i and the predicted Y_i (Copur, 2002). The confidence interval at chosen error level may be calculated by Eq.3 The confidence limits for the prediction error, Se, is

$$S_e = \sqrt{\frac{1}{n_0}\sigma_e^2 + \frac{1}{n_r}\sigma_e^2} \qquad (3)$$

If experimental results are in percent (%), before evaluating eqs 2 and 3 Ω transformation of the percentage values should be applied first using the following Eq.4 Values of interest are then later determined by carrying out reverse transformation by using the same equation

$$\Omega(\text{db}) = -10\text{Log}\left(\frac{1}{P} - 1\right) \qquad (4)$$

where Ω(db) is the decibel value of percentage value subject to Ω transformation and P the percentage of the product obtained experimentally (Phadke, 1989).

$$\sigma_e^2 = \frac{\text{sum of squares due to error}}{\text{degrees of freedom for error}} \qquad (5)$$

$$\frac{1}{n_o} = \frac{1}{n} + \left[\frac{1}{n_{Ai}} - \frac{1}{n}\right] + \left[\frac{1}{n_{Bi}} - \frac{1}{n}\right] + \left[\frac{1}{n_{Ci}} - \frac{1}{n}\right] \qquad (6)$$

Where S_e is the two-standard-deviation confidence limit, n the number of rows in the matrix experiment, n_r the number of repetition in confirmation experiment and n_{Ai}, n_{Bi}, n_{Ci}, … are the replication number for variables quantity level A_i, B_i, Ci,… If the prediction error is outside these limits, it should be suspected of the possibility that the additive model is not adequate. Otherwise, it can be considered that the additive model is adequate. A verification experiment is a powerful tool for detecting the presence of interactions among the control parameters. If the predicted response under the optimum conditions does not match the observed response, then it implies that the interactions are important. If the predicted response matches the observed response, then it implies that the interactions are probably not important and that the additive model is a good approximation.

The order of experiments was obtained by in sending parameters into columns of OA L_{25} (5^6), chosen as the experimental plan given in the Table 3.The order of experiments was made random in order to avoid noise sources which had not been considered initially and could occur during an experiment and affect results in a negative way (Copur, 2002). The interactive effects of parameters were not taken into account in the theoretical analysis because some preliminary tests showed that they could be neglected. The validity of this assumption was checked by confirmation experiments conducted at the optimum conditions. In general, this assumption is not easily made, nor should it be taken lightly. We should not eliminate certain interactions from the model without conclusive evidence that is the appropriate to do so. A procedure advocated by some experiments is to test the interactions found to be in signification and than assume that there interaction are zero when testing other effects in the same experiment. Although sometimes done in practice, this procedure can be dangerous, because any decision regarding an interaction is subject to errors. A variation of this idea is to pool the mean squares in the analysis of error with more degrees of freedom. A sure way to determine if a factor or interaction effect should be pooled is to perform a test of significance. The process of disregarding an individual factor's contribution and then subsequently adjusting the contributions of the other factors is known as pooling. By pooling, the error term is increased and in comparision the other factors, appear less influential. Pooling is recommend usually accomplished by starting with the smallest sum of squares and continuing with the ones having successively larger effects. Pooling is when a parameter is determined to be significant by performing a test of significance against the error term at a desired confidence level. Approaching the matter technically, one could test for significance and pool all parameter influences below the 90% confidence level. A general guideline for when to pool is obtained by comparing error degrees of freedom (DOF) with total parameter DOF. Taguchi recommend pooling parameters until the error DOF is approximately half the total DOF of the experiment. Taguchi guideline for pooling reguires a start with smallest main effect and successively includes larger effects, until the total pooled DOF. The larger DOF for the error term, as e result of pooling, increase the confidence level of the significant parameters (Roy, 1990).

2.3 Experimental

In these studies, reaction temperature, reaction time, solid-to-liquid ratio, stirring speed, roasting temperature and roasting time were chosen as parameters. Parameters and their ranges are given in Table 2.

Table 2. Parameters studied in experiments and their levels.

Parameters	Levels				
	1	2	3	4	5
A. Reaction temperature(°C)	15	25	35	45	60
B. Reaction time(min)	10	20	30	40	50
C. Solid-Liquid ratio(g mL^{-1})	0.025	0.05	0.075	0.1	0.15
D. Stirring speed(rpm)	400	500	600	700	800
E. Roasting temperature(°C)	500	550	600	650	700
F. Roasting time(min)	30	45	60	75	90

The experiments were carried out according to a experimental plan in Table 3 prepared considering $L_{25}(5^6)$ orthogonal array.

Table 3. $L_{25}(5^6)$ orthogonal array experimental plan.

Exp. No.	A	B	C	D	E	F
1	1	1	1	1	1	1
2	1	2	2	2	2	2
3	1	3	3	3	3	3
4	1	4	4	4	4	4
5	1	5	5	5	5	5
6	2	1	2	3	4	5
7	2	2	3	4	5	1
8	2	3	4	5	1	2
9	2	4	5	1	2	3
10	2	5	1	2	3	4
11	3	1	3	5	2	4
12	3	2	4	1	3	5
13	3	3	5	2	4	1
14	3	4	1	3	5	2
15	3	5	2	4	1	3
16	4	1	4	2	5	3
17	4	2	5	3	1	4
18	4	3	1	4	2	5
19	4	4	2	5	3	1
20	4	5	3	1	4	2
21	5	1	5	4	3	2
22	5	2	1	5	4	3
23	5	3	2	1	5	4
24	5	4	3	2	1	5
25	5	5	4	3	2	1

First, the concentrate was roasted according to the conditions in the experimental plan. Roasting process was carried out on a 5cmx25cm dimension-stainless steel tray in ash oven. Predetermined amount of concentrate chalcopyrite was spread out 2mm thick into the tray. A constant air flow of 60 cm^3/min was pumped into the oven and roasting process carried out at predetermined time and temperature. Cu^{2+} and Fe^{3+} contents of the samples obtained in roasting experiments were analyzed. These results are seen in Table 4. In addition to these, DTA and TG measurements were carried out using a thermal analyzer, Netzsch STA 409 PC/PG. For this, 50 mg chalcopyrite concentrate was used and measurements were recorded sending air of 90 mL/min under ambient conditions, at heating rate of 10K/min, over the temperature range of 298-1173 K. Alumina was used as reference material. The result is seen in Figure 2.

The dissolution experiments were carried out in a jacketed glass reactor of 250 mL volume equipped with a mechanical stirrer to control stirring speed and a thermostat to control the temperature of the reaction medium within ±1 ^oC. First 200 mL of water was put into the reactor and SO_2 gas at a constant flow rate of 10 cm^3/min fed into the reactor; then when a desired temperature of the reactor content was reached, a predetermined amount of the roasted concentrate was added into the solution while the contents of the vessel was stirred at a certain speed. At the end of the reaction period, the content of the vessel was filtered, and amounts of Cu^{+2} and Fe^{+3} passing through the solution were then analyzed by a volumetrical method. These results given in Table 5. were used in calculating the performance characteristics.

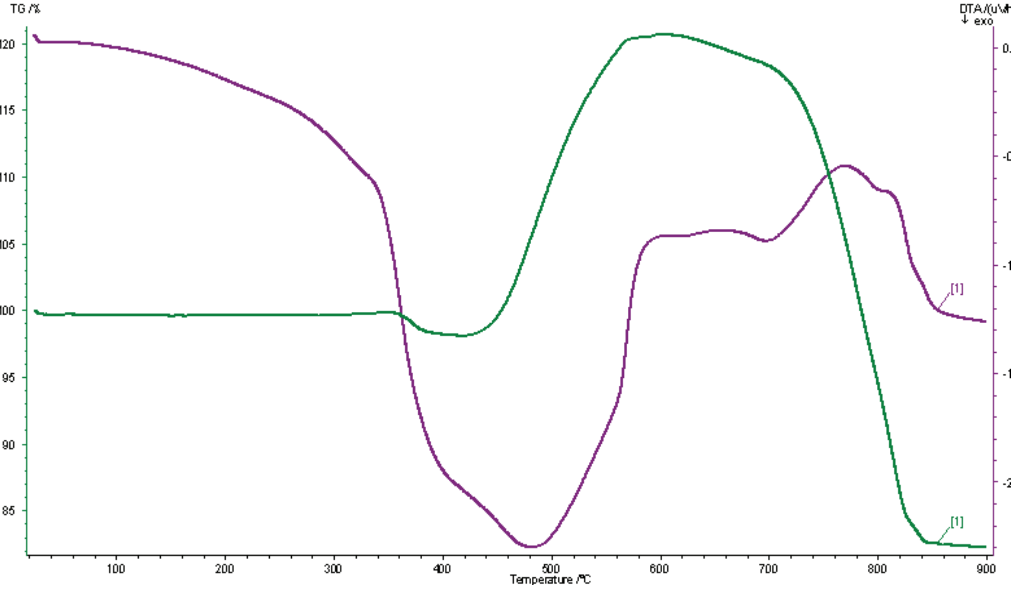

Figure 2. 10 K/min DTA- TGA analyzer.

Table 4. Cu^{2+} and Fe^{3+} contents of the samples obtained in roasting experiments

Temperature (°C)	Time (min)	% Cu Test 1	% Cu Test 2	% Cu Average	% Fe Test 1	% Fe Test 2	% Fe Average
500	30	24.52	25.29	24.911	25.33	25.66	25.495
500	45	24.64	24.33	24.485	25.79	25.79	25.79
500	60	25.41	24.79	24.79	25.94	25.94	25.94
500	75	24.17	23.55	24.86	25.40	24.86	25.13
500	90	24.96	26.96	24.96	26.13	26.13	26.13
550	30	25.38	25.38	25.38	27.01	27.01	27.01
550	45	24.85	24.85	24.85	27.10	26.55	26.83
550	60	24.94	24.32	24.63	27.13	24.42	25.78
550	75	25.29	26.52	25.91	27.19	27.73	27.46
550	90	25.71	25.39	25.55	26.48	26.21	26.35
600	30	25.88	25.88	25.88	27.48	27.48	27.48
600	45	25.92	25.30	25.61	27.74	27.74	27.74
600	60	26.91	26.58	26.95	27.93	29.85	28.89
600	75	26.19	26.11	26.15	27.35	27.35	27.35
600	90	27.10	25.85	26.48	29.10	27.45	28.275
650	30	26.29	25.98	26.14	28.18	28.18	28.18
650	45	27.35	27.05	27.12	28.66	29.19	28.93
650	60	27.84	27.84	27.84	30.14	29.88	30.00
650	75	27.16	27.76	27.46	28.98	27.92	28.45
650	90	27.35	26.75	27.05	30.46	29.67	30.07
700	30	30.96	32.45	31.51	31.79	33.39	32.59
700	45	29.74	29.18	29.46	31.42	34.13	32.78
700	60	31.42	30.87	31.15	32.65	31.65	32.15
700	75	31.96	30.25	31.05	33.05	34.17	33.61
700	90	34.41	33.87	34.14	34.60	34.60	34.60

Table 5. The amounts of copper and iron dissolved in the experiments.

Experiment No	Test 1 Cu%	Test 2 Cu%	Test 1 Fe%	Test 2 Fe%
1	73.20	72.30	10.00	8.00
2	73.20	74.60	6.00	2.00
3	72.05	73.23	2.00	2.00
4	59.20	59.75	2.00	2.00
5	27.90	26.30	2.00	1.00
6	62.85	62.43	1.00	2.00
7	36.70	35.95	1.00	5.00
8	80.05	80.23	8.00	7.00
9	75.20	74.55	5.00	5.00
10	76.60	76.50	6.00	6.50
11	76.15	73.38	6.00	5.50
12	79.00	79.35	5.50	4.00
13	66.05	67.43	6.00	4.50
14	52.40	54.75	6.00	6.50
15	81.04	80.72	20.10	21.00
16	38.65	37.48	1.00	1.50
17	81.70	83.95	16.00	13.00
18	84.50	85.25	13.50	14.00
19	81.40	80.55	16.00	16.50
20	82.30	80.30	12.00	17.00
21	41.30	40.25	17.00	19.00
22	77.55	78.13	19.00	21.50
23	50.50	50.65	28.00	15.00
24	73.70	74.60	30.00	25.00
25	72.30	71.25	30.00	33.50

3. RESULTS AND DISCUSSION

3.1. Roasting and dissolution reactions

3.1.1. Roasting and dissolution reactions

DTA curve in Figure 2 had exothermic peaks in low temperatures (713-833 K) and endothermic peaks in high temperatures (833-1113 K). According to Sokic et al. exothermic peaks corresponding to oxidation of sulphides and endothermic peaks to dissociation of sulphates and oxy-sulphates which form during oxidation of sulphides. Here the occurring reactions are as follows (Copur, 2002):

$$2CuFeS_2 + O_{2(g)} \rightarrow Cu_2S + 2FeS + SO_{2(g)} \quad (7)$$

$$2FeS_2 + 2O_{2(g)} \rightarrow 2FeS + 2SO_{2(g)} \quad (8)$$

$$FeS + 2O_{2(g)} \rightarrow FeSO_4 \quad (9)$$

$$Cu_2S + SO_{2(g)} + 3O_{2(g)} \rightarrow 2CuSO_4 \quad (10)$$

$$2Cu_2S + 5O_{2(g)} \rightarrow 2CuO.CuSO_4 \quad (11)$$

$$CuSO_4 \rightarrow CuO + SO_{3(g)} \quad (12)$$

$$CuO.CuSO_4 \rightarrow 2CuO + SO_{3(g)} \quad (13)$$

Sulphur dioxide easily dissolves to form H_2SO_3 in water. The solubility takes place through formation of two ions; first bisulphate (HSO_3^{-}) and then sulphite (SO_3^{-2}). The pH of the solution varies depending on the equilibrium between HSO_3^{-} and SO_3^{-2}. Distribution of sulphur(IV) species at various pH has been investigated by Gbor et al. According to the data reported, dissolved SO_2 is the dominant species for pH smaller than 1.5, HSO_3^{-} at pH between 1.5 and 6.5, and SO_3^{-2} at pH above 6.5. The chemical reactions during the dissolution of SO_2 in water are as follows(Copur, 2002).

$$SO_{2(g)} + H_2O \rightarrow H_2SO_{3(aq)} \quad H=1.23 \ atm^{-1} \quad (15)$$

$$H_2SO_{3(aq)} + H_2O \rightarrow H_3O^{+}_{(aq)} + HSO^{-}_{3(aq)} \quad (16)$$

$$H_2SO_{3(aq)} + H_2O \rightarrow H_3O^{+}_{(aq)} + SO^{-2}_{3(aq)} \quad (17)$$

A catalytic action of Cu(II) has been reported during the leaching of iron (III) oxides with SO_2. This is believed to be due to the formation of cuprous sulfite species, which are adsorbed on iron oxide surface and behave as an electron transfer catalyst by reducing the lattice from Fe(III) to Fe(II) (Das, 2000; Hiroyoshi, 2007). As results, when roasted chalcopyrite dissolved in SO_2-saturated water, the following reactions take place in solution.

$$CuSO_{4(s)} \rightarrow Cu^{2+}_{(aq)} + SO^{2-}_{4\,(aq)} \quad (18)$$

$$SO_{2(g)} + H_2O \rightarrow H_2SO_{3(aq)} \quad (19)$$

$$CuO_{(s)} + 2H_2SO_{3(aq)} \rightarrow Cu^{2+}_{(aq)} + 2HSO^{-}_{3(aq)} + H_2O \quad (20)$$

$$Fe_2O_{3(s)} + 2H_2SO_{3(aq)} \rightarrow 2Fe^{2+}_{(aq)} + 2SO^{-2}_{4(aq)} + SO^{2-}_{3(aq)} + 2H_2O \quad (21)$$

$$2CuO_{(s)} + H_2SO_{3(aq)} \rightarrow 2Cu^{+}_{(aq)} + SO^{2-}_{4(aq)} + H_2O \quad (22)$$

The collected data were analyzed by an IBM compatible PC for evaluation of the effect of each parameter on the optimization criteria. In order to see effective parameters and their confidence levels on the dissolution process, an analysis of variance was performed to see which process parameters are statistically significant. F-test is a tool to see which process parameters have a significant effect on the dissolution value (Copur, 2002). The F value for each process parameter is simply a ratio of mean of the squared deviations to the mean of the squared error. Usually, the larger the F value, the greater the effect on the dissolution value due to the change of the process parameter. With the performance characteristics and ANOVA analyses, the optimal combination of process parameters can be predicted. The results of variance analysis are given in Table 6. The last column of the ANOVA Table 6 indicates the percent contribution of the process parameter.

According to Table 6, parameter E contributes the most 66.074% and that of parameter F (0.73%) not significant. When the contribution of a parameter is small, as for F in this study, the sum of squares for that parameter is combined with error. The result of the error of varience is zero F value cannot be calculated (Roy, 1990). To increase the statistical significance of important parameters, those factors with small variances should be pooled. The result can be shown in the ANOVA Table 7 with effect of the parameter pooled. In this study, parameter F is pooled.

Table 6. Result of the analysis of variance for dissolution of values of Cu.

	Parameters	Degrees of freedom	Sum of squares	Mean squares	F value	Percentage contribution
A	Reaction temperature (^{o}C)	4	664.03	166.01	-	9.62
B	Reaction time (min)	4	591.41	147.85	-	8.57
C	Solid-to-liquid ratio (gL^{-1})	4	611.57	152.89	-	8.86
D	Stirrinp speed (rpm)	4	423.81	105.95	-	6.14
E	Roasting temperature (^{o}C)	4	4559.67	1139.92	-	66.07
F	Roasting time (min)	4	50.37	12.59	-	0.73
	Error	0			-	
	Total	24	6900.86		-	

Table 7. Result pooled of the analysis of variance for dissolution of values of Cu.

	Parameters	Degrees of freedom	Sum of squares	Mean squares	F
A	Reaction temperature (^{o}C)	4	664.03	166.01	12.15
B	Reaction time (min)	4	591.41	147.85	12.51
C	Solid-to-liquid ratio (gL^{-1})	4	611.57	152.89	11.89
D	Stirrinp speed (rpm)	4	423.81	105.95	7.55
E	Roasting temperature (^{o}C)	4	4559.66	1139.92	87.31
F	Roasting time (min)	POOLED			
	Error	4	50.36	12.59	

As can be seen in Table 7 for dissolution of Cu, it has been found that respectively roasting temperature, the reaction temperature, the solid-to-liquid, the reaction time and the stirring speed have significant effects on the dissolution process while the roasting time having no effect within the working range.

To obtain optimal dissolution performance, the larger the better performance characteristic in Eq.1 has been taken for dissolution of Cu. The degrees of the influences of parameter on the performance characteristics are given at the graph in Figure 3. According to this, the optimal level of a process parameter is its level corresponding to the highest performance characteristic.

If the experimental plan given in Table 3 is studied carefully together with Table 2, it can be seen that experiments corresponding to the working conditions in Table 8 have not been carried out during the planned experimental work in Table 3. Thus, it should be noted that the dissolution percentages in Table 8 are predicted results obtained by using Eq. 2 and Eq .3 and observed results for the same conditions. Also, results in Table 8 are with in the 95% significance level confidence interval of predictions. In order to test the predicted results, confirmation experiments were carried out twice at the same working conditions. The corresponding two standart-deviation confidence limits for the prediction error are ±8 (Phadke, 1983). This case states that there is a good agreement between the predicted values and experimental values, and interactive effects of parameters are indeed negligible. It may be concluded that the additive model is adequate for describing the dependence of this dissolution process on the various operational parameters.

In each graph, the numerical value of the maximum point is corresponds to the best value for that parameter. These values are seen to be A_4, B_3, C_1, D_1, E_1, and F_4 in Figure 3. The optimum conditions are 45°C, for reaction temperature, 30 min for reaction time 0.025 g mL^{-1} for solid-to-liquid ratio, 400 rpm for stirring speed, 500°C for roasting temperature, and 75 min for roasting time. Under these conditions copper extractions of approximately 91% were achived while iron extraction of approximately 25%.

On the other hand, for economic considerations, it is desirable that the dissolved amount of Fe is minimum and solid-to-liquid ratio is high. For this reason, we wanted to investigate how Cu recovery would change by changing solid-to-liquid ratio, which make up most of the total cost so that the total cost would be reduced significantly. Thus, we selected the values of the solid-to-liquid ratios higher than the optimum solid-to-liquid ratio value. Three series of experiments were carried out and the results analyzed statistically are given in cases 2–3 columns of Table 8.

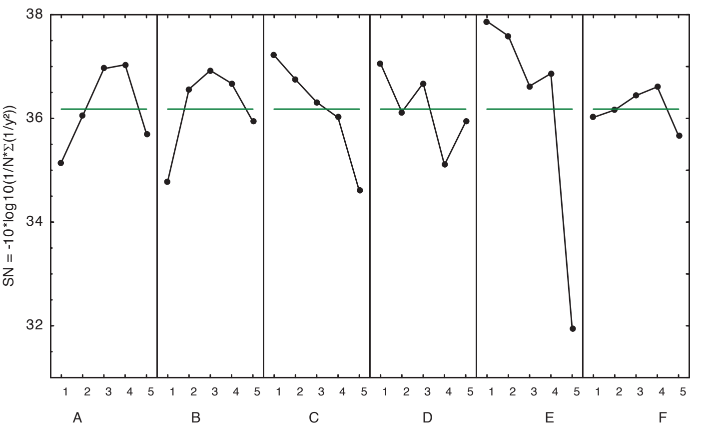

Figure 3. The effect of each parameter on the optimization criteria for Cu.

Table 8. Optimum working conditions and alternative working conditions for four different experimental conditions, observed and predicted dissolved quantities of roasted chalcopyrite.

Parameters		*Optimum Conditions 1		Optimum Conditions 2		Optimum Conditions 3		Optimum Conditions 4	
		Value	level	Value	level	Value	level	Value	level
A	Reaction temperature (°C)	45	4	45	4	45	4	35	3
B	Reaction time (min)	30	3	30	3	30	3	30	2
C	Solid-to-liquid ratio (g mL^{-1})	0.025	1	0.075	3	0.15	5	0.15	4
D	Stirrinp speed (rpm)	400	1	400	1	400	1	400	1
E	Roasting temperature (°C)	500	1	500	1	500	1	500	3
F	Roasting time (min)	75	4	75	4	75	4	75	4
Observed dissolved quantity for Cu (%)		91		86		83		79	
Predicted dissolved quantity for Cu (%)		92		87		86		82	
Observed dissolved quantityfor Fe (%)		25		22		11		4.75	

4 CONCLUSIONS

- In this laboratory study, chalcopyrite was selectively dissolved in SO_2 solutions, and Taguchi experimental design method was used to determine optimum leaching conditions.
- The most important parameter on the dissolution of chalcopyrite in SO_2-saturated water is roasting temperature while the roasting time having no effect within the working range.
- The optimum conditions are 45°C, for reaction temperature, 30 min for reaction time 0.025 g mL^{-1} for solid-to-liquid ratio, 400 rpm for stirring speed, 500°C for roasting temperature, and 75 min for roasting time. Under these conditions copper extractions of approximately 91% were achived while iron extraction of approximately 25%.
- High solid-liquid ratio and lower iron in solution is preferred, the optimum conditions are 35°C for reaction temperature, 30 min for reaction time 0.15 g mL^{-1} for solid-to-liquid ratio, 400 rpm for stirring speed, 500°C for roasting temperature, and 75 min for roasting time. Under these conditions copper extractions of approximately 79% were achieved while iron extraction of approximately 4.75%.
- The predicted and observed dissolution values are close to each other, so it may be concluded that the additive model is adequate for describing the dependence of dissolution process on the various parameters.
- It is hoped that this process is an environmentally friendly process for copper recovery from roasted chalcopyrite by making use roasting waste gas which is environmentally harmful.

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An Overview of the Rheological Behavior of Drilling Fluids

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ABSTRACT Investigation the rheological behaviour of fluids used in drilling process is a crucial subject since it directly determines the efficiency of the drilling operation. The thixotropic behavior of fluids is a convenient property in the drilling process. Similarly, viscosity and viscoelasticity of the drilling fluids are other important properties that should be considered carefully through the drilling operation. It is possible to modify the rheological properties of these systems by some organic/inorganic additives, by changing solids concentration, temperature and pressure. It is the aim of this paper to review parameters that may influence the rheology of drilling muds. In the article after a general introduction about the drilling fluid technology, the general topics of particulate suspension rheology and discussion on the rheological behavior of water based drilling fluids are presented. The effects of various additives on these properties are also discussed in the following sections.

1 INTRODUCTION

Drilling fluids (or muds) are used to control subsurface pressures, lubricate the drill bit, reduce friction, stabilize the well bore, clean the well (Hamed and Belhadri, 2009), carry the cuttings to the surface (Caenn and Chillingar 1996), reduce sedimentation of the cuttings, create a non-diffusive layer around the bore and protect it against corrosion (Fabbri and Vidali, 1970). During an operation stop, it is desirable that the drilling fluid suspends cuttings during a certain period of time (Perez et al. 2004).

It is known that success of the drilling fluid systems mainly depends on the physical and chemical properties of the muds, the type and the concentration of the additives used, properties of the cuttings and the fluid phase utilized during drilling operation. Among the properties of the water based drilling fluids the density, viscosity, yield stress, thixotropy, lubricating and wetting potential are most important parameters that may affect the efficiency of the operation (Robinson 2004; Kelessidis et al.2006).

Bentonite, a smectite type clay mineral, is highly available for the usage in the formulation of drilling fluids due to its swelling, colloidal and rheological properties (Duman and Tunc, 2009). The flow characteristics of water based muds are largely governed by the electrochemical properties of the colloidal bentonite clay particles that form a network with certain strength. When compared to drilling fluids used in the petroleum industry, the bentonite concentration is generally higher (5–10 wt%) and as a result, the shear thinning and thixotropic behaviour of this type of suspension is more emphasized in tunnelling fluids (Talmon and Huisman, 2005).

In this article some aspects of the rheological behavior of water based drilling fluids were summarized. A special attention was given for the discussion about the properties of water based muds containing colloidal solid particles and additives.

1.1 Drilling Fluids

There are different types of drilling fluids. Some wells require that different types be used at different parts in the hole, or that some types be used in combination with others. The main categories of drilling fluids are water based muds, non aqueous muds, usually called oil based mud, and gaseous drilling fluids.

In most of the drilling operations water based muds are used (Caenn and Chillingar, 1996). Water based muds can be formulated with fresh water with or without added salts. They are generally used in the drilling operation of the water bores. There are a great number of salt-type muds formulated by adding a specific salt to fresh water, or sea water (Caenn and Chillingar 1996).

Water based drilling fluids generally consist a liquid phase as well as a colloidal solid phase and cuttings obtained through the drilling process. Clays (Luckham and Rossi 1999), bentonite, silt, and sand are added to the drilling fluids to support the walls of the borehole. Furthermore, cuttings needed to be brought to the surface by the circulating fluid and use of water is insufficient for this purpose. To overcome this problem clays are utilized for thickening of the circulating fluid by increasing viscosity. Table 1 gives some examples of water based drilling fluids.

Bentonite is added to water based drilling fluids in quantities varying between 3 to 7 wt.% (Kelessidis et al., 2006). The high clay concentration in drilling fluids greatly reduces the rate of penetration. Additionally, it may increase the chance of sticking and is the major cause of excessive torque and drag. On the other hand, at low solid concentration, bentonite clay is unable to provide necessary rheological properties required for a good performance especially in oil well drilling. Therefore, polymers are added to achieve the desired result (Mahto and Sharma, 2004). Some of the chemicals used in the mud formulations includes esters, ethers, polyalphaolefins, glycols, glycerines (Caenn and Chillingar 1996; Nelson 1982). Surfactants are also utilized in mud preparation and have major influence on the dispersion, gel formation, wetting and lubrication properties.

Table 1.Common types of water based drilling fluids (Robinson, 2004; Hoshan 2007, Perez et al. 2004; Guo et al.2006).

Type	Principle component
1.	Fresh water
2.	Seawater
3.	Water, bentonite
4.	Salt water, polymer, bentonite
5.	Fresh water, polymer
6.	Water, polymer, silica sand
7.	Seawater, clay, starch, cellulosic polymer
8.	Fresh water, bentonite, caustic, lignite or lignosulfonate, CMC
9.	KCl, caustic, KOH, XCD polymer, poly anionic cellulose, starch, calcium carbonate, glycol.

The main parts of the wellbore and circulation of the drilling fluid is illustrated in Figure 1. The clay suspension carries the cuttings to the surface. Afterwards, the clay suspension is recirculated. During circulation of the fluid around the wellbore, strain rates may vary from zero to approximately 1000 s^{-1}. Also, temperature and pressure variations take place and may reach values 200°C and 1000 Bar. The viscosity of the fluid can be significantly affected by these conditions (Luckham and Rossi, 1999).

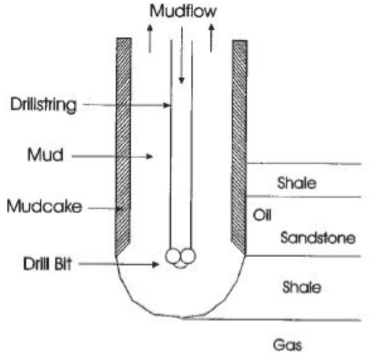

Figure 1. Schematic representation of the drilling fluid circulation through the wellbore (Luckham and Rossi, 1999).

2 RHEOLOGICAL BEHAVIOUR OF DRILLING FLUIDS

To optimize the drilling operation it is important to understand how the solid phase affect the bulk mud properties such as rheology, lubricity and wetting properties. The drilling fluid rheology is an important factor particularly affecting the hole cleaning (Robinson 2004). Furthermore, measuring the rheological properties is important to achieve hydraulic requirements and good carrying capacity for drill-cuttings (Mahto and Sharma, 2004). The typical rheological analyses include plastic viscosity, yield value, gel strength, and viscosity at low shear rate (Quintero 2002). In the following section properties which has a special importance in rheological characterization of drilling fluids are described.

2.1 Viscosity

Viscosity is resistance of a fluid which is deformed by a stress. For a uniform flow, the shear stress, τ=F/A, between two parallel layers is proportional to the velocity gradient ($\partial u / \partial y$), in the direction perpendicular to the layers (Slawomirski, 1975). Figure 2 shows the schematic representation the flow of a fluid between two plates one of which is moving relative to the other (known as couette flow). The relation between the shear stress and the velocity gradient can be given by the following equation (Larson 1999; Barnes et al. 1993; Luckham and Rossi 1999) :

$$\tau = \mu \frac{\partial \mathbf{u}}{\partial \mathbf{y}} = \mu . \gamma$$

Here the constant known as the viscosity coefficient or simply viscosity. Shear controls most of the viscosity related aspects of drilling operation. Because of that, viscosity of drilling fluids is the property that is most commonly measured. According to the Falode et al. viscosity of a mud is a function of three components: plastic viscosity, yield point and gel strength. Plastic viscosity is that part of the resistance to flow in mud caused by the friction between suspended particles and the viscosity of the base liquid (Falode et al., 2008). Yield point and gel strength will be explained in the flowing sections.

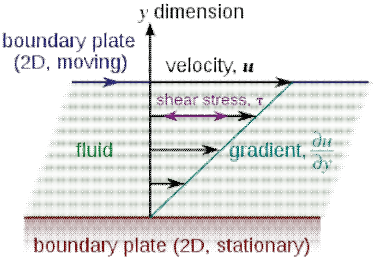

Figure 2. Schematic represantation of fluid flow between two plates.

Drilling fluids generally show non-Newtonian character, which means that viscosity is dependent on shear rate. When polymers are used as additives, drilling fluids normally exhibit a shear-thinning behavior (Perez et al.2004). Retention of drilling fluids on cuttings is primarly a function of the viscosity and the wetting characteristics of the mud. Drilling fluids having high viscosity at high shear rates tend to exhibit greater retention of the mud on cuttings (Robinson, 2004).

Temperature and pressure affect the viscosity of all external emulsion muds, and that temperature is the dominating variable in case of water-base muds (Singh et al. 1992; Panfil, 1987). Plastic and apparent viscosity decreases with increasing temperature. Temperature and pressure dependency of the drilling muds has a great importance especially in drilling operations in geothermal areas. In that case sepiolite does not exhibit an anomalous viscosity enhancement common to bentonite-based muds (Panfil, 1987). Annis studied the effect of temperature on Bentonite suspensions and found that the flow curves became more non-Newtonian and shear-thinning as the temperature increased (up to 150°C), displaying higher yield stresses and lower plastic viscosities (Annis, 1967).

The pH of the drilling mud should be kept between pH 8-9 for minimum viscosity. The pH of these systems can be adjusted generally using potassium or sodium hydroxide (Hamed and Belhadri, 2009). For the oil well drilling the dispersions have normally alkaline pH ranging between 9 and 12 (Kelessidis et al., 2007)

2.1.1 Rheological models

Various models are used to analyze the rheogical behaviour of drilling fluids. Bingham, Power law and Herschel-Bulkley are the commonly utilized models for this purpose. Figure 3 shows the types of rheological behaviour of particulate suspensions.

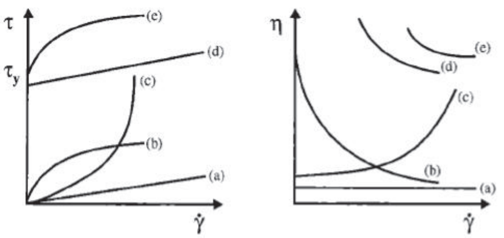

Figure 3. Types of rheological behaviour by colloidal dispersions: (a) Newtonian (b) Shear thinning (c) Shear Thickening (d) Bingham Plastic (e) Pseudo plastic with yield stress (Lewis, 2000).

Pseudoplastic (shear thinning) and dilatant (shear thickening) liquids are Stokesian fluids the constitutive equations of which are not linear (Slawomirski, 1975). Bingham plastic model has a yield stress value at zero shear rate. The equation for Bingham model is given as:

$$\tau = \tau_0 + \mu\gamma$$

However, for the case of high shear rates this model may overestimate the low shear viscosity of drilling fluids. In this case, the power law model can be utilized to fit the experimental data. Very dilute clay suspensions or drilling fluids that contain polymers behave as pseudoplastic fluids, which may be described by the power-law equation (Luckham and Rossi, 1999). The model equation can be described by the following equation (Robinson 2004; Larson 1999):

$$\tau = \mathbf{K}.\gamma^{\mathbf{n}}$$

Where K is the consistency index and n is an exponent that indicates deviation from the Newtonian behavior. When n<1 suspension shows shear thinning flow and when n>1 flow described as shear thickening or dilatant (Macosko, 1994). However, the power law model sometimes may underestimates the low shear viscosity. For the hole cleaning purposes in a vertical hole laminar flow with a low n value and high K value (in Power law model) produce a flat viscosity profile and carry cuttings out of the hole (Robinson, 2004).

The power law model, although useful as a first correction to Newtonian behavior, it may lead to substantial errors if the fluid exhibits yield stress. Kelessidis et al. showed that drilling fluids containing bentonite and bentonite–lignite as additives exhibit non-Newtonian behavior which can be described well by the Herschel–Bulkley model. The equation of this model can be written as (Kelessidis et al. 2006):

$$\tau = \tau_0 + \mathbf{K}.\gamma^{\mathbf{n}}$$

For clay based drilling fluids Herschel–Bulkley model works better than the Bingham model. Other models including Carreau or Krieger-Dougherty describe the structured particle suspensions well (Robinson, 2004). Talmon and Huisman claimed that Herschel–Bulkley model is in much stronger agreement with the true rheology especially at low shear rates. This is particularly important to horizontal directional drilling where the flow is laminar and has a low shear rate (Talmon and Huisman, 2005). Similarly, Hamed and Belhadri showed that water based mud including clay, polysaccharide xanthan gum and scleroglucan present a shear thinning behaviour with a low yield stress; it was shown that the Herschel–Bulkley rheological model described all formulations (Hamed and Belhadri, 2009).

Perez et al. used Cross–Williamson model to fit experimental data obtained from water based mud containing sand and polymer. The model equation is shown below.

$$\frac{\mu}{\mu_0} = \frac{1}{1+(\mathbf{a}\gamma)^{1-\mathbf{b}}}$$

Where μ_0 is the zero shear viscosity, a and b are the model parameters (Perez et al.2004). Other two parameter models like the Casson has been used to model the rheological behaviour of drilling muds in previous studies (Kelessidis and Maglione, 2008) although it is not widely utilized as other models. Model equation for the Casson is given as (Slawomirski 1975):

$$\sqrt{\tau} = \sqrt{\tau_C} + \sqrt{\mu_C \gamma}$$

Where μ_C and τ_C Casson viscosity and yield stress.

Duman and Tunc investigated the rheological behaviuor of Na-bentonite suspensions at low solid concentration (0.5 wt%) in the presence of various electrolytes. Shear stress data obtained for the 1 and 100 s^{-1} shear rates interval were treated according to the Bingham, Casson, Herschel-Bulkley models. It was found that the shear stress and shear rate values fitted well the Bingham model (Duman and Tunc, 2009).

Al-Zahrani developed a rheological model for shear thinning fluids such as drilling muds. This model was found to correlate the shear stress and shear rate for a variety of drilling fluids better than power law and the Herschel-Bulkley models. Besides, it can describe Newtonian and non-Newtonian fluids with or without yield stress at low and high shear rates (Al-Zahrani, 1997).

2.2 Thixotropy

Thixotropy means a reversible change from a high viscosity gel state at rest to a lower viscosity sol state by the application of shear stress. During this process, the microstructure of the material is reversibly destroyed (Barnes et al.1993). Thixotropy is a property based on viscosity, it implies a time-dependent decrease of the viscosity induced by flow and the effect is reversible when the flow is decreased (Mewis and Wagner, 2009).

Thixotropic behavior of muds is a convenient property in drilling process. When drilling mud is at rest, it is required to act as a suspending solid for cuttings, while moving drilling mud is required to behave as a viscous fluid capable of transporting this detritus (Dolz et al.2007).

There are different procedures used to evaluate thixotropy. For example, from the flow curves using hysteresis cycles thixotropic behaviour can be analyzed (Mewis and Wagner 2009; Dolz et al., 2000) by determining the areas enclosed between a upcurve rheogram for increasing shear rates, and down-curve rheogram for decreasing shear rates, after different stirring times. These areas are referred to as thixotropic areas (Mewis and Wagner, 2009; Macosko 1994). It is accepted that increasing thixotropic area is associated to increased thixotropy. Figure 4 shows different type up and down curves (in shear stress versus shear rate graph) that is used to determine the thixotropy.

In drilling, concentrated particulate suspensions such as bentonite mud in the presence of some organic compounds are often described as thixotropic, shear-thinning fluids with a yield stress (Taylor and Nasr-El-Din, 1998).

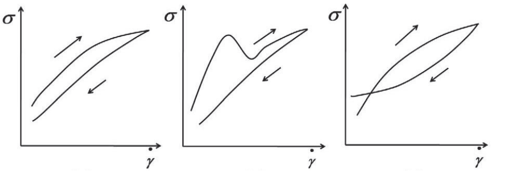

Figure 4. Different types hysteresis cycles which indicates the thixotropic behavior

obtained in shear stress vs shear rate graph (Mewis and Wagner, 2009).

2.3 Yield Stress

The flow behavior is described as plastic when the viscosity decreases with increasing shear rate after an initial threshold stress called the yield stress. It is the stress above which the material flows like a viscous fluid (Barnes et al.1993; Larson, 1999). Similar to yield stress the gel strength is a measure of the same inter-particle forces of the mud as determined by the yield point, except that gel strength is measured with the mud at rest (Falode et al., 2008).

Drilling process relies on the existence of a yield stress of the drilling fluids for suspending high gravity solids (such as barite) and for transferring drill cuttings to surface. Aqueous bentonite dispersions at concentrations more than 1% exhibit a yield stress (Kelessidis and Maglione, 2008).

The high yield point/viscosity ratio indicates a shear thinning mud which is desirable for drilling as it sets to a gel, which is sufficient to suspend the cuttings when circulation is stopped and which breaks up quickly to a thin fluid when it is agitated (Mahto and Sharma, 2004).

The rheological parameters such as apparent viscosity (μ_a), plastic viscosity (μ_p) and yield point (τ_0) was determined by Zhang and Li, using following equations:

$\mu_a = \mu(600)/ 2=$ (mPa. s)

$\mu_p = \mu(600)- \mu(300)=$ (mPa. s)

$\tau_0 =0.511(\mu(300)- \mu_p=$ (Pa)

where $\mu(600)$ is the viscosity at the rotating rate of 600 rpm and $\mu(300)$ is the viscosity at the rotating rate of 300 rpm (Zhang and Li, 2001).

2.4 Viscoelasticity

Viscoelasticity is the property of materials that exhibit both viscous and elastic characteristics. Unlike purely elastic substances, a viscoelastic substance has an elastic component and a viscous component. Viscoelastic substance loses energy when a load is applied, and then removed. Hysteresis is observed in the stress-strain curve, and the area of the loop is equal to the energy lost during the loading cycle (Larson 1999; Barnes et al. 1993; Macosko, 1994). Figure 5 shows the difference between the stress-strain curves of pure elastic and viscoleastic materials.

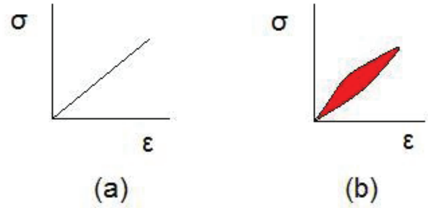

Figure 5. Shear stress vs strain curves for a) Elastic, b) Viscoelastic material.

The Maxwell and the Kelvin-Voight model are most well known models which is developed to describe the viscoelastic behavior of materials. According to the Maxwell model (elastic and viscous components connected in series) if the material is put under a constant strain the stress gradually relax. If it is under constant stress the strain has an elastic component (spring) and the viscous component (dash-pot) that grows with time (see Figure 6). On the other hand, Kelvin Model (elastic and viscous components connected in parallel) is used to explain creep behavior (Larson 1999).

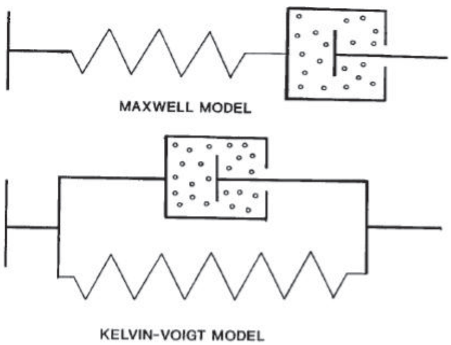

Figure 6. Schematic representation showing the Maxwell and Kelvin-Voight models.

Viscoelastic behavior can be characterized by the dynamic rheological measurements (or oscillatory techniques). Dynamic mechanical testing allows the sample to be subjected to a strain and the viscous and elastic properties of the sample to be measured simultaneously. During oscillation measurements, a frequency dependent shear stress or strain is applied to a suspension, and the shear moduli is obtained (Peker and Helvacı, 2003).

The complex modulus, G^*, (the total energy required to deform the material) includes the complete information on the viscoelastic behavior.

$$G^* = G' + iG''$$

Where G′ is the storage or elastic modulus, G″ is the loss or viscous modulus. The storage or elastic modulus is a measure of the energy stored and recovered in the material, on the other hand viscous or loss modulus is a measure of the energy lost as heat in the material through flow or deformation (Larson 1999; Barnes et al. 1993; Peker and Helvacı, 2003). Absolute magnitude of the complex modulus can be written as the ratio of the maximum stress to the maximum strain (Barnes et al. 1993, Macosko 1994):

$$\left|G^*\right| = \frac{\tau_0}{\gamma_0}$$

From this value elastic and loss modulus values can be calculated using trigonometric relations:

$$G' = \left|G^*\right| \cos\delta$$
$$G'' = \left|G^*\right| \sin\delta$$

Similarly tangent loss, tan δ, is an important parameter which is a measure of the energy lost to the energy stored.

$$\tan\delta = \frac{G''}{G'}$$

Some drilling fluids can exhibit elasticity as well as viscosity. These type of viscoelastic fluids posses some solid like properties especially at low shear rates. Shear thinning drilling fluids such as xantham-gum based muds tend to be viscoelastic and may lower the efficiency of static seperation tanks and centrifuges (Robinson, 2004). On the other hand, viscoelastic drilling mud systems can be used to avoid the fluid loss.

3 EFFECTS OF ORGANIC/INORGANIC ADDITIVES ON THE RHEOLOGICAL BEHAVIOUR

Different types of organic and inorganic additives are used in drilling operation. During the initial stages of drilling (Zone I), a drilling fluid with cuttings-carrying ability is needed to clean and clear the hole. This fluid is characterized by a high yield point to plastic viscosity ratio. The mud of the first zone consists of water, clays, and a viscosifying agent that is an additive used to increase the yield point of the mud. In the second zone (Zone II), additives are included in the drilling fluid to control its plastic viscosity and reduce its yield point. These additives are known as thinning agents. For the last zone (Zone III), additives are used to reduce further the yield point of the mud and increase its thermal stability (Dairanieh and Lahalih, 1988). In the following paragraphs types of these additives and their effect on the flow behavior of drilling fluids are described.

3.1 Polymers

Organic polymers are commonly used to control the rheology of drilling fluids. For adaquate carrying capacity of cuttings in horizontal holes a critical concentration of polymer must be present (Singh et al.1992; Caen and Chillingar, 1996).

The polyanionic cellulose (PAC), guar, xanthan and tamarind gum, starch, carboxymethyl cellulose (CMC), hydroxyethyl cellulose are the examples of the polymeric additives used for this purpose. Starch is used in salt water system as a fluid loss agent for all types of mud systems (Dairanieh and Lahalih, 1989). On the other hand, CMC is a linear polysaccharide polymer based on a cellulose backbone. NaCMC is used in drilling as a thickener; for stabilizing suspensions and; for the formation of gels. At high concentrations, CMC solutions show viscoelastic and thixotropic behavior (Dolz et al., 2007).

PAC is used as a fluid loss reducer for fresh water and salt-water muds. It also acts as viscosity modifier (Xie and Lecourter, 1991). The use of biopolymers such as guar gum (Perez et al.2004; Guo et al.2006) as additive in drilling fluids is widespread due to their low cost. Improvements of transport and suspension capacity by biopolymers have been correlated with their high viscosity in aqueous solutions at low shear rates and with their shear-thinning behavior. Similarly, xanthan gum is a high molecular weight polysaccharide that can give interesting properties to the drilling fluids (Xie and Lecourtier, 1991). It is used as a rheology control agent in aqueous systems and as a stabilizer for emulsions and suspensions. Hamed and Belhadri showed that scleroglucan is more effective in the presence of salt than the xanthan to modify the rheological properties in water based muds. (Hamed and Belhadri, 2009).

In water-based drilling fluids, polyethylene glycols (PEG) have demonstrated to be effective shale inhibitors, that is, they prevent clay cutting from dispersing in the medium and reducing wellbore problems, increasing drilling rates (Luckham and Rossi, 1999).

Guo et al. investigated the rheology of water based bentonite muds in the presence of PAC, potassium chloride (KCl) and sodium silicate. They showed specifically formulated KCl/silicate drilling fluids exhibit ideal rheological properties, perfect filtration control capacity and good inhibitive character. It was found that the favorable values of plastic viscosity (19-28 mPa.s), yield point (9-15 Pa), gel strength (5-9 Pa) and filtration rate could be obtained when the ratio of KCl, silicate and bentonite was kept within the range of (5~ 8) : (7~ 11) : (1~ 3) during drilling operations (Guo et al. 2006).

Mahto and Sharma showed that the apparent viscosity, plastic viscosity, yield point, yield point/plastic viscosity and gel strengths of the mixture containing 0.1% PAC and 3% bentonite increases with increase in the concentration of tamarind gum. Combinations of tamarind gum, PAC and bentonite clay produce favorable rheological properties and optimum fluid loss at very low concentrations (Mahto and Sharma, 2004).

Similarly, Yun-kui et al. investigated the rheology of drilling fluid mainly composed of heteropolysaccharides (PG gum). Results indicate that PG gum based drilling fluids behaves non-Newtonian shear thinning fluids. With the increase of PG gum concentration from 0.5 to 2.5% the consistency coefficient also increases and the viscosity index decreases (Yun-kui et al., 2007).

3.2 Surfactants

Surfactants are surface active agents having a hydrophilic head that carrying a positive or negative charge and hydrophobic tail containing hydrocarbons (Myers, 1991). Surfactants are utilized in drilling muds to suspend and transport the cuttings, to control the loss of fluids and to reduce corrosion etc. They act as dispersing and wetting agent, thinner, emulsifier, and lubricant (Fabri et al.1970; Hamed and Belhadri, 2009).

Lignosulfates are the most widely used thinners that may help disperse the clay components of the mud. Soaps of sulfurized vegetable oils are also widely used surfactants to increase the lubricity of water based muds (Nelson, 1982). Similarly, long chain alkylpolyglucosides may be utilized to form water based muds. These type of muds have high yield stress value which is useful in preventing settling of drilling cuttings (Balzer and Luders, 2000). Additionally, alkylolamine based esterquats hydrophobize clays for their use as viscosity promoters in drilling fluids (Holmberg, 1998).

Sodium dodecyl sulfate (SDS) is an anionic surfactant (Holmberg, 1998) used in the formulation of drilling fluids. Figure 7 shows the chemical structure, surface tension and the critical micelle concentration of SDS solutions.

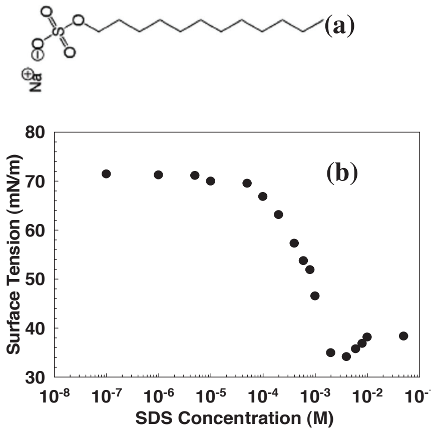

Figure 7. a) Chemical structure of SDS, b) surface tension of SDS solutions as a function of concentration (Deliormanli 2009, in press).

Surfactants are also used as shale inhibitors. The inhibitive property of polyoxyalkyleneamine (POAM) to sodium montmorillonite (Na-MMT) was investigated by Qu et al., and the shale cuttings recovery ratio and the rheological properties of drilling fluids were measured before and after adding POAM in several water-based drilling fluids. The results showed that POAM was completely water-soluble, exhibited the superior performance to inhibit the hydration of Na-MMT and reduced the swelling or hydration of shale cuttings effectively (Qu et al., 2009).

To solve the problem of bit balling some surfactants have been used as additives in water based drilling fluids. Alkyl aryl sulfonate, nonionic surfactants such as alkyl aryl ethoxylates and alcohol ethoxylates are the examples of surfactants utilized for this purpose. These surfactants may adsorp onto the drill bit by dispersion forces, dipole interactions, and electrostatic forces. On the other hand, adsorption of surfactant onto the drill cuttings such as clay occur by the hydrophilic part of the surfactants and with the hydrophobic part of the surfactant molecules oriented toward the aqueous phase. By this mechanism the clay surface will become more hydrophobic and less easily wetted by water phase. Figure 8 shows the photographs of a drill bit used in drilling operations with water-based fluid, where the bit balling problem is observed and the same type of bit used in drilling operations with a water-based mud formulated contains a surfactant (Quintero, 2002).

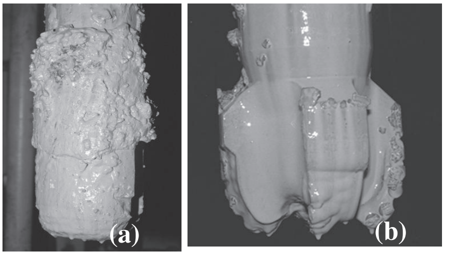

Figure 8. Drilling bit with water based mud a) in the absence of surfactant b) in the presence of surfactant (Quintero, 2002).

3.3 Inorganics

Several types of clays can be used as additives to prepare water based drilling muds. Mixed metal hydroxides and metal silicates are charged materials and electrostatically interact with bentonite/clay forming a gel structure. This gel structure allows the fluid to move as a solid like phase when pumped carrying all solids along with it (Caen and Chillingar, 1996). Mix metal inorganics also have excellent shale stability ability and formation protection characteristics.

High filtration loss problem of sepiolite mud was investigated by Serpen. Results showed that sepiolite along with some minerals such as loughlinite, borax, talc and nontronite could be utilized in making drilling mud at high temperature for geothermal use. According to the results of this study sepiolite-nontronite mixture has the best apparent viscosity. Also, cutting transport capacity indicated by flow behavior index is also favorable for this mixture. On the other hand, results of experimental work carried out on sepiolite-talc and sepiolite-borax combinations were not encouraging from the viewpoint of both rheology and filtration properties (Serpen, 2000). Previous study of the same author indicates that leonardite and alumina can be used for similar purposes.

On the other hand, some inorganic electrolytes such as potassium chloride is used in drilling fluids to aid in the stabilization of shales and to control swelling clays. Inhibition is obtained with KCl in two ways. The chloride ion (Cl^-) prevents water from entering the clay matrix. The KCl, sodium chloride (NaCl) or calcium chloride ($CaCl_2$) can be used for this purpose. Potassium works as an inhibitor by exchanging with the Na^+ or Ca^{+2} found in native clays. The potassium ion is small and fitting ideally into the space where water can be drawn into the clay, and reduce swelling (Hamed and Belhadri, 2009).

The effect of pH and of electrolyte concentration on the rheological properties of bentonite dispersions has been studied by Kelessidis and co-workers. The rheograms have been fitted very well by the Herschel–Bulkley model. For the range of pH values studied, between 7.7 and 10.5, a maximum on the yield stress, the flow consistency index and the apparent viscosity at all shear rates has been observed at approximately the natural pH 8.7, while at the highest pH (10), the flow consistency index increased drastically. Addition of salt (NaCl) decreased the yield stress, the flow consistency index and the apparent viscosity at all shear rates for all three studied bentonite concentrations of 2%, 5% and 6.42%, over the range of electrolyte concentrations from 0.0 M to 1.0 M. The flow behavior index increased up to 0.1 M and then decreases. For the lower than 0.5 M salt concentrations, the decrease of the rheological parameters, was attributed to the compression of the electric double layer (Kelessidis et al.,2007)

4 CONCLUSIONS

Different types of water based muds are utilized in horizontal drilling. Rheological behavior of these muds greatly influences the efficiency of the drilling operation. In this review, rheological properties that are important for all aspects of the drilling were described. Effects of additives on the rheological behavior were discussed. Following are the general conclusions:

-In general water based muds show shear thinning behavior, has a yield stress and thixotropic character.

-Herschel-Bulkley model describe well the rheological behavior of these types of drilling fluids.

- Drilling process relies on the existence of a yield stress of the drilling fluids for suspending high gravity solids and for transferring drill cuttings to surface.

- Inorganic electrolytes, pH of the overall system, temperature and pressure greatly affect the viscosity, and the rheology of the drilling muds.

- Polymers, surfactants, polymer/surfactant mixtures and inorganic solids have some important influence on the flow properties of water based muds.

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Genleşmiş Kil Agregası Üretiminde Katkı Kullanımının Önemi

Importance of Additives Use in Production of Expanded Clay Aggregates

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ÖZET Genleşmiş kil agregası üretiminde kullanılan killer genellikle kendiliğinden genleşmediğinden dolayı genleştirmek ve agrega kalitesini artırmak için çeşitli katkılar kullanılmaktadır. Bu çalışmada genleşmiş kil agregası üretiminde kullanılan katkılar ile ilgili bilgiler verilmiştir. Ayrıca, yapılan genleştirme deneylerinde kömür katkısı kullanılarak genleşme sıcaklığında 50°C düşüş sağlanmıştır.

ABSTRACT Non-selfbloathing clays used in production of expanded clay aggregates, are expanded by using additives. At the same time, additives are also used to emerge aggregate quality. In this study, information are given about additives used in production of expanded clay aggregates. In the expanding experiments made, reductions about 50°C in the expansion temperatures were achieved by using coal as an additive.

1 GİRİŞ

Genleşen killer, ısı ile işlem gördüklerinde, gaz çıkışı ile birlikte hacimleri 5 - 6 kat artarak genleşebilen killerdir. Bu tür killer içerdikleri gazların çıkışı esnasında, dış yüzeylerinde sinter kabuk oluşmakta, çıkan gazlar bünyede hapsolmakta ve böylece, hafif, gözenekli, cürufumsu ve sert bir yapıya dönüşmektedirler. Genleşen kil agregalarından elde edilen blokların tipik özellikleri şunlardır:

1. Düşük yoğunluk,
2. Uygun yük taşıma kapasitesi,
3. Düşük su emme oranı,
4. Yüksek dona doyanım,
5. Uygun sıva tutma,
6. Isı izolasyonu,
7. Yüksek ateş direnci,
8. Ses izolasyon avantajı sağlaması,
9. Kimyasallara direnç sağlamasıdır.

Genleşen killer, teknolojik özellikleri ve birçok endüstriyel hammadde türüne göre değişik avantajlara sahip olması nedeniyle, giderek artan bir eğilimle farklı endüstri dallarında, yaygın kullanım alanlarına sahiptir. Genleşmiş kil agregalar özellikle inşaat sanayinde hafif yapı malzemesi yapımında kullanılmakta olup, hafif olması nedeniyle bina üzerindeki ölü yüklerinin azaltılmasında; basınç dayanımlarının yüksek oluşu nedeniyle köprü ve tünel inşaatlarında; gözenekli yapısından dolayı ise ısı ve ses yalıtımı sağladığı için enerji tasarrufunda önemlidir. En büyük avantajları, istenilen miktar ve boyutlarda, teknik açıdan çok değişik taleplere cevap verecek şekilde üretilebilmektedir.

Genleşen killer sahadan alındıktan sonra boyut küçültme (kırma – öğütme) işlemlerinden geçirilmektedir. Öğütülen kil, katkılar ve su ile karıştırılarak şekillendirilmekte ve ısıl işleme tabi tutularak genleştirilmektedir.

Killer, doğal olarak genleşme için yetersizdir. Bunları genleşebilir yapmak için uygun katkıların ve yardımcıların eklenmesi gerekmektedir. Böylece özel karakteristikte daha kolay ve güvenilir, genleşmiş kil üretilebilmektedir. Örneğin termal izolasyon karakteristiğini en üste çıkarmak için, genleştirici ilaveleri genleşen kile daha çok eklemek gerekir.

Çoğu araştırmacı, genleştiren gazların kaynağının kendiliğinden olduğu konusunda şüpheci davranmışlar ve genleştirme için bazı kaynakları önermişlerdir. Bunlar: hematit, pirit, grafit, ankerit, linyit sülfatlar, küller, kalsit ve/veya organik maddedir. Birincil genleştirici madde ise henüz tanımlanamamıştır.

Katkılar için örnekler Çizelge 1'de verilmiştir. Bu maddelerin güvenilirlikleri tecrübelerle sabittir (EIPPCB, 2005).

Katkı ve yardımcıların çeşidi; hammaddenin bileşimine, üretim yönteminin teknolojisine, ekonomik olarak sağlanarak kullanılabilirliğine ve emisyon karakteristiklerine bağlıdır (EIPPCB, 2005).

Çizelge 1. Genleştirmede kullanılan katkılar

Eriticiler	Genleştiriciler	Ayırıcılar
Demir oksitler	Ağır yağ	Kireç bileşikleri
Demir hidroksitler	Lignosülfatlar	Dolomit
İllitik killer	Çok ince kok / kömür	
	Bitümlü killer	

2 KATKILI GENLEŞTİRME DENEYLERİ

Kendiliğinden genleşme oranı çok düşük olan Kastamonu – Küre kiline kömür tozu ilave edilerek genleşme özellikleri tespit edilmeye çalışılmıştır.

Deneylerde kullanılan kömür katkısı Tunçbilek'ten temin edilmiştir. Bu kömürün seçilmesinin nedenleri, ülkemizde fazla miktarda linyit bulunması, kolay temin edilebilmesi ve maliyetinin düşük olmasıdır.

Genleşen kil üretiminde kullanılan katkıların oranları, ticari nedenlerle gizli tutulmaktadır. Bu nedenle ilk olarak katkı oranının genleşmeyi nasıl etkilediğine ve ne kadar olacağına yönelik çalışmalar yapılmıştır. Kullanılacak katkı oranının tespit edilmesine yönelik olarak 5 farklı oranda deneme yapılmıştır. Denenen katkı oranları; % 1, 5, 10, 20, 35 ve 50 dir. Bu katkı oranlarından en iyi genleşmenin sağlandığı oran kile eklenerek detaylı genleştirme deneyleri yapılmıştır.

100 µm altına öğütülen kile % 1 kömür katkısı eklenerek hazırlanan peletler 1000°C'den 1200°C'ye kadar 50°C artırılarak 5 farklı sıcaklıkta ısıl işleme tabi tutulmuştur. Ayrıca her bir sıcaklıkta 5, 10, 15 ve 20 dakika süre ile fırında tutulmuşlardır. Elde edilen birim hacim ağırlık ve genleşme oranları Şekil 1'de verilmektedir.

3 SONUÇ

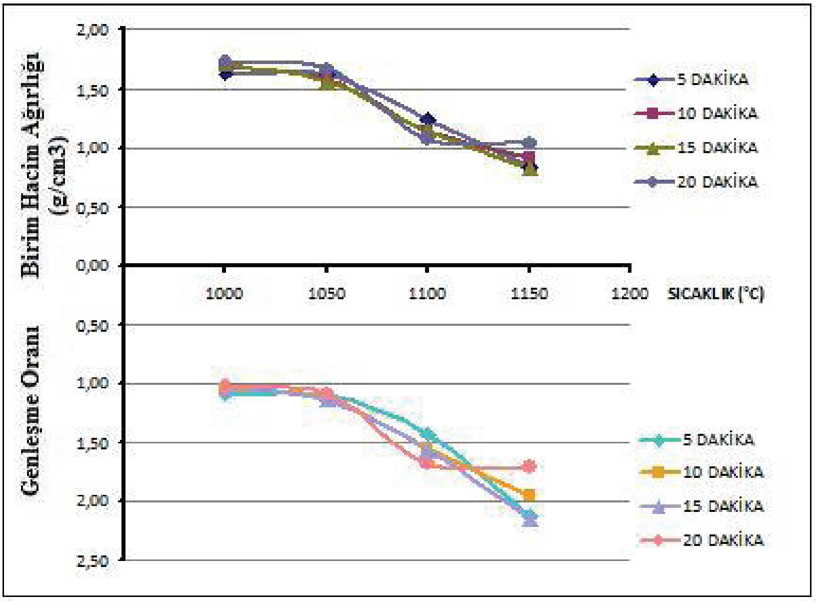
Günümüzde hafif agregaların, doğal agregalar ile başarılı bir şekilde rekabet edebilmesi için yeni üretim yöntemleri ve malzemeleri neyin genleştireceği üzerinde kapsamlı araştırmalar yapmak gerekmektedir.

Hazırlanan peletlerin bir kısmını katkılar oluşturduğu için, katkıların yanması ve gaz çıkışı ile fırından çıkan genleşen kil agregasının pelet çapları katkısız genleştirmelere oranla daha düşük olmaktadır.

Katkı miktarının çok fazla olması iki nedenle istenmemektedir. Birincisi ekonomik olmaması, ikincisi ise agrega fırından alındığında bile katkıların hala yanmaya devam etmesidir. Ayrıca fazla kömür katkısı, peletlemeyi zorlaştırmakta ve ilave katkılar ile birlikte kullanılmasını da zorunlu hale getirmektedir.

Kendi kendine çok iyi genleşemeyen Kastamonu - Küre kili, katkısız olarak 0,94 g/cm^3 birim hacim ağırlığında üretilebilen agregalar, katkı ilavesiyle bu birim ağırlık değerleri 50°C daha düşük sıcaklıklarda elde edilebilmektedir.

Kastamonu – Küre kili katkısız olarak 1200°C'de genleşmeye başlarken, katkılı olarak 1150°C'de genleşmeye başlamaktadır.



Şekil 1. Kömür katkısı ile elde edilen birim hacim ağırlık ve genleşme oranının sıcaklığa bağlı değişimi.

Kastamonu – Küre kili için kömür katkılı en yüksek genleşme 1200°C fırın sıcaklığında 5-10 dakika fırında kalma süresinde gerçekleşmektedir. 0,60 g/cm^3 birim hacim ağırlığında agregaların üretimi bu koşullarda mümkün olabilmektedir. Genleşme oranları ise katkısız 1,9'lardan 2,9 kata kadar ulaşabilmektedir.

Bu çalışma, uygun katkılar kullanıldığında, genleşmiş kil agregası üretimi için gerekli fırın sıcaklığında en az 50°C düşürülebileceği göstermiştir. Böylelikle endüstriyel boyutta agrega üretim maliyetleri oldukça düşürülebilecektir ki rekabet şansını daha da artırmasını sağlayacaktır.

Kömür katkısı ile birlikte yardımcı katkıların katılması ile üretim sıcaklığı ve agregaların birim hacim ağırlığı daha da düşürülebilecektir.

KAYNAKLAR

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Özgüven, A., 2009. Genleşen Kil Agrega Üretimi ve Endüstriyel Olarak Değerlendirilmesi, Doktora Tezi, Süleyman Demirel Üniversitesi, Fen Bilimleri Enstitüsü, 307 s., Isparta.

Stability Analysis of Excavations in Jointed Rocks – The Computer Program RESOBLOK

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ABSTRACT The paper presents the computer program RESOBLOK able to represent a fractured rock mass in the form of an assembly of blocks.

The fractures are represented by plans, (infinite or limited by other fractures) in a statistical or deterministic way or like polygons. RESOBLOK has a downstream module **bsa** allowing the stability analysis of blocks by limit equilibrium or by minimization of energy. The potential movements considered are the free fall, the slip (according to one or more plans) and the rotation. The blocks are supposed to be rigid. Parameters necessary for the analysis are the density of blocks, cohesion and the angle of friction of the joints. Computation is iterative: in a first phase the stability of the blocks in the vicinity of a free face is examined and those whose factor of safety is lower than a fixed value (generally 1) are considered unstable and removed, which modifies the free face of the model. In the following iterations, the stability of the blocks at the boundary of the new free face is studied until there are no more instable blocks. The geometrical movements and the computation of the resultant of forces are based on a vectorial method developed by Warburton. A bolting pattern, specified by the spacing, the length and the resistance of the bolts can be taken into account in the analysis of stability.

The code was used for the representation and dimensioning of the underground excavations (tunnel, mines) or in the road slopes, open pit mines, natural slopes, etc.

1 INTRODUCTION

The stability analysis of mining workings must generally take in account the role of discontinuities. The fractures crossing the rock mass, because of their reciprocal intersection, determine blocks of different size and shape. Among these blocks, those that are situated next to the surface of the excavation risk to move toward the excavated area. The instability appears thus as fall or slip of blocks (Figure 1). Its evolution leads, either to obtain a new steady shape of the excavation, either the total loss of the stability while returning the excavation unusable.

The progress made the last twenty years has led to better represent the geometry of the discontinuities in the rock mass as well as to improve the representation of the behaviour of the discontinuities and of the rock mass. The geometrical representation of the rock mass has an important influence on the subsequent stability computation.

Many computer codes are developed in order to study the fractured rock mass and are able to represent their geometry and their mechanical, hydraulical or thermical behaviour as well as coupled actions.

Among these, the code RESOBLOK, that knew its beginnings toward the end of the years '80 (Heliot, 1988), is developed in the goal of creation of databases representing the rock mass.

Figure 1. Examples of unstable blocks on the sidewall of an underground excavation.

Modules able to make a quick stability analysis (Baroudi et al. 1990), to display these databases and the histograms of the volume of blocks (and of the unstable blocks) or of the area of the surface have been added to the first version. The quite flexible structure of RESOBLOK has able to include in its data base a lot of the information usable by other modelling software either directly or indirectly through appropriate interfaces. Although the different 3D discontinuous computer codes of the rock mechanics have developed their own data base, due to the richness of joint generation method and integrated elements RESOBLOK is useful for numerous practices.

2 PRESENTATION OF COMPUTER CODE RESOBLOK

RESOBLOK is an integrated modelling tool taking into account a fractured rock mass represented as blocks assembly and included joints. It architecture is organized in modules as describes below:

- at first, a geometrical module able to represent the rock mass as blocks separated by joints "from geological evidence" (Heliot, 1988) and including joints of finite extent;
- next, a set of downstream modules allowing to study the blocks assemblage (Figure 2).

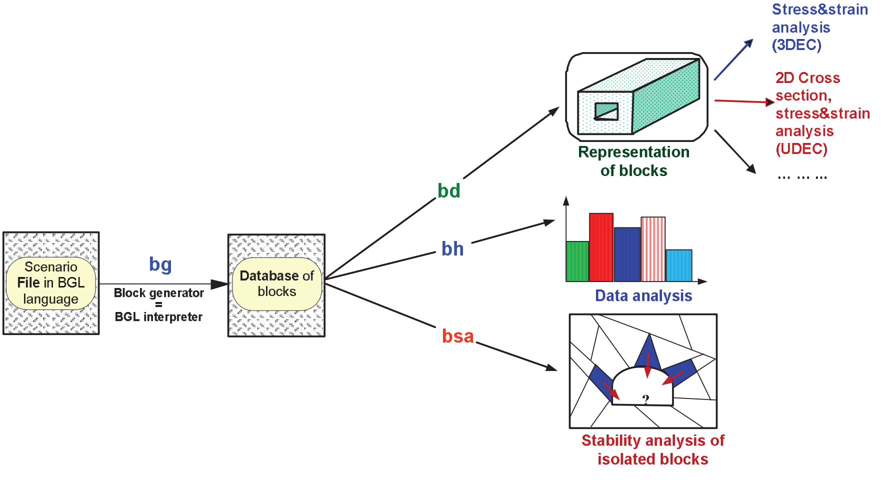

Figure 2. The modular organization of RESOBLOK.

2.1 Creation of Blocks Database

The geometrical representation is computed by a blocks generator named "**bg**". This generator needs as input a "scenario file" which gathers the information representing the studied rock mass and supply an output file describing the blocks assemblage generated. The output file is the so-called "data base" and depict the geometry of the blocks, the joints and the mechanical properties associated with both the joints and the rock mass.

The studied zone called "zone of interest" defined as a rectangular parallelepiped is subdivided in blocks by the progressive insertion of discontinuities (fault, strata…) defined deterministically or stochastically. In the last case a set of fractures is defined by a first statistical law characteristic of the orientation and a second one defining the spacing. The fractures persistence is taken into account by stopping their extension on another set of fractures or by defining the fracture as polygonal (Bennani, 1990, Thoraval et al, 2005)

In order to help the data input, and to manage the data error a processing specific language named BGL (Block Generation Language) has been developed (Heliot, 1988). BGL allow writing a scenario-file which will be interpreted by the "**bg**" command for generating the blocks assemblage as a database file. When the discontinuities are stochastically defined, the same scenario file allows generating, via the module "**bg**", different blocks assemblies distinguished by a so called "simulation number".

The exploitation of the database thus created is done by different modules (Figure 2). Among these, we can distinguish the module "**bd**" (Block Display), the module "**bh**" (Block Histogram) and the module "**bsa**" (Block Stability Analysis).

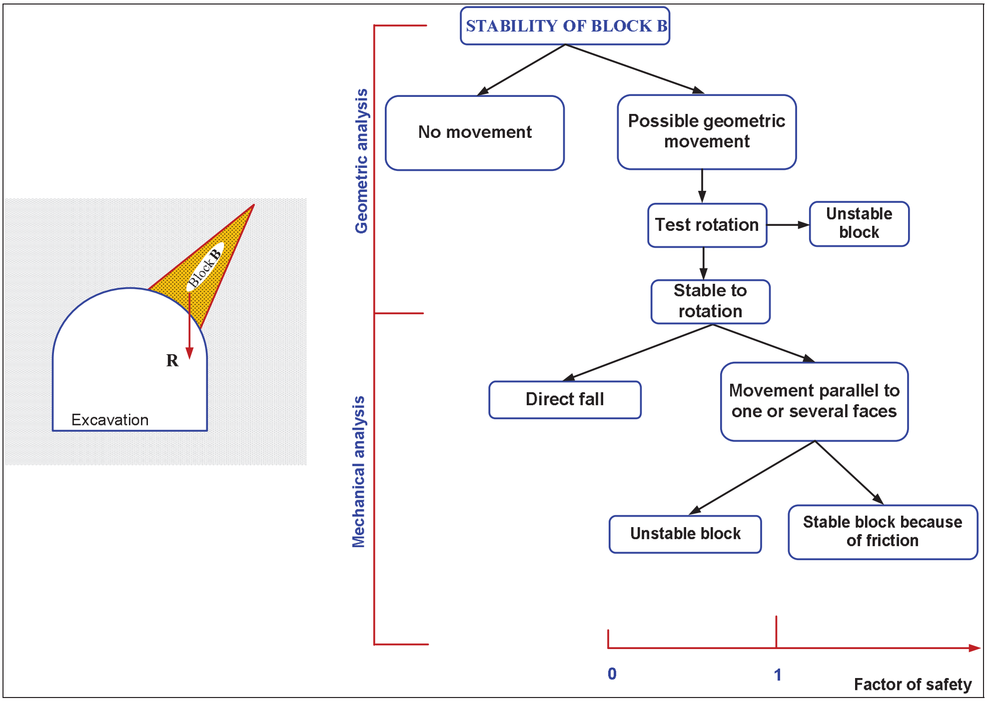

Figure 3. Block stability analysis.

2.2 Stability Analysis Module (Bsa)

The module "**bsa**" has been realized in order to perform a stability analysis of the excavations realized in blocky rock masses. It allows analysing the stability of blocks in contact with an excavation by simple computations based on limit equilibrium (Asof, 1991). **bsa** checks if isolated blocks next to an underground or an open pit excavation are able to detach. If possible, the boundaries of excavation change and the computation can goes one iteratively, until no more blocks are unstable. The stability analysis is based on Warburton's algorithm (Warburton, 1981) and followed two stages (see Figure 3):

- the geometric analysis permit to exclude of further analysis the blocks that are geometrically unremovable,
- the mechanical analysis look at the possible movement of geometrically removable blocks and if possible compute the safety factor of the block taking into account the geotechnical properties of discontinuities and the density of blocks that are introduced in the scenario file. The potential movements examined are successively:
  - Direct fall. If, due to gravity, direct fall is possible the block is declared unstable, but no safety factor can be computed (except in the case of support) because the joint is supposed to have no tension resistance,
  - Movement parallel to one or several faces. In this case a factor of safety can be computed. If the movement is parallel to one or two faces limit equilibrium analysis is possible and the only parameters needed are the cohesion and friction angle of discontinuities that border the block as well as the density of the block. If the movement is parallel to more than two faces, limit equilibrium analysis is not possible but an energy based analysis can be used. The energy based analysis required additional parameters as normal and shear stiffness of the discontinuities.

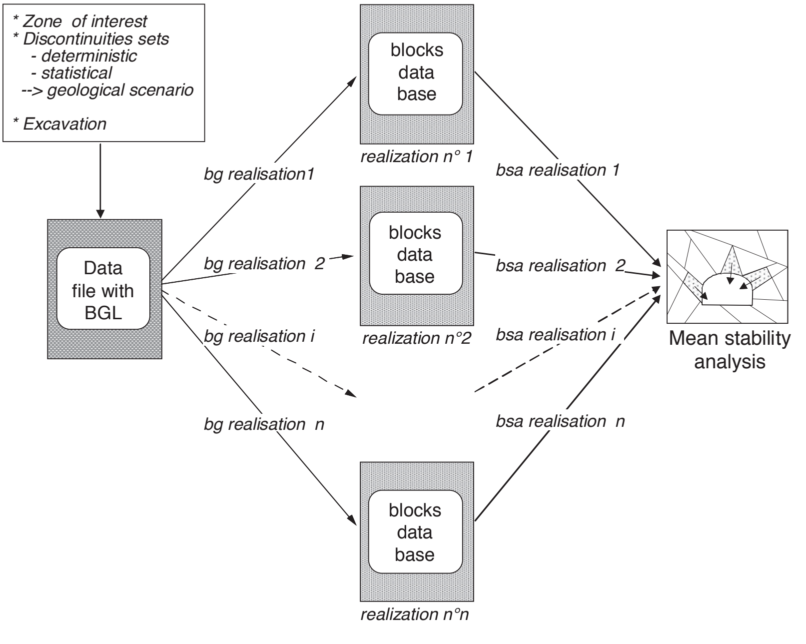

Figure 4. Stochastic aspects of RESOBLOK and **bsa.**

A **bsa** run need 3 stages: the data input, the computation and the output analysis. In the case fracture are stochastically defined, for a unique scenario file there are several realisations of the blocks assembly (identified by different simulation number). **bsa** module realizes a stability analysis for each blocks assemblage and the results can be processed statistically: number and volume of unstable block, min, max volume… (see Figure 4).

One of the advantage of RESOBLOK software lies on the fact that the result of the stability analysis can be analysed statistically. Baroudi et al. (1990 and 1992) have shown on a case of slope stability problem that a minimum of 50 geometrical simulations was to be performed in order to achieve a reasonable outcome in the stability analysis. Running 50 simulations, in this case allows getting stable values of the mean, the standard deviation and the histogram of variables such as the mean volume of unstable blocks, the total unstable volume and the number of unstable blocks. This result is also confirmed by later studies (Merrien-Soukatchoff et al., 2007).

2.3 Stability Calculation of Blocks Reinforced By Bolting

If the number of unstable blocks is important, in order to ensure the stability of the working, one possible remediation can be the use of bolts or cables to reinforce the ground. Its role is taken into account in BSA by the fact its resistance prevents the direct fall or the sliding of isolated block and may avoid the propagation of instability by stabilizing the key blocks.

2.3.1 Calculation of factor of safety of blocks reinforced by bolting

Two types of behaviours can be taken into account in the analysis: the active reinforcement or the passive reinforcement. The first one rather corresponds to prestressed bolts and the second one to fully grouted anchor (Seegmiller, 1982).

In both cases the effect of bolts is taken into account (Korini et al., 1993) by adding in the forces balance a force which module is the strength of the bolt and acting in the direction of the bolt. The force induced by the bolt is resolved in:

- A force parallel to the movement (opposite direction);
- A force perpendicular to the movement direction.

In the case of direct fall of the block only the first component is taken into account (suspension role). In the case of sliding along one or more faces the force parallel to the movement direction is added to the shearing of the discontinuity and the force perpendicular to the movement direction increase the normal stress, so the limit shear strength. For active reinforcement the force resolved parallel to the movement is numbered among the driving forces whereas for passive behaviour it is numbered among the resisting forces. Depending on the case (active or passive reinforcement) the factor of safety is calculated as:

a) active reinforcement

$$f_{sa} = \frac{F_c + F_{nw} \tan\varphi + F_{nb} \tan\varphi}{F_{tw} - F_{tb}}$$

b) passive reinforcement

$$f_{sp} = \frac{F_c + F_{nw} \tan\varphi + F_{nb} \tan\varphi + F_{tb}}{F_{tw}}$$

with: f_{sa} and f_{sp} – factor of safety (respectively for active and passive reinforcement);

F_c – cohesion force on the sliding face;

F_{nw} – normal component of active forces;

F_{nb} – normal component of the force induced by bolts;

F_{tw} – tangent component of active forces;

F_{tb} – tangent component of the force induced by bolts.

For all possible cases of instability (direct fall**,** sliding along a single face, wedge sliding, etc.) the formulation of factor of safety calculation is performed (Korini et al., 1993)

2.3.2 Setting up a bolting pattern

Generally, bolting is applied according to a regular pattern and the problem arising for an excavation is to find a bolting pattern to prevent the block fall.

Given that the distribution of fractures it's known, generally, only statistically, we obtain, by simulations, various block models ("**bg**" command) and for every model are performed stability calculations with various bolting patterns.

The bolting pattern is applied over a given surface, which corresponds to one or multiple faces defining the excavation. It can be the roof or the roof and the lateral faces of the underground room or gallery, or the slope of an open pit.

The necessary parameters to define a bolting pattern ensure to obtain various patterns that we can be meet during the practice of exploitations.

2.3.3 Optimization of the bolting pattern

The stability analysis consists of the determination of factor of safety of potentially instable blocks that eventually we reinforce with bolting.

For every simulation and for every bolting pattern, the results are presented as number of unstable blocks, volume and weight of every unstable block and the total weight of unstable blocks. It is possible also to furnish the results by number (or volume) of blocks corresponding to various types of instability (free fall, plan sliding,... etc).

Finally, for the set of simulations it is possible to calculate the average volume (and/or the average weight) of instable block for every bolting pattern. So, it is possible to choose as the best pattern the one that minimize the block instability. That choice can be confirmed by the use of histograms presenting the distribution of the number and the volume of unstable blocks for all the simulations (these histograms allows to verify if the blocks still unstable after bolting have a limited volume).

In fact, the fall of small blocks going throw the bolting mesh is not really significant, because in practice the bolting goes with wire mesh that prevent the fall of small blocks.

It is to remember that the choice of an optimal pattern must include the economical considerations.

3 STABILITY ANALYSIS EXAMPLE

The analysis concerns an underground mining gallery (a length of 100m) realized in a blocky rock mass. The main sets of discontinuities are summarized in the Table 1 and are the result of a statistical analysis of the orientation and the interfractural distance.

The zone of interest is a parallelepiped of 100m of length (x axis), 20m of width (y axis there) and 15m of height (z axis), while the excavation is composed of a mining gallery with dimension 4x4m (Figure 5a).

The purpose of our analysis was to study the influence of discontinuity sets to the stability of the gallery and to propose solution to the improvement of the stability. Three sets of calculations were performed:

1-a model with no support and with a flat roof (Figure 5a);

2-a model with no support but with a cylindrical roof (Figure 5b);

3-a model with a flat roof and with a bolting support (Figure 5c). The schema adopted is the one with a density of 0.7 bolts/m^2, length of 2m and a bearing capacity of 200kN.

Figure 5d shows the representation of blocks for the case of a cylindrical roof. The module **bsa** were used to calculate the stability of each model.

For each set of calculations were realized 60 simulations. Figure 6 shows the evolution of average volume of unstable blocks vs. number of simulations. It is clearly shown that the average volume of unstable blocks is divided by two when the roof is changed from flat to circular one and the reduction of unstable blocks is very remarkable for the model using a bolting mesh.

The Figures **7**, 8 and 9 show examples of unstable blocks for particular simulations for the three models used for calculations.

Table 1. Data for discontinuity sets.

Set	Dip	Dip Direction	K of Langevin Fisher Law	Spacing (m)			Mechanical properties of joints	
				Law	Average	Standard deviation	Cohesion (kPa)	Friction angle(°)
1	50	144	500	Exp.	3.6	3.6	0	30
2	28	68	300	Lgnor.	3.33	0.31	0	30
3	68	232	1000	Nor.	2.5	0.25	0	30
4	62	85	59	Exp.	5	5	0	30

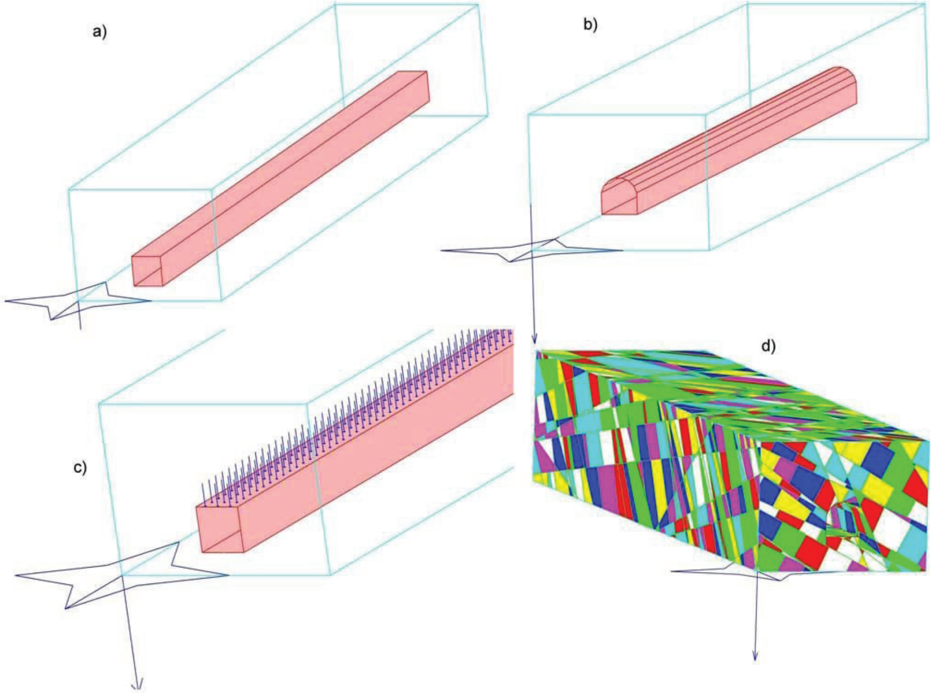

Figure 5. Models for stability analysis: a) no support and with a flat roof; b) no support with a cylindrical roof; c) flat roof with bolt support; d) representation of blocks for the case of a cylindrical roof.

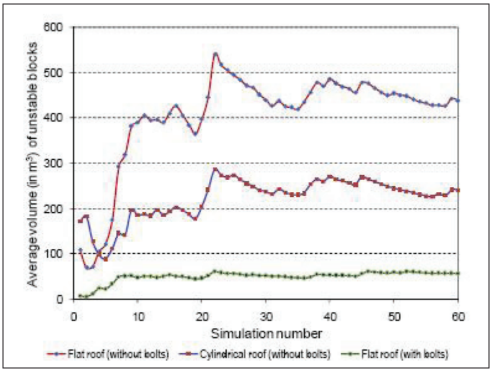

Figure 6. Variation in the average volume of unstable blocks vs. number of simulations.

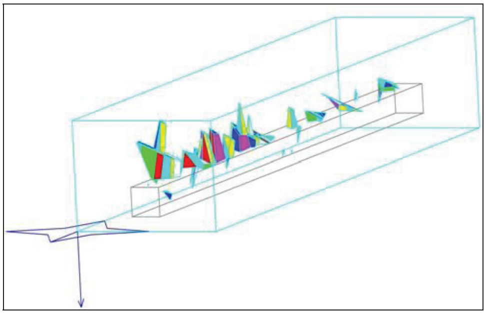

Figure 7. Unstable blocks for the flat roof, without support, for the simulation number1.

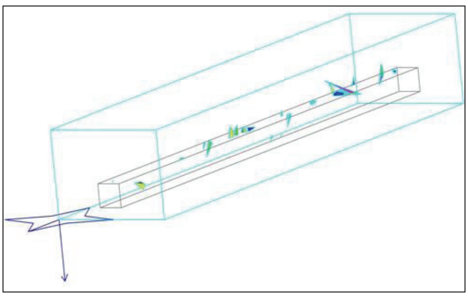

Figure 8. Unstable blocks for the flat roof, with bolting support, for the simulation number1.

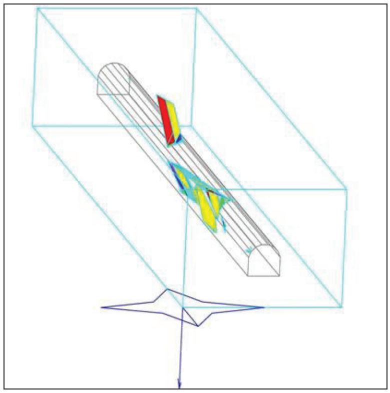

Figure 9. Unstable blocks for the cylindrical roof, without support for a particular simulation.

4 CONCLUSIONS

The software RESOBLOK is a powerful computer tool permitting the simulation of the jointing of the rock mass from deterministically or statistically processed data. Associated to a method of analysis of stability of isolated blocks, it becomes a mean to forecast the instability risks and permits, the choice of optimal excavation orientation and shape as well as the calculation of bolting support.

The bolting, introduced according to regular patterns, plays its role of support by anchoring the unstable blocks to the unremovable part of the rock mass. The answer on the stability of a bolted block is given by the calculation of the factor of safety, using the vectorial method of Warburton. The formulation of the security factor is given for all the possible cases of instability.

The modelling has been limited to the only behaviour in traction of the bolt, what excludes some other modes of ruptures (shearing etc.).

In the general case, the choice of a bolting pattern is made by the analysis of several possible propositions. The retained criterion is the minimization of the number of unstable blocks and their size, while also taking in account of the economic considerations and the complementary technical solutions (wire mesh and shotcrete). The method of calculation adopted, relatively simple, returns the fast stability analysis and permits the analysis of several patterns of bolting for several geometries of joints.

The methodology of stability analysis has been partially illustrated by one example. Nevertheless, the application to real cases must be made progressively while comparing the results of the model to the observations and measures in situ and while improving progressively the knowledge concerning the jointing of the rock mass, the mechanical features and the hypotheses of the calculations

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Some Aspects of the Elementary Ore Pieces Statics and Kinematics in the Draw Process

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ABSTRACT This paper considers basic moments of some balanced states, involving one ore piece, then two contacting each other ore pieces, supplemented by contacting ore pieces and a rock piece with spherical shape, all tangential to a draw wall. They are depicted by the static methods used for solving a plain task.

Shown are the rates of speeds and accelerations of points from the ore pieces and the contacting them rock piece. The trajectories of points from the same are shown as well. The trajectories of ore and rock pieces points performing a uniform motion are depicted by cycloids.

The considered aspects are an attempt to complement the theory of drawing broken ore, used in the systems for underground mining.

1 FORMULATION OF THE ISSUE

In the theory of drawing broken ore, the kinematics and dynamics of the process are investigated in details.

Nevertheless, some formulations of the elementary ore pieces statics and kinematics are still not investigated.

In this paper some new complementing moments of the statics, such as:

- balanced state of an elementary ore piece contacting a draw wall;
- balanced state of two elementary pieces contacting each other on a draw wall;
- balanced state of two elementary ore pieces contacting each other and a rock piece on a draw wall are considered.

From the kinematics view point the following new moments are provided:

- rate and direction of the full acceleration experienced by a point from an elementary ore piece;
- rate and direction of the acceleration of points from two elementary ore pieces, and the motion theory of points from these pieces;
- rate and direction of acceleration, as well as a motion trajectory of points from two elementary ore pieces and a rock piece point.

It is assumed that under those conditions the ore and rock pieces are with a spherical shape.

2 INVESTIGATION METHODS

In Figure 1 a scheme of balance is shown (1) for an elementary ore piece with spherical shape that contacts a lateral draw wall under angle α. In order to determine the balanced state are taken into account the following equations:

$$\sum x_i = 0$$

$$\sum y_i = 0$$

$$\sum M = 0$$

Basic determining factors are the forces along $X \rightarrow T$ and $G.\sin\alpha$; and along $Y \rightarrow N$ and $G.\cos\alpha$, as well as the moments $M_{fr.}$ and $G.\sin\alpha.r$.

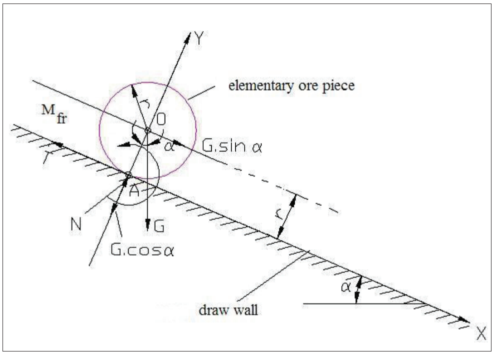

Figure 1. Balanced state scheme for a rock piece upon a lateral draw wall.

We apply the balance conditions, releasing the ore piece from the imposed on it connection, considering the borderline situation between rest and rolling.

$$\sum x_i = -T - G.\sin\alpha$$

$$\sum y_i = N - G.\cos\alpha$$

$$\sum M_{ai} = M_{fr.} - G.\sin\alpha.r$$

Where:

$$T = G.\sin\alpha$$

$$N = G.\cos\alpha$$

$$tg\alpha = f/r\,;\; M_{fr.} = f.N\,.$$

N – Is the normal reaction, [N];
T – Is the friction force, [N];
$M_{fr.}$ – is the friction moment at rolling [N.m];
f – is the friction coefficient at rolling.

In Figure 2 is shown a balanced state scheme for two elementary ore pieces with spherical shape that contact a draw wall.

From the condition for balance along *X*, decisive are the forces T, T_1, $G_1.\sin\alpha$ and $G_2.\sin\alpha$. Along *Y* the constituent forces are M_{fr1}, M_{fr2}, $G_1.\sin\alpha.r$, $G_2.\sin\alpha.3r$ and $T_1.2.r$.

For the moments, main constituents are:

$$\sum x_i = -T - T_1 - G_1.\sin\alpha - G_2.\sin\alpha$$

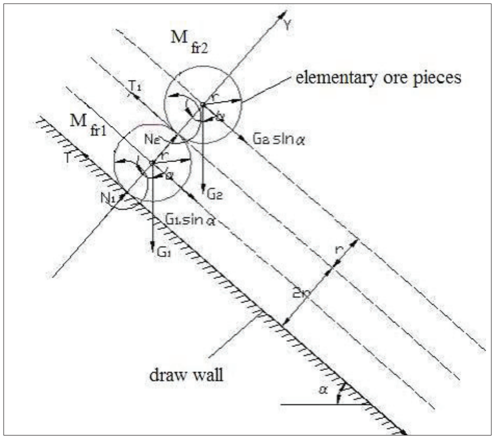

Figure 2. Balanced state scheme for two rock pieces contacting each other and a lateral draw wall.

$$\sum y_i = N_1 + N_2 - G_1.\cos\alpha - G_2.\cos\alpha$$

$$\sum M_{Ai} = M_{fr1} + M_{fr2} - G_1 \sin\alpha.r - G_2 \sin\alpha.3r + T_1.2r$$

In Figure 3 is given a balanced state scheme for two elementary ore pieces and an ore piece contacting a draw wall.

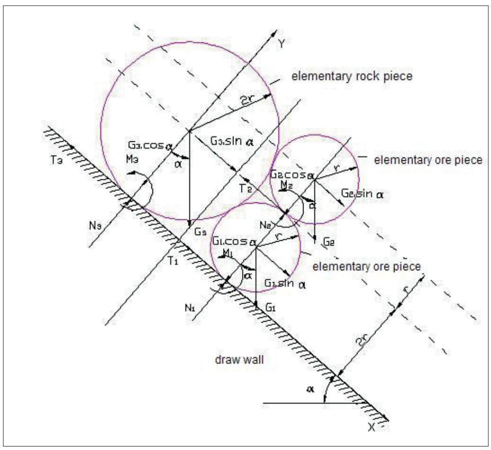

Figure 3. Balanced state scheme for two ore pieces and a rock piece.

From the condition for balance along *X*, decisive are the forces T_1, T_2, T_3, $G_1.\sin\alpha$, $G_2.\sin\alpha$ and $G_3.\sin\alpha$.

Along *Y* that are the constituent forces N_1, N_2, N_3, $G_1.\cos\alpha$, $G_2.\cos\alpha$ and $G_3.\cos\alpha$. From $\sum M_{oi} = 0$ we obtain the moments M_1, M_2, M_3, $G_1.\sin\alpha.r$, $G_2.\sin\alpha.3r$, $G_3.\sin\alpha.2r$, $T_2.2r$, $(N_1 + N_2).3r$ and $G_1.\cos\alpha + G_3..\cos\alpha.3r$.

$$\sum x_i = -T_1 - T_2 - T_3 - G_1.\sin\alpha - \\ - G_2.\sin\alpha - G_3.\sin\alpha$$

$$\sum y_i = N_1 + N_2 + N_3 - G_1.\cos\alpha - \\ - G_2.\cos\alpha - G_3.\cos\alpha$$

$$\sum M_{oi} = M_1 + M_2 + M_3 - G_1.\sin\alpha - \\ - G_3.\sin\alpha.2r + T_2.2r + (N_1 + N_2).3r - \\ - (G_1\cos\alpha + G_3\cos\alpha.3r)$$

In Figure 4, the rate of the acceleration "a" obtained in case of a circular motion of point M along a circle with radius *R* is given.

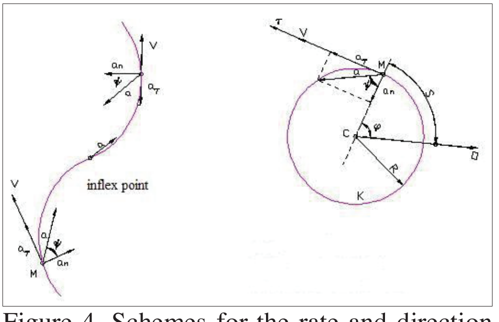

Figure 4. Schemes for the rate and direction of the full acceleration "*a*" of the ore piece point M.

Rate of the acceleration is $\vec{a} = \overrightarrow{dV}/dt = d^2r/dt^2$.

Full acceleration is

$$a = \sqrt{a_\tau^2 + a_n^2} = \sqrt{(dv_\tau/dt)^2 + V''/S^2} = Y''$$

in $\psi = \angle(a_n, a) = \mathrm{arctg}(a_\tau/a_n)$

Where: a_n - is the normal acceleration;
a_τ - is the tangential acceleration.

In the inflex point we have a change of the acceleration direction.

- In $a_\tau = a_\tau(t)$ we have the algebraic projection of the speed

$$Y' = V_\tau = V_0 + \int_{t_0}^{t} a_\tau(t)dt$$

Where V_0 - is the momentary speed of the point $t = t_0$; $[m/s]$.

- With circular motion the trajectory of point M is a circle with radius R. In this case the natural motion law is

$$\varphi = \varphi(t)$$

Where $\varphi = \angle OMC$ is the central angle between an initial radius and the current radius of point M (Fig. 4).

Due to the natural relation $S = R.\varphi$ between the curvilinear abscissa and angle *φ*, for the speed of point M is valid the formula:

$$V_\tau = R(d\varphi/dt) = R.\tilde{\omega},\ m/s$$

- With uniform motion $a_\tau = 0$; $V = V_0 = \mathrm{const}$, while the natural motion law is:

$$S = S_0 + V_0(t - t_0)$$

The full acceleration of the point is equal to the normal, i.e. $a = a_n$. It is equal to zero, if the curve is a straight line, because the curvature $\rho = \infty$.

- With uniformly variable motion $a_\tau = \mathrm{const}$, while the speed is $V_\tau = V_0 + a_\tau(t - t_0)$

The motion law being

$$S = S_0 + V_0(t - t_0) + 1/2.a_\tau.(t - t_o)^2$$

Where a_τ is the normal acceleration, read with its sign.

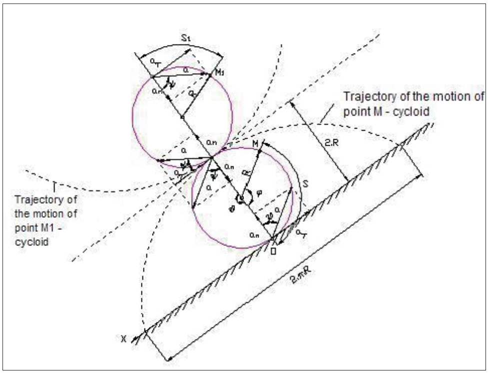

Figure 5. Scheme for ore pieces points M and M_1 rate, and direction of the full acceleration. Trajectory of the motion when the pieces contact each other.

The trajectory of points M and M_1 motion is shown in Figure 5, being for a uniform motion under the above conditions a cycloid, which is described by the following parametric equations:

$$x = R(\vartheta - \sin \vartheta)$$
$$y = R(1 - \cos \vartheta)$$

The full acceleration for point M_1 is

$$a_1 = \sqrt{a_{\tau_1}^2 + a_{n_1}^2}$$

The full acceleration for point M_3 is

$$a_3 = \sqrt{a_{\tau_3}^2 + a_{n_3}^2}$$

The cycloid of point M_1 is shown in Figure 6 and is described by the parametric equations:

$$x_1 = R_1(\vartheta_1 - \sin \vartheta_1)$$

$$y_1 = R_1(1 - \cos \vartheta_1)$$

The cycloid of point M_3 is depicted by the parametric equations:

$$x_3 = R_3(\vartheta_3 - \sin \vartheta_3)$$

$$y_3 = R_3(1 - \cos \vartheta_3)$$

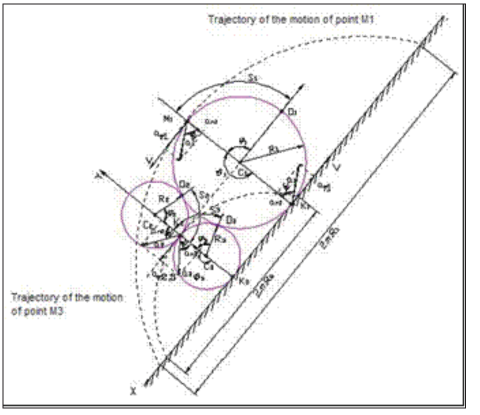

Figure 6. Motion and acceleration of points from contacting ore pieces and from a rock piece.

3 CONCLUSIONS

- The created new schemes for the balance state of elementary ore pieces supplement the draw theory, reflecting the statics view point;
- With the new schemes, employing the kinematics of ore pieces elementary points, is solved the issue related to determination of the full acceleration of those points;
- The laws for motion of ore pieces points are determined, as well as the cycloid form of the trajectories of those points.
- Determining importance for the statics and kinematics of the ore pieces and points thereof has the angle of the draw wall.

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Rock Mass Properties and Evaluation of Stability of Underground Openings – Case of Rreshen-Kalimash Tunnels

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ABSTRACT Determining rock mass properties and evaluating the stability of underground openings is a key issue for preliminary site investigation. To determine rock mass properties from intact rock and the structural features of the rock mass, are used different methods and a comparison between them is made.

Subsequently the stability of underground openings by different methods is treated. First the method of "critical depths" is used, which allows to determine the possible loss of stability of the contour and to propose the appropriate support. On the other hand, calculations are made by the finite element method and a comparison between two methods is performed. The use of finite element method is made taking also into account the technological process.

The case study concerns with Rreshen-Kalimash motorway tunnels. The Rreshen–Kalimash motorway has a total length of approximately 60 km and includes two parallel tunnels with a length of approximately 5.5 km each. The variation of rocks, structural properties and cover thickness allow to examine different stability situations and to propose the appropriate solution concerning the necessary support of the tunnels.

1 INTRODUCTION

The calculation methods for excavations in rocks have known a rapid development in last decades. The classical methods are assisted by powerful numerical ones such as finite element, finite differences, discrete element, boundary element etc. The need for reliable data is more and more present in order to correctly predict the behavior of excavations. The data, that more influence the results of calculations, concerns the rock mass properties, so determining them has become a key question in rock mechanics.

2 ROCK MASS PROPERTIES AND STABILITY CALCULATIONS

2.1 Classification Systems

As it is mentioned by Cai et al. (2004), many rock mass classification systems have been proposed and used in engineering practice, such as the RQD, Rock Mass Rating (RMR), Q, GSI, and RMI (Rock Mass index) system.

The Geological Strength Index (GSI), developed by Hoek et al. (1995) uses properties of intact rock and jointing to determine/estimate the rock mass deformability and strength. GSI values can be estimated based on the geological description of the rock mass. The GSI system concentrates on the description of two factors, structure and block surface conditions (Figure 5).

Although imperfect, the GSI system is the only system that provides a complete set of mechanical properties (Hoek–Brown strength parameters; or the equivalent Mohr–Coulomb strength parameters as well as elastic modulus E) for design purpose.

2.2 Proposed Schema for Determining Rock Mass Properties

In order to better respond to specific applications, some systems are based on the modification of the existing ones.

Such a modification of RMR system with the purpose the determination of rock mass properties is developed by Sauku (1990, 1992).

2.2.1 The cohesion of rock mass

The cohesion of rock mass (C_m) is determined from the cohesion of intact rock (C) by the following relation:

$$C_m = \frac{C}{k_r}$$

where: k_r is the reduction factor calculated by $k_r = 1 + \beta_{com} \ln (B, h) n_s$

(B, h) – the maximal dimension (width or height) in the cross-section surface of the underground working,

β_{com} –coefficient, in which are represented all the kinds of influence in rock cohesion reduction,

$n_s = 1/l_s$, number of joints per linear meter;

l_s – distance between joints (m).

Used criterion for the complex coefficient β_{com} estimation are:

a. Differentiation in three influencing factor groups: natural, technical and technological factors;
b. Gradation of influencing degree of each factor;
c. Analytic evaluation of the complex coefficient.

In the first group of natural factors are distinguished four factors: rock masses structure, rock masses jointing systems, cohesion in joints, the water containing of the rock mass after the draining by the working.

In the second group are represented technical factors as: kind of working in rock masses (underground workings and surface workings as foundations or rock slopes) and their orientation, service deadline of workings.

In the third group technological factors are represented: rock excavations technologies, rock support and buildings rhythms.

In each factor group are differentiated five degrees of influencing with minimal, low, middle, high and very high weakening effects. For each influencing degree is used a characterizing coefficient, selected after a cautious comparison procedure of all the data from our and other countries practice. Each coefficient increases gradually according to his influence in the five weathering degrees that change from one factor to another.

All these coefficients (8x5=40) are represented in Table 1. For each evaluation it must be selected eight of them (one for each factor) and finally "β_{com}" is their product:

$$\beta_{com} = \prod_{i=1}^{8} k_i$$

As it is seen, the minimal value of this coefficient is from the first column (β^{I}_{com}=0.41) and the maximal value is from the fifth column with values: for underground workings (β^{V}_{com}=5.85), for foundations (β^{V}_{com}=7.52) and for rock slopes (β^{V}_{com}=12.54).

2.2.2 The internal friction of the rock mass

While between rock and rock mass cohesion are very different values, in rock mass interlocks friction, the differences are nearly insignificant, and they are connected mostly with the change of water presence and stress conditions. It can be accepted the relation:

$$\varphi_m = k_1 k_2 \varphi$$

where, k_1 – water presence influence coefficient with values in Table 2;

k_2 – stress situation coefficient, which values, in correlation with rock loading index, are presented in Table 3.

Table 1. Values of k_i coefficients.

k_i	Influencing factor	Values of coefficient for influencing index				
		Very low	Low	Average	High	Very high
	A. Natural factors					
k_1	Rock masses tectonic structures	Undisturbed	Monoclinal	Wrinkled	Wrinkled and faulted	Very disturbed
		1.0	**1.05**	**1.10**	**1.15**	**1.20**
k_2	Joint systems (by joint orientations)	Scarce	2 systems	3 systems	Many systems	Irregular
		0.8	**1.0**	**1.10**	**1.20**	**1.30**
k_3	Rock joints cohesion *) n_c = 100 Co/C, %	Very high n>15	High n=15~10	Average n=10~5	Low n=5~1	Very low n<1
		0.8	**1.0**	**1.10**	**1.20**	**1.30**
k_4	Water containing and draining from rock mass (l/min per m)	Very dry	Humid	wet	leaking	Trickling
		-	<1.0	1~2.5	2.5~10	>10
		1.0	**1.05**	**1.10**	**1.20**	**1.30**
	B. Technical and technological factors					
k_5	Working axis **) orientation for Galleries (G), Foundations (F), Slopes (S)	Very favourable	Favourable	Fair	Un-favourable	Very unfavourable
		G / F / S	G / F / S	G / F / S	G / F / S	G / F / S
		0.8 / 0.8 / 0.8	**1.0 / 1.0 / 1.2**	**1.1 / 1.3 / 1.6**	**1.2 / 1.0 / 2.0**	**1.4 / 1.8 / 3.0**
k_6	Service dead line	< 1 year	1~3 years	3~10 years	10~25 years	> 25 years
		0.9	**1.0**	**1.05**	**1.10**	**1.15**
k_7	Excavation technology	Mechanical	Hydraulical	Combined with blasting	Controlled blasting	Uncontrolled blasting
		1.0	**1.05**	**1.1**	**1.15**	**1.20**
k_8	Excavation speed (excavation + support)	Very high	High	Average	Low	Very low
		0.9	**0.95**	**1.0**	**1.10**	**1.20**

Table 2. Values of k_1 coefficient.

Water presence	dry	minimal	average	high	Very high
k_1	1	0.98	0.95	0.9	0.85

Table 3. Values of k_2 coefficient.

$\gamma H/\sigma_{ci}$	<0.5	0.5-1	1-2	2-3	>3
k_2	1.0	0.98	0.96	0.94	0.92

Exceptions are clayey rocks, that, in water presence, may develop swelling phenomena and lower values of φ_m.

Finally, from rock mass properties we can calculate the uniaxial compressive strength of the rock mass by the following relation:

$$\sigma_{cm} = \frac{2 \cdot C_m \cdot \cos\varphi_m}{1 - \sin\varphi_m}$$

2.3 Stability Calculations

The study of stability of an underground opening is performed distinguishing the critical stability in roof, sidewall and floor and is based in the method of "critical depths" (Sauku, 1990, 1992).

2.3.1 The critical depths

Three critical depths are distinguished (H_1, H_2, H_3), respectively for roof, sidewall and floor instability (Table 4).

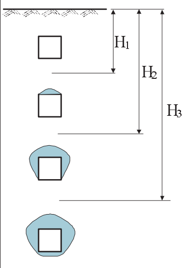
For a general view of underground workings stability with variability in cross section area and depth, it is used a graphical presentation in parametric coordinates ($\sqrt{S/S_0}, \gamma H / \sigma_{ci}$) (Figure 6), where, S_0 = 4m^2 is a minimal cross section of reference.

The H_1, H_2, H_3 curves delimit four stability zones:

- Full stability for $H<H_1$;

- Roof instability for $H_1<H<H_2$;
- Roof and sidewall instability for $H_2<H<H_3$;
- Full instability for $H>H_3$.

Table 4. Calculation schema for "critical depths".

	$H_1 = \dfrac{C_m}{k_n \cdot \gamma \left(\dfrac{a}{h_s} - \lambda \cdot \tan \phi_m \right)}$ $H_2 = \dfrac{2 \cdot C_m \cdot \tan \dfrac{90 + \varphi_m}{2}}{k_t \cdot k_n \cdot \gamma}$ $H_3 = \dfrac{C_m}{2 \cdot k_t \cdot k_n \cdot \gamma \cdot \tan \varphi_m} \cdot E_0$ $E_0 = \left(e^{\pi \tan\varphi_m} \cdot tan^2 \dfrac{90 + \varphi_m}{2} - 1 \right)$
a -	half width of the opening
h_s -	the thickness of first rock bed on the roof, m. If does not exist, then $h_s=l_d$
l_d -	the distance between discontinuities, m
$\lambda = \nu/(1-\nu)$	, ν - Poison ratio
k_n -	coefficient due to adjacent openings influence
k_t -	coefficient due to sidewall stresses
Cm -	interblock cohesion, Mpa
φ_m -	interblock friction, degrees
γ -	unit weight of rock, kN/m^3

2.3.2 *Rock pressure*

Rock pressures are calculated in kPa as roof normal pressures (p_r), sidewall normal pressure (p_s) and floor normal pressure (p_f) (Sauku, 1990, 1992). Usually p_r represent the maximal normal pressure.

Graphically, by contours, are separated the normal pressure fields. They are:

- Low pressure field, p_r = < 50 kPa;
- Average pressure field: p_r = 50 - 150 kPa;
- High pressure field: p_r = 150 - 300 kPa;
- Very High pressure field: p_r > 300 kPa.

2.4 Calculations Using Numerical Methods

The most common failure criteria used in rock mechanics calculations are Mohr-Coulomb, Drucker-Prager and Hoek-Brown criteria.

In order to have reliable results the parameters needed to perform numerical calculations (by finite element method, finite differences method etc.) must reflect the values corresponding to the in-situ rock mass.

2.4.1 *Deformability of rock mass*

Among the different formulations allowing determining the deformation modulus from rock mass classification systems we may distinguish the followings:

1. The relationship between the deformation modulus and the Rock Mass Rating (Serafim & Pereira, 1983) as follows:

$$E_m = 10^{\left(\frac{RMR-10}{40}\right)} \quad \text{(GPa)}$$

2. Barton *et al.* (1985) have found good agreement between measured displacements and predictions from numerical analyses using in-situ deformation modulus values estimated from

$$10\log_{10} Q < E_m < 40\log_{10} Q \text{ and}$$

$$E_{m(mean)} = 25\log_{10} Q$$

3. Hoek *et al.* (2002) proposed the following relation in order to obtain the deformation modulus form GSI, disturbance factor D and intact rock mass uniaxial compressive strength σ_{ci}:

$$E_m = \left(1 - \frac{D}{2}\right) \cdot \sqrt{\frac{\sigma_{ci}}{100}} \cdot 10^{\left(\frac{GSI-10}{40}\right)}$$

where E_m is calculated in GPa.

4. Zhang & Einstein (2004) proposed the following relations between RQD and deformation modulus:
Lower bound: $E_m / E_r = 0.2 \cdot 10^{0.0186RQD-1.91}$
Upper bound: $E_m / E_r = 1.8 \cdot 10^{0.0186RQD-1.91}$
Mean: $E_m / E_r = 10^{0.0186RQD-1.91}$
where E_m and E_r are the deformation moduli, respectively, of the rock mass and the intact rock.

2.4.2 Parameters of strength criterion

The parameters needed are determined using rock mass classification systems.
For example for Hoek-Brown criterion

$$\sigma_1^{'} = \sigma_3^{'} + \sigma_{ci}\left(m_b \frac{\sigma_3^{'}}{\sigma_{ci}} + s\right)^a$$

the empirically determined parameters m_b, s and a are given by Hoek *et al.* (2002):

$$m_b = m_i \cdot e^{\frac{GSI-100}{28-14D}}$$

$$s = e^{\frac{GSI-100}{9-3D}}$$

$$a = \frac{1}{2} + \frac{1}{6}\left(e^{\frac{-GSI}{15}} - e^{\frac{-20}{3}}\right)$$

For Mohr-Coulomb failure criterion

$$\sigma_1^{'} = \sigma_{cm} + k \cdot \sigma_3^{'}$$

where $k = \dfrac{1+\sin\varphi_m}{1-\sin\varphi_m}$

and (Hoek *et al.* 2002):

$$\sigma_{cm} = \frac{2 \cdot C_m \cdot \cos\varphi_m}{1-\sin\varphi_m} = \sigma_{ci}\frac{\left[m_b + 4s - a(m_b - 8s)\right] m_b^{a-1}}{2(1+a)(2+a)(4+s)^{a-1}}$$

To calculate GSI it is possible to use the following relation (Hoek, Kaiser & Bawden, 1995):

$GSI = RMR - 5$ (for $GSI > 25$),
(for $RMR < 23$ use Q classification).

For C_m and φ_m it is also possible to use parameters obtained by the formulation in the section 2.2.

3 APPLICATION TO THE RRESHEN-KALIMASH MOTORWAY TUNNELS

3.1 Description of the Site

The Rreshen–Kalimash motorway has a total length of approximately 60km and includes two parallel tunnels with a length of approximately 5.5km each. They have a horse-shoe shape with a total height of approximately 8.7m and a width of 11.3m (Figure **1**).

Figure 1 shows the typical cross section of the tunnels and Figure 2 a view of west portal.

Figure 3 shows a longitudinal profile along one of the tunnels. The geological profile of the ground consists of gabbro, pyroxenite and dunite rocks with various degrees of jointing and alteration.

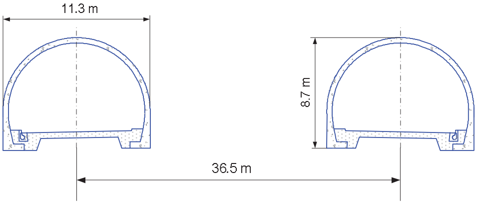


Figure 1. Cross section of tunnels.

Figure 2. View of the west portal.

Table 5 and Table 6 show the distribution of RMR classes for different depths and the rock loading index (RLI=$\gamma H/\sigma_{ci}$) for each depth for three categories of uniaxial compressive strength (UCS) of rocks (60, 80 and 100 MPa).

Table 5. RMR class repartition.

RMR class	Length (m)	%
II	1230	22.5
III	3385	61.9
IV	715	13.0
V	140	2.6

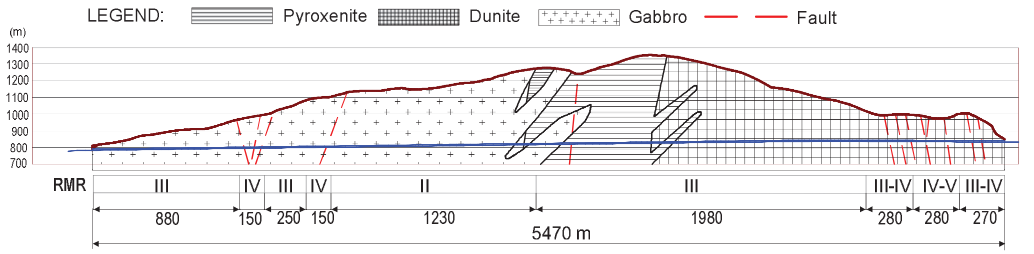


Figure 3. Longitudinal profile of the tunnel.

Table 6. RLI & the distribution of RMR classes.

RMR Class	DEPTH (m) & $\gamma H/\sigma_{ci}$											
	< 200			200 - 300			300 - 400			400-500		
	0.09	0.07	0.06	0.14	0.11	0.08	0.19	0.14	0.11	0.23	0.18	0.14
II							▨	▨	▨			
III	▨	▨	▨	▨	▨	▨				▨	▨	▨
IV	▨	▨	▨	▨	▨	▨	▨	▨	▨			
V	▨	▨	▨									

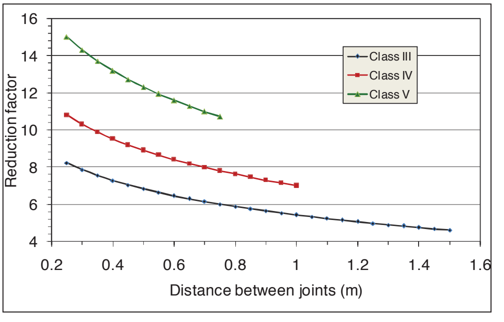


Figure 4. Reduction factor vs. distance between joints.

Tunnels are excavated according to the principle of the New Austrian Tunneling Method (NATM). First stage supports has been installed by using rockbolt, shotcrete and steel support.

3.2 Evaluation of Stability

3.2.1 Calculation of rock mass properties

For the calculation of rock mass properties is used the procedure described in the section 2.2.

Three values for the intact UCS of rocks are considered, respectively 60, 80 and 100 MPa. Using Table 1 we calculate the reduction factor (k_r) as function of distance between joints for different situations of natural and technological factors (Figure **4**) corresponding to the RMR classes. As we can see the reduction factor for rock mass properties may vary from 5 (in the best case) to 15 (in the worst case).

On the other hand analyzing the joint surface quality and the blocky composition of the rock mass it was possible to provide the GSI for the different RMR classes (Figure 5) in the form of a zone corresponding to each class.

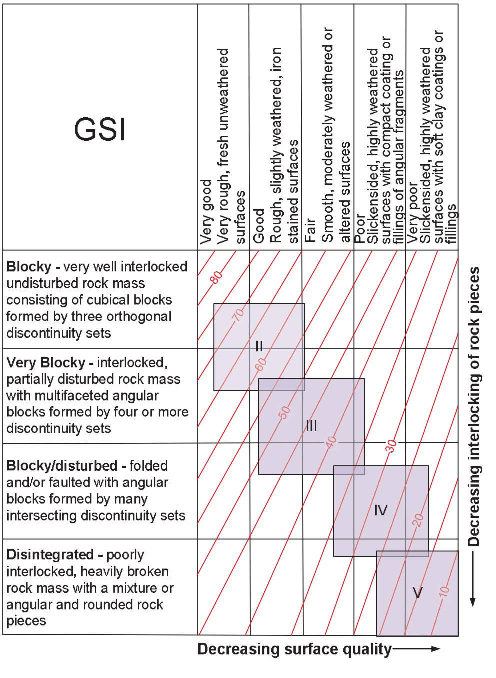


Figure 5. GSI for different RMR classes.

For the friction angle, for $\gamma H/\sigma_{ci}$<0.3 and for dry conditions practically the friction angle doesn't change so we have $\varphi_m \approx \varphi$.

Considering the selected intact rock UCS it is possible to calculate rock mass properties as function of distance between joints.

3.2.2 Stability analysis

For RMR classes III, IV and V we have found that the values of β_{com} are respectively 2, 2.7 and 3.9. With these values it was possible to represent the rock pressure fields.

Figure 6 shows the stability fields for the case of β_{com}=2 and the position of tunnels for the depth from 0 to 500 m corresponding to an average UCS of rocks.

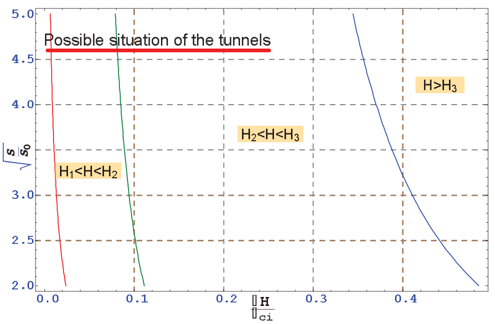


Figure 6. Stability fields and the position of tunnels.

Figures 7 and 8 show the stability fields and the normal pressure corresponding to β_{com}=2 and β_{com}=2.7 for various number of joints per meter. We can notice that for the number of joints per meter going from 2 to 3 we stay in average pressure field until a depth of 200 or 250m, and for a higher number of joints per meter the average field pressure is until 150m and for higher depths we enter to the high pressure field.

After analyzing different situations corresponding to various RMR classes (taking account of β_{com}), RLI and depths, it is possible to give recommendations concerning the pressure field and support characteristics (Table 7).

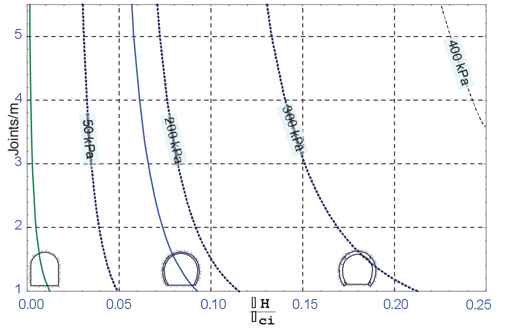


Figure 7. Stability fields and rock pressure for β_{com}=2.

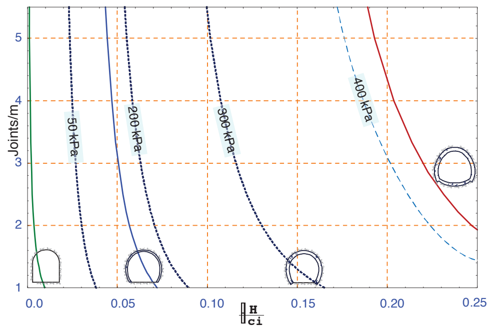


Figure 8. Stability fields and rock pressure for β_{com}=2.7

Table 7. Recommendation for support characteristics.

RMR Class	DEPTH (m) < 200	200 - 300	300 - 400	400-500
II			150 -250 kPa	
III	50 - 100 kPa	150-250 kPa		250-3 50 kPa
IV	150-250 kPa	200-300 kPa	300-400 kPa	
V	200-3 50 kPa			

3.3 Numerical Simulation of the Studied Tunnels

3.3.1 Outline of numerical analysis

Numerical simulations were conducted by finite element method using the code ASTER (a software package for finite element analysis in structural mechanics developed by the French company EDF).

Geometry and boundary conditions of the finite element mesh are shown in Figure 9.

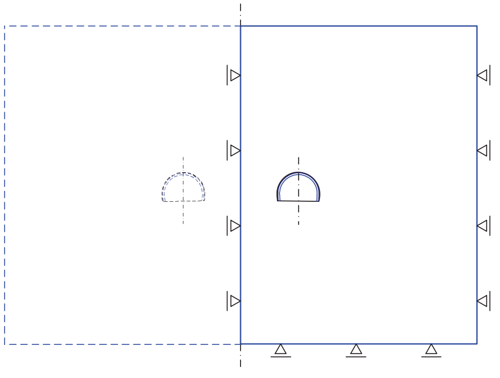


Figure 9. Geometry and boundary conditions of the finite element model.

Due to symmetry in the geometry, only the right half of the tunnels (only one tunnel) has been analyzed. In the analysis, boundary domain has a sufficient extent of some 4 times (downward) and 4.5 times (sideward) of tunnel diameter from tunnel axis.

Left and right sides of the mesh are horizontally fixed and the bottom nodes are vertically fixed. The ground behavior was modeled with two different constitutive laws, namely 1) an elastic model and 2) elastic-plastic material model with a Drucker-Prager failure criterion.

Simulations have been performed in several computational steps in order to simulate the excavation and the installation of support.

The support, modeled as elastic element, has been put in place after a stress release of 50%, simulating the technological process and the face advance.

3.3.2 Rock mass properties and calculation parameters

The horizontal stress ratio is admitted equal to 0.5.

An effort was made to determine the rock mass properties needed for the calculations using rock mass classifications. Particularly the two most important parameters, the Young's modulus and the cohesion were determined from the procedure explained above. The results obtained for the rock mass modulus and cohesion are presented in the Table 8 & Table 9.

Finally using parameters of Tables 8 & 9 and the data concerning the concrete support and other bibliographical data for finite element calculations, we accepted the material property values as in Table 10.

Table 8. Estimated rock mass deformation modulus.

RMR Class	RMR	Estimated GSI	E_m (GPa) using RMR	E_m (GPa) using GSI*)
III	41 – 60	36 - 55	5.7 - 18	3.4 - 10
IV	21 – 40	20 - 35	1.8 – 5.6	1.3 – 3.2
*) Calculated for D=0.5 and σ_{ci}=100 MPa				

Table 9. Estimated rock mass cohesion and UCS.

RMR Class	Reduction factor k_r	C_m*) (MPa)	σ_{cm}*) (MPa)
III	6.8	4.2	14.5
IV	9.9	2.9	10
*) Calculated for σ_{ci}=100 MPa and φ=30°, distance between joints 0.35m for class IV and 0.5m for class III.			

Table 10. Rock mass and concrete parameters.

	RMR Class	E_m (GPa)	ν_m	φ_m (°)	C_m (MPa)	σ_{cm} (MPa)
Rock material	III	9.3	0.25	30	4.2	14.5
	IV	3	0.25	30	2.9	10
Concrete		20	0.2	-	-	-

For the concrete we adopted a thickness of 20 cm.

Numerical simulation, firstly, is carried out with an elastic model with and without support for the two rock mass properties. After, it was performed with the elastic-plastic material model with a Drucker-Prager failure criterion, again, with and without support, for the two rock mass properties.

3.3.3 Results

Figure 10 shows the displacements for the elastic model without and with support, corresponding to the class III and to the class IV of rock mass material.

Figure 11 shows the evolution of plasticity zone for rock mass of class III (on the left) and class IV (on the right) in presence of support. The development of such zone indicates the loss of stability. In those cases only an appropriate support can guarantee the stability of the tunnels. The extension of plasticity zone may lead to the failure of support on the sidewalls of the tunnels (Figure 12). These results are in accordance with the results of calculations obtained by the method of "critical depths" developed above.

Figure 13 shows the displacements induced to the support. The maximum of displacement is reached in the crown and can conduct to the failure of the support as it was observed in-situ (Figure 14).

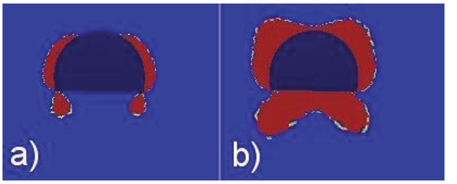


Figure 11. Evolution of plasticity zone: a) Class III rock with support; b) class IV rock with support

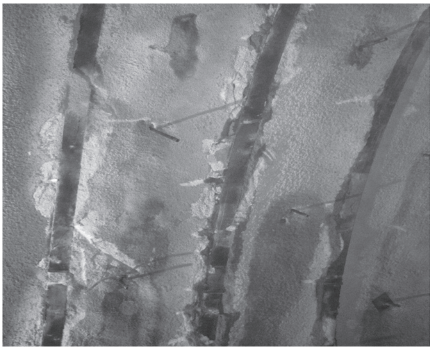


Figure 12. Observed initiation of failure in the sidewall support.

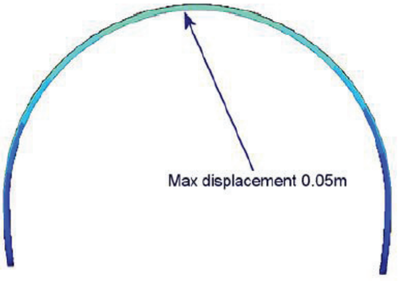


Figure 13. Displacements of the support.

Figure 14. Roof failure of the support.

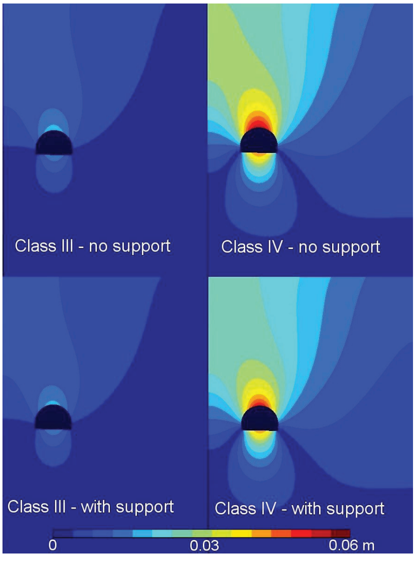


Figure 10. Displacements for elastic calculations.

4 CONCLUSIONS

This paper has studied the determination of rock mass properties and the evaluation of stability of underground openings.

Parallel to widely spread methods of rock mass classification systems, was developed a particular one in order to better respond to specific applications. The proposed method makes use not only to the structural properties of the rock mass but also to the technical and technological factors of construction of the underground opening.

This method was developed in accordance with other methods like RMR, GSI etc., making efforts to give the relations and the possible correspondence among each other.

Subsequently the stability of underground openings by different methods was treated. First, the method of "critical depths" was used, which allows to determine the possible loss of stability of the contour and to calculate rock pressures in order to propose the appropriate support.

A particular attention was paid to the methods of obtaining the necessary parameters in order to perform calculations by numerical methods.

The case study concerns with Rreshen-Kalimash motorway tunnels. It was possible to consider the variation of rocks, structural properties and cover thickness in order to examine different stability situations and to propose the appropriate solution concerning the necessary support of the tunnels.

The numerical analysis produced results that are in good agreement with the results of calculations by the method of "critical depths" and also in agreement with the observations in-situ for the case study.

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Stability Aspects of Excavations in Landslide Zone for a Coal mine "Suvodol" - R.Macedonia

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ABSTRACT: The paper presents some elements of stability aspects and approach used for excavation at the toe of the large landslide which exists on a NE part of coal mine "Suvodol" in R.Macedonia. Its volume is about 30,000,000 m^3. Upper of the main scarp, the earth-fill dam with a length of about 1000 meters exists. As a result of mass movements, about 8,000,000 tonnes of coal is blocked at the toe of the landslide. After a longer time of exposition to the air, the crushed coal is partially involved in a process of self-burning.

Specific combination of natural and man-made elements that control the stability of the area and specific methodology used for an excavation at the toe of sliding mass is shortly explained.

The results from the stability analyses, risks scenarios, some specific comments and recommendations, are summarized in this paper**.** Some specific comments are also given.

1 INTRODUCTION

The coal mine "Suvodol" is placed on SW part of the Republic of Macedonia.

It is a main source for thermal-electricity plants with coal production of about 6,500,000 tonnes per year.

At NE part of the mine, during 1995 large landslide appeared. The appearance of the landslide caused some difficulties in the normal work of the exploitation systems and it was a potential danger for the upstream earth dam which is spaced about 250 m from the main scarp of the landslide.

In order to adopt the technology of the excavation to such complex conditions, several interrelated steps of investigations and design were applied, starting from the time of occurrence till present moment.

For example, authors of the paper were involved in several phases of landslide investigations, as well as in some design phases. The investigations were complex and with large quantity, in order to prepare data for physical and analytical modeling.

Later, the data are used as a base for stability and dewatering analyses, protection from surface and groundwater's, excavation conditions and so on.

The methodology and results from investigation are explained by Gapkovski *et.al.* (1996a,b); Jovanovski & Gapkovski (1996 and 1997). The design elements are given by Jovanovski *et.al.* (2007a,b) and Panovski *et.al.* (2007).

2 OVERVIEW ON LANDSLIDE ELEMENTS AND CHARACTERISTICS

2.1 History Of Event, Problems And Scale Of Event

The initial phase of landslide activation was at the end of 1995, but several large reactivation phases were also present in 1997 and 1998. Some smaller movements were also present in parts of the landslide continuously.

One of the main problems (risks) was to solve problem with possible extension of the sliding process retrogressively to the upstream earth-fill dam direction.

This was solved using of techniques of dewatering on the space between dam and main landslide scarp with vertical and horizontal systems. The main idea was to decrease hydrostatic and hydrodynamic forces and to reduce groundwater inflow to the landslide zone. Other aspects were to define newly created conditions and to see possibility of excavate the blocked coal at the toe of landslide. In order to illustrate the scale of the event, the main elements of the landslide are given in Table 1.

Table 1. Main landslide elements.

Landslide element	*Value*
Length (m)	About 1700
Wide (m)	Min 650
	Max 880
Area (m^2)	About 1,050,000
Volume (m^3)	About 30,000,000
Depth to sliding zone (m)	Min 14
	Max 55

In Figure 1, we present map with relative subsidence and uprising of the field, after the phase of main activation.

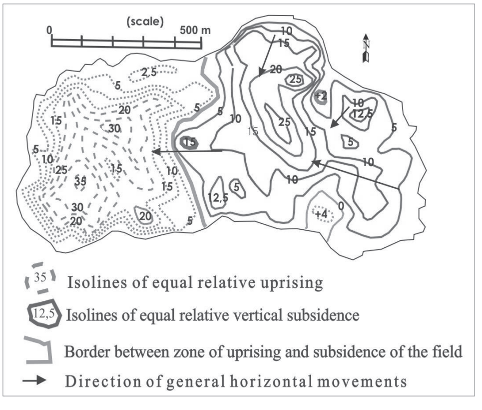

Figure 1. Map with relative subsidence and uprising of the field.

The map is prepared using elevations of the ground before and after main sliding process.

In Figure 2, we present the map with thickness of sliding mass.

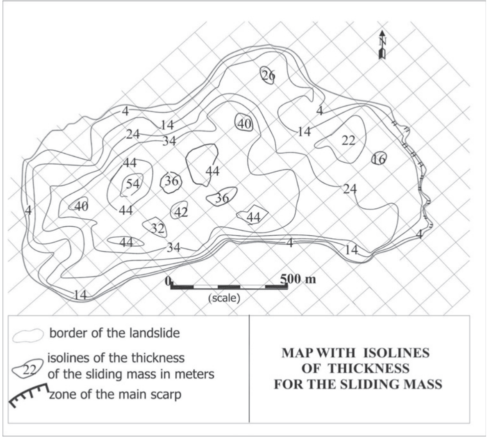

Figure 2. Map with thickness contour lines of sliding mass (in meters).

In this case, groundwater conditions have the greatest influence on the stability. The presence of the aquifer zone under pressure and gas influences is defined for all area bellow the disturbed coal layer.

An important and very restrictive additional factor was a process of self-burning, which happened because of coal direct exposition on the fresh air.

Results from the investigations, indicates that the sliding surfaces are very deep, usually along coal-like clay and silts with high plasticity. One of the main characteristics is also the great heterogeneity of the lithological composition.

A lot of secondary scarps and zone of “secondary toe” were also defined. Artesian effects are directly observed during drilling. Huge quantity of sand was transported from drilling bottom to the ground surface, because of high artesian pressures and hydraulic gradients.

Shortly, the problem is too complex and unique, that every technical action is always connected with numerous restrictions and risks, which is shortly explained in this article. To illustrate better the scale of the process, landslide elements in horizontal view are given Figure 3.

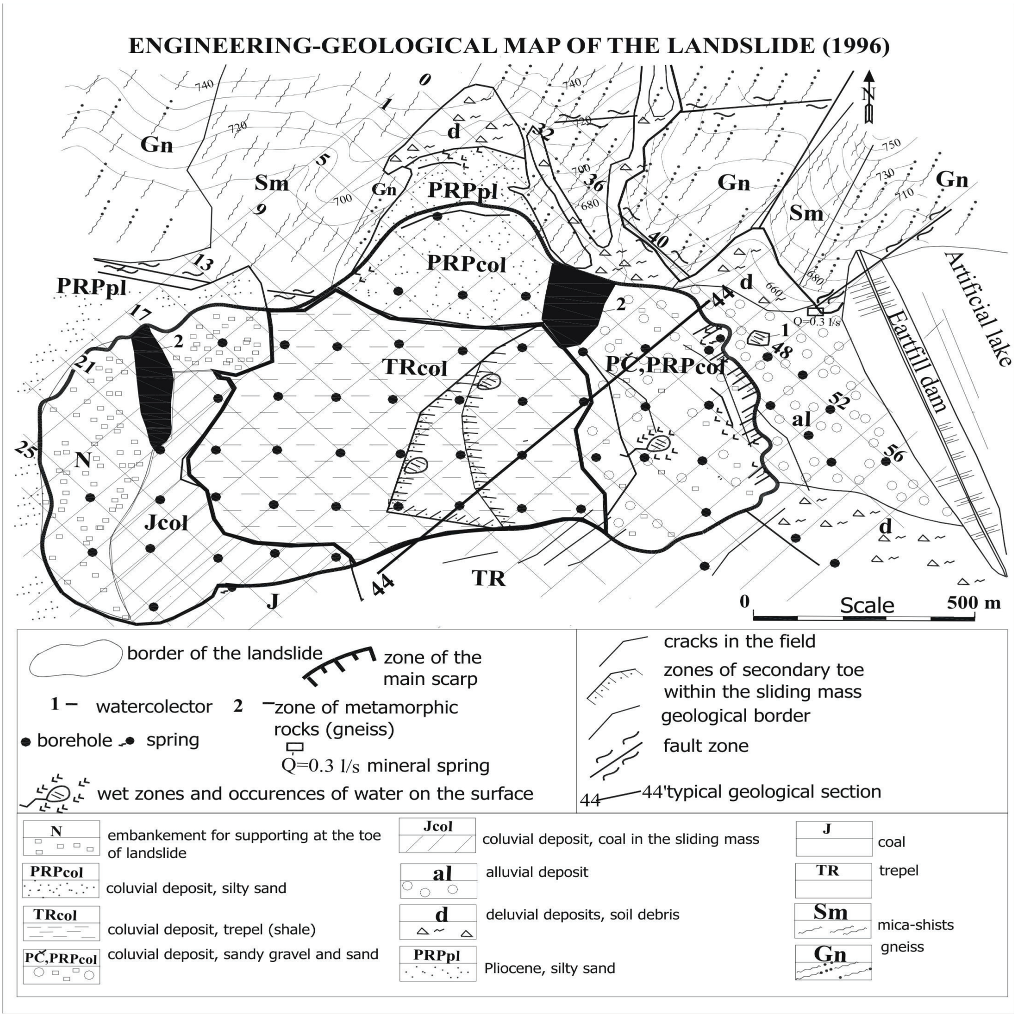

Figure 3. Engineering-geological map of landslide zone and surrounding.

2.2 Geotechnical Parameters

The most important characteristics for the main sediments are: the high plasticity and water content of coal like clay and silts, high value of uniformity coefficient, low shear strength of coal-like clay and silts with high plasticity, and high decreasing of shear – strength properties of main group of sediments after sliding process.

The typical range of physical properties is given in Figure 4.

The range of values for shear strength is given in Figure 5. The cases before and after sliding are presented.

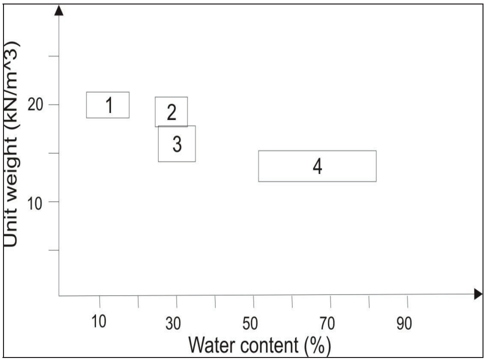

Figure 4. Range of water content and unit weight for quarterian sandy clay (1), silt (2), coal-like clay (3) and trepel (4).

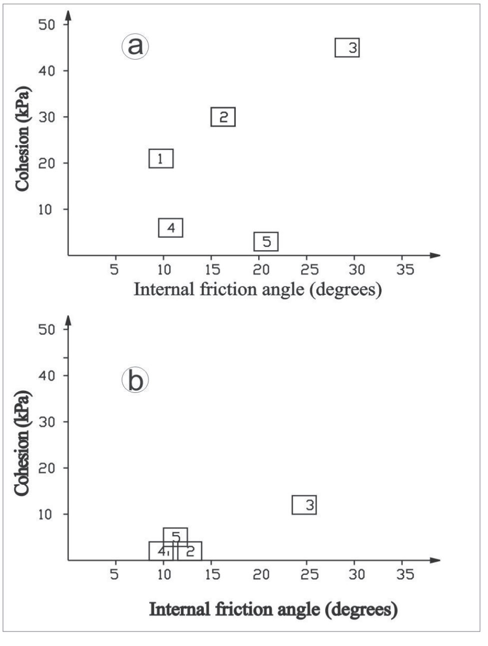

Figure 5. Range of internal friction angle and cohesion for quarterian sediments (1), trepel (2), coal (3), coal-like clay (4) and sandy silts (5) before (a) and after sliding (b).

It can be noted that, the shear strength after sliding is reduced to residual shear strength parameters, which is taken into the consideration in stability analyses.

3 POSSIBLE RISKS SCENARIOS CONNECTED WITH STABILITY

It is a well known fact, that the landslides are occurrences where time-dependent behavior can be very important.

A special problem for management team at coal mine was negative influences on the environment because of gas production during self-burning. This leads to constant loosing of the coal mass and decreasing of mechanical properties of the exposed coal.

All this is at the most critical zone which is not favorable from stability aspect. Covering of the zone of self-burning will lead to final closure of this zone. This is very unfavorable for future exploitation in so-called bottom coal series (which lies below main coal layer). Having in mind such restrictions, an adequate design analyses are prepared in order to find best-possible solution for this not typical mining case. In general, two scenarios are possible (Fig.6 and Fig. 7).

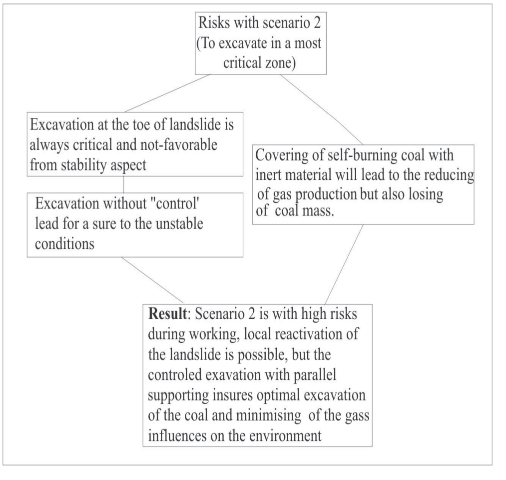

Figure 6. Scenario 1 (Do nothing).

We want to note, that in decision making, we used also methodology of so-called interaction matrix method firstly introduced by Hudson 1993.

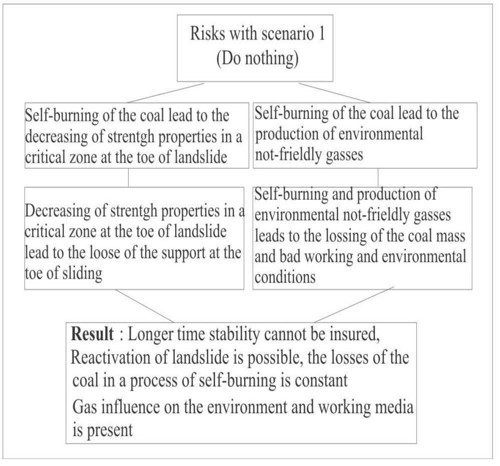

Figure 7. Risk scenario 2.

The most important step in this methodology is to establish the objectives of the project and the analysis. The relevant problems are placed along the leading diagonal of conceptual interaction matrix.

Then, all the interactions are established and the problem structure is developed. Example of relevant interaction scheme is given in Figure 8.

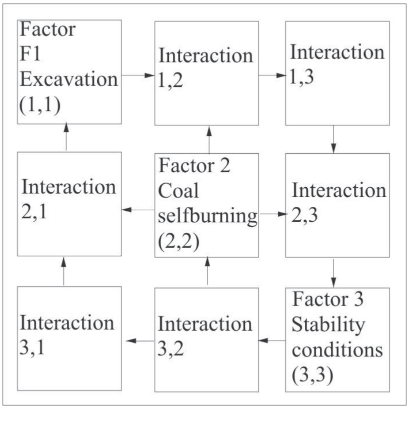

Figure 8. Conceptual matrix of interaction between tree basic factors.

F1 group of factors is related to the technology of excavation such as: applied method, depth of excavation, way of transportation, dewatering concept and so on.

F2 group is related to the characteristics of self-burning process (area of burning, intensity, gas production etc.).

Group of factors F3 is related to the stability of the field, defined with movements of the masses and safety during the working.

All possible interactions in a fist place are defined qualitatively, which is very important step for engineering judgment and decisions. Explanations are given as follows:

Interaction 1, 2 means that elements of the excavation can have an influence on the process of self-burning, because faster and efficient nearby excavation can stop the spreading of burning in wider areas.

Interaction 1, 3 means that the elements of the excavation have a direct influence on the stability conditions, because correctly designed and applied technology of excavation create stable field conditions.

Interaction 2, 3 mean that the process of self-burning during longer time has an influence on the shear strength parameters, and leads to possible unstable conditions (beside other negative influences).

Interaction 3, 2 shows that the stability of the field is governing element which affects possible access to zones of self-burning.

Interaction 3, 1 means that stability of the field affects the way of excavation technology in numerous ways.

Interaction 2, 1 means that the process of self-burning influence the excavation process, because of difficulties in access and on heavy working conditions.

It is obvious that such "simple" matrix shows several complex mutual influences between the environment and the engineering activities, and all of this shall be incorporated in design.

4 STABILITY ASPECTS

4.1 Methodology Of Analyses

An important step in mining practice is to find solution for stability problems in an appropriate way. This is certainly valid for this case.

After detailed analyses, a heavy decision was accepted.

It was decided that it is better to start with excavation with all possible negative consequences, than to allow losing a high quantity of coal in a process of self burning. In fact, this is scenario 2.

In decision making, the fact that the final result, in both scenarios is the same situation - to have instability due to decreasing of the volume of the coal in the toe of the landslide, was a reason to go into acceptable risk.

During the design, the technical solution with so-called methodology of parallel excavation and supporting was analyzed. After that, this technology was applied in a practice.

Detailed stability analyses were the basis for development of strategy for excavation.

The analyses are prepared on some representative profiles using the software package SLIDE 5. This is a known product of Canadian company Rocscience, and allows analyses with known limit equilibrium methods (Bishop, Spencer, Janbu and others).

In calculating, different phases of excavations and scenarios are involved.

For example, in Figure 9a, we illustrate a value of safety factor (Fs=1.04) before any kind of engineering activities.

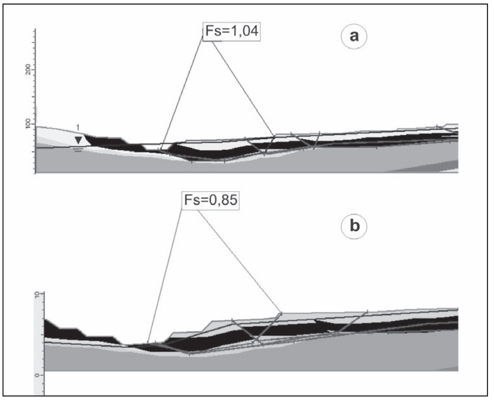

Figure 9. Outputs from stability analyses for initial phases of excavation.

In Figure 9b, we illustrate hypothetical value of safety factor. This is a case if we have a case without parallel support of excavated zone when the safety factor is bellow Fs=1.

Figure 10 explains cases of parallel support and decreasing of artesian pressure, when the safety factor has values Fs=0.98 and Fs>1.1 respectively.

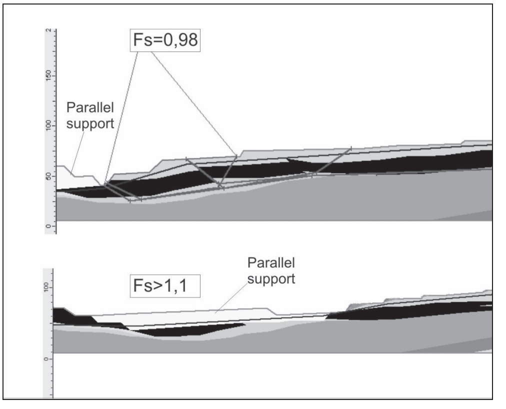

Figure 10. Analyzed cases with parallel excavation and support.

In a practice, this case can be explained as a state of allowable deformations in a term of slow (controlled) sliding, which is expected during initial phases of excavations.

In Figure 11, we give an estimation of influence of self-burning process during a longer time.

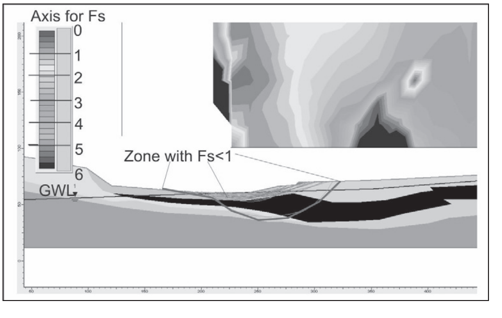

Figure 11. Hypothetical case to predict self burning influence on the stability conditions.

For the case in Figure 11, it was estimated that the upper zones of coal, during longer time will be transformed into coal ash. Minor unit weight and internal friction angle are used in analyses.

Results shows that the "new" sliding surfaces can be expected with values of safety factor Fs<1 (unstable state).

4.2 Presentation of the Executed State

It is a well known fact, that all engineering solutions should be checked directly in a practice.

This was especially important in this unique case, because almost all analyzed cases gives safety factor values that are usually not allowed in the mining practice.

The question is why the designers went into the calculated risk and allow excavations in critical zone are explained in risk scenarios.

The idea was to excavate maximal quantity of the coal from one side, and to stop the process of self-burning, from other side.

Of course, all measures of surface dewatering, visual and geodetically observations for control of possible rapid movements was strictly applied.

The excavation was allowed only with discontinued type of equipment, because it can be evacuated from critical zones in a fast way (if necessary).

We can underline that with such approach, till now, about 3,000,000 tonnes of coal is already excavated from this critical zone.

As it was expected, during excavation phase, some minor mass movements were observed. In every case, some rapid movements don't happen.

5 CONCLUSIONS

The given article is an example, that sometimes, in the practice, it is necessary to deal with unusual cases and to face with high risks.

This must not being not a rule but only the exceptions from the rules.

In every case, this article shows clearly that, it is fundamental for successful design of each engineering activity to get acquainted in detail with the properties and conditions of the working and natural environment.

All approaches in investigating and design shall be completely adapted to the characteristics of the natural environment; it is not possible to define the physical model of the terrain.

The physical model of the terrain must be the base for all numerical and mining analyses.

It is obvious that qualitative aspect of a problem, defined with checking procedure and interaction matrix method, can be useful approach in decision making.

Defined interactions are a good basis for complex analytical and numerical analyses, where the interactions can be defined with all necessary outputs (safety factors, stress-strain conditions, groundwater quantities etc).

Such approach can be adapted for numerous engineering problems, but it is necessary to have a team of specialists in mining, geological and geotechnical engineering, to solve such heavy engineering problems in an appropriate way.

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Project for Observation of the Deformations of the Massif in Mine Ellatzite

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ABSTRACT The open-pit mine "Ellatzite" is between the deepest and the steepest mines in the world because of this it is a case of high importance to create a continuous monitoring system of the deformations. In the article are present combination of surveying methods for observation of the deformations in mine "Ellatzite", the used methodologies, specialized software for processing and analysis of the data from the measurements.

1 GENERAL INFORMATION

The open pit mine "Ellatzite"is situated on 65 km east from the city of Sofia in the highest area of "Etropole Stara planina" mountain. The mine is 14 km south from the city of Etropole and North from the major crest of "Stara planina" mountain.

The pit wall has dynamic and constantly changing configuration. On Northeast and Southwest it resembles to an ellipse with approximate dimensions of 1750x1500m and total surface 2000 dka. The final design of the pit wall is done on 30 m high benches, created on two stages of 15 m. At the moment the actual depth of the mine in its Southern part is 540 m and in its next development is designed to reach 690 m deep with the interramp slope angles ranging from 37° in the upper lithologies to about 48° in the competent constrained ground toward the base of the pit. The dimensions of the mine "Chuquikamata" in Chili are similar – its depth is approximately 650 m, the interramp slope angle is 46° as well as the mine "Palabora" in South Africa is approximately 800 m deep with the interramp slope angles ranging from 37° in the upper lithologies to about 58°.

1.1 Geological Characterization

The area of the mine "Ellatzite"consists mainly of three types of rocks:

- Paleozoic metamorphic complex /phylite, schists, hornfelse and other /
- Paleozoic granodyorite and accompanied by phase with veins of dykes rocks
- Upper – creda dykes rocks going through both complexes mentioned above

Metamorphic rocks that are not contact changed are presented mainly by phylites, schists, clay-sericit schists quartz–chloride–sericit schists and others.

Contact changed schists are presented mainly by hornfelse schists, biotit–sericit schists, quartz biotit schists and more rarely by actinolit, tremolit, biotit and schists with spots.

Granodyorite are most spread in the deposit and built its central and north parts. They are leuco to mesocrate rocks and they are grey with pale rose nuance, medium to big grain size often light porpfyre in feldspate. There is a clear metasomatose expressed in change of potassium and silicate and late in sericite type.

The upper –creda dykes rocks in the area are present mainly by quartz -syenit –diorite porphyry and they are incorporated in the schists and in the granodyorites.

1.2 State of the Problem

The slope stability is of great importance for the normal project and operations development of the open pit mines.

The appearance of ground failures, especially on large areas, over several benches, could lead to serious consequences and in the most unfavorable cases to the loss of people's life and equipment.

The theory supported by the on-site experience postpones that the slopes must be absolutely stable if their angles are low-grade. That is why in practice it is imposed to work with controlled slope movements.

An important condition is the constant instrumental determination of the vectors of movement of pre-defined benchmarks, which are situated in the most sensitive area – where the rocks have low physic–mechanical properties, zones with unfavorable orientation of the cracks.

At the end of the summer of 2008 after big rainfalls on the territory of the mine is observed an activation of the landslides and deformation processes in the upper parts of the Southern area over the level 1330 of the mine. This area is built of weak types of rocks (phylite schists) with low indicators of robustness – the cohesion is under 100 kPa and the angle of internal friction is 35^{o}, which are dark grey to black containing in majority clay and organic substances.

The deformations are related with the appearance of the systems of cracks which have parallel orientation to the open pit slopes, with easily seen vertical displacements of the separate rocks blocks.

The created temporal surveying monitoring system proved displacement rates in the range from 1cm to 5 cm per day. To prevent the occurrence of major incident it was decided to undertake and discharge of the rock massif.

2 SURVEYING METHODS FOR MEASURING THE DEFORMATIONS

The idea of the used surveying methods is to determine the vectors of movement relative to points that stay stationary. For this purpose is created a system of points -"Observation station", which in the general case includes:

R1, R2, R3, – observed points (known as controllable benchmarks or just benchmarks) with measured horizontal and/or vertical movements. The benchmarks are single or situated on profile lines on places in the area that is suitable for this purpose;

St1, St2, St3 – stations from which is measured the displacement from reference marks R1, R2,

O1, O2, O3 – points for orientation; they serve as beginning ("zero") of the measurements from the stations St1, St2,

K1, K2, K3 – control points for checking the stationary of the stations St1, St2 …

Depending on the accepted measurement technology the "Observation station" can consist of points from different types. The minimal configuration contains benchmarks, one station and one point of orientation.

The diversity of the sites as well as the importance of the task on one part and the limited possibilities of the surveying instruments of the other, leaded to creation of a large number of technologies for determination of the deformations. Nowadays the development of the technical means gives us the possibility to measure the deformation of every point of the open pit mine including the waste dumps, using a limited number of methods: angle–linear network; polar; GPS-measurements; geometrical leveling and a combination of all.

In the measurement of the deformations it is often is used a local coordinate system with axes X or Y, with direction to the expected maximal displacements. The projections of the vector of displacement of the observed point are found from the measurements in the times t and t-1:

$$\left.\begin{aligned}\Delta X_{t,t-1} &= X_{t-1} - X_t,\\ \Delta Y_{t,t-1} &= Y_{t-1} - Y_t,\\ \Delta H_{t,t-1} &= H_{t-1} - H_t.\end{aligned}\right\}$$

From here you can calculate the horizontal vector of displacement:

$$\Delta S = \sqrt{\Delta X^2_{t,t-1} + \Delta Y^2_{t,t-1}}$$

And the direction and the grade of the movement:

$$\alpha_\Delta = \mathrm{arctg}\frac{\Delta Y_{t,t-1}}{\Delta X_{t,t-1}},$$

$$i_\Delta = \mathrm{arctg}\frac{\Delta H_{t,t-1}}{\Delta S}.$$

Rarely is used such called "full" vector of displacement:

$$\Delta S' = \sqrt{\Delta X^2_{t,t-1} + \Delta Y^2_{t,t-1} + \Delta H^2_{t,t-1}}.$$

3 METHODOLOGY OF THE MADE MEASUREMENTS

3.1 Used Surveying Methods and Instruments for Measuring the Deformations in the Mine "Ellatzite"

For the measurements related to the determination of the deformations in the mine "Ellatzite" was chosen the Polar method.

The 3D coordinates of the controlled benchmarks (situated on the benches) are determined from one base point that is stationary (the station) and 2÷4 orientation points. The coordinates and the level of the station are checked, using "resection" (angles and distances measurement) to the orientation points before each cycle of measurement.

Also in determined periods of time the orientation points are checked using accurate GPS – measurements from suitable pair of control points, situated far from the pit rim outside the supposed zones of deformation.

The polar measurements are done using robotized total station Topcon GPT -9000 A and reflection prisms. The mean quadratic error for angle measurements is ±0.0003 gon, and for length measurement ±2mm+2ppm. The instrument is equipped with fast servo motors and with technology for automatic targeting on maximal level of the reflected signal in the centre of the reflection prisms that helps to eliminate the error for targeting.

3.2 Accuracy of the Measurements

Although mentioned advantages of the advanced surveying instruments there are many factors that influence negatively the accuracy of the measurements. The accuracy of the angle measurements is getting worse due to the horizontal and the vertical refraction of the ray of sight, which are not very well studied in general. The changes of the atmosphere conditions influence negatively the accuracy of the distance measurements. To minimize this influence it is obligatory before each measurement to take the values of the air temperature and the atmosphere pressure and to upload them into the software of the instrument. The correction (ppm) of the measured distance is calculated automatically.

The practical results shows that the mean quadratic error when finding the coordinates of the observed benchmarks on a distance 1-1,2 km is approximately 10-15 mm which is a fully accepted accuracy having in mind that the eventual values of the critical deformations for one cycle are several times bigger.

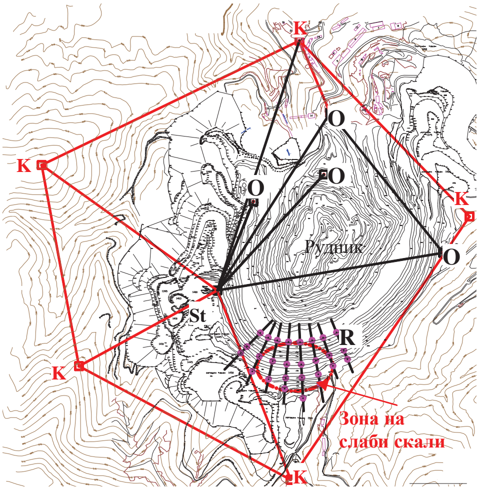
3.3 Scheme of the Observation Station

Taking in consideration the stage of the development of the mine now and in the future – at the end of the present year or in the beginning of 2010 is planned to initiate the development of a new project, which will start from the upper overburden levels of the mine where the benchmarks are placed. Having in mind that the actual benchmarks will be destroyed with the development of this project it will be necessary to look for the best possible solutions for the observation of the problematic area without interrupting the surveying process.

Following the best known world practice it will be necessary to have as minimum: 3 benchmarks on each of the profiles (the medium distance between the profiles is 80 - 90 m), which have to be situated in the lowest, medium and the highest part of the corresponding profile. In our case we will place the benchmarks on the following levels:

level 1270, level 1390 and level 1440, as well it will be necessary to put three benchmarks in the highest part of the area: level 1470 and level 1510. The number of the required benchmarks will be 27.

The positions of the stations from which is measured the displacement, the points for orientation, the control points and the benchmarks are shown on Figure 1.



K – control points, O – points for orientation, St –station, R – benchmarks

Figure 1. Position of the base network and the observation station.

Based on the accumulated experience from the measurements made so far – the registration of large displacements and velocities /by several cm per day in group of benchmarks, situated in the concerned area in the Southern pit wall/ we expect at the beginning to organize the measurements in a periods of 10 – 15 days. Depending on the size of the deformations /after doing several cycles of measurements/ the period between them can be shortened or prolonged.

4 DESIGN OF THE POINTS AND THE BENCHMARKS, NECESSARY CONTRUCTION ACTIVITIES

The design of the starting points, the control points and the points for orientation is shown on Figure 2. They are constructed from: concrete, 250 mm in diameter PVC tube with thick walls, and a device (metal plate) for forced centering of the total station. The plate must be made of stainless material.

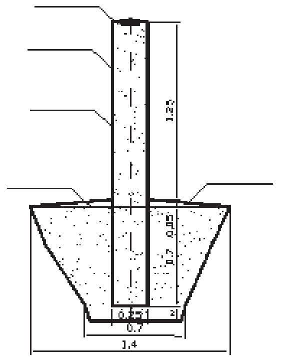


Figure 2. Construction of the pillar of the station, control points and points for orientation.

The construction of the pillars of the controlled benchmarks is shown on Figure 3. A 110 mm in diameter PVC tube with thick walls /2.5 meters in length/ is placed in a 1 meter deep concrete filled drill hole. In the centre of the tube concrete filled is put a ϕ 16 mm steel bar /2.5 meters in length/, its upper end is adapted for installing a surveying prism. The height of the benchmark over the surface is 1.5 m. The drill hole will be done using a drill type CUBEX QXR 1320, with diameter of 165mm. The depth of the drill hole has to be minimum 1 m.

It is necessary to install a suitable protection cabin that will allow making the measurements in all weather conditions, as well as in any directions.

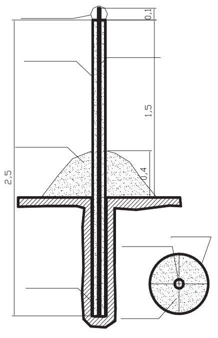


Figure 3. Construction of the pillars of the controlled benchmarks.

5 APPLIED PROGRAMS FOR PROCESSING OF SURVEYING OBSERVATIONS OF THE SLOPES OF THE OPEN PIT MINE "ELLATZITE"

The results of surveying are large data of a dynamic, one type information collected for long time. Special type of calculation is the comparison between benchmarks, changing in the time, between vectors and etc. By considering this type of work it was a need to make a packet of programs for calculation measures made.

On Figure 4 is presented the technological scheme of processing the data from periodical surveying measurements.

The data from the measurements goes in **Block A**. The real processing starts with an evaluation of the stationary of the station on which the instrument is centered.

1. The evaluation is done using a strict adjustment, using the method of smallest squares:

- for horizontal displacement using a program for processing an angle–linear network. In the program the problem related to the relation between linear and angle errors is solved by a strict and simple way
- For vertical displacement – using a program for processing of a trigonometric leveling; the program gives us the possibility to evaluate the specific coefficient of refraction for our conditions /usually it is pre-set for the country conditions/.

The next step is the calculation of the vectors of displacement (2). We use a universal program that processes the data from the different measurement systems and obtains the vectors: of direction (ΔH or ΔS), in plane (ΔX, ΔY) or in the space (ΔX, ΔY, ΔH).

The obtained results are written in a DB (data base) (3).The information is organized by profile lines, date of the observations and the numbers of the benchmarks.

An important element form the organization of the DB is the program for replacement, respectively "recovering" of a benchmark and the processing of omitted observations.

Block B includes the processing of the data from the informational DB.

Important parameters are determined with the processing of a separate profile line (4), such as deformations (of strain or stress), inclinations of some sections, etc.

With the processing of the group of benchmarks (5) we are following the behavior of one or more benchmarks in the time. The most important parameters are the velocity and the accumulated displacements.

The system is opened and allows other modules, for example: mathematic models for finding an eventual sliding surface.

The outgoing information is presented in table and in graphics format.

On Figure 1, as well as on Figures 5 and 6 are illustrated the initial results of the program for real measurements of the deformations in mine "Ellatzite".

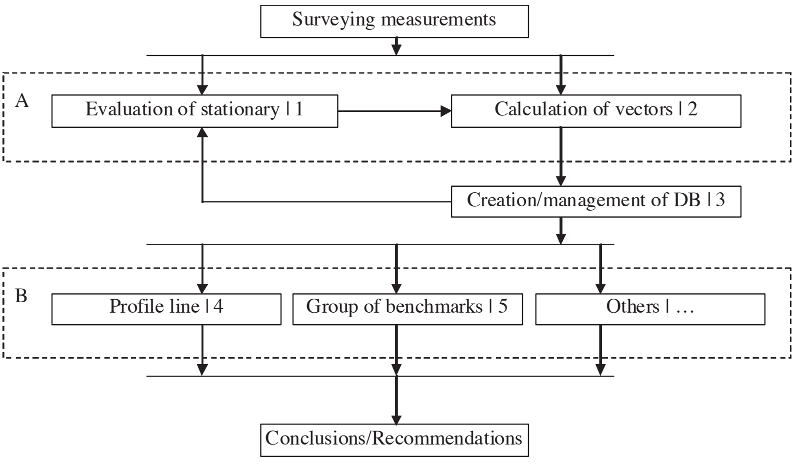


Table 1. Vertical and horizontal displacements and velocities.

Number	Date	Date	Horizontal deformation [mm]	Velocity [mm/d]	Indicated angle [gon]	Subside	Velocity [mm/d]	Inclination of the vector	Resultant of the deformations [mm]
361	24.10.2008 г.	11.11.2008 г.	162.2	9.01	391.6	-88	4.89	-0.5424	184.5
362	24.10.2008 г.	11.11.2008 г.	310.6	17.25	393.57	-144.3	8.02	-0.4647	342.5
363	24.10.2008 г.	11.11.2008 г.	235	13.06	0.36	-133.3	7.41	-0.5674	270.2
364	24.10.2008 г.	11.11.2008 г.	85.2	4.73	389.49	-36	2	-0.4227	92.5
365	24.10.2008 г.	11.11.2008 г.	11.1	0.62	223.51	1	0.06	0.0902	11.1
330	24.10.2008 г.	11.11.2008 г.	35.9	2	255.02	8.7	0.48	0.2412	36.9
331	24.10.2008 г.	11.11.2008 г.	55.3	3.07	382.92	52.3	2.91	0.9461	76.1
332	24.10.2008 г.	11.11.2008 г.	190.1	10.56	395.75	-53.7	2.98	-0.2823	197.5
333	24.10.2008 г.	11.11.2008 г.	176.6	9.81	385.95	-30.3	1.69	-0.1717	179.2

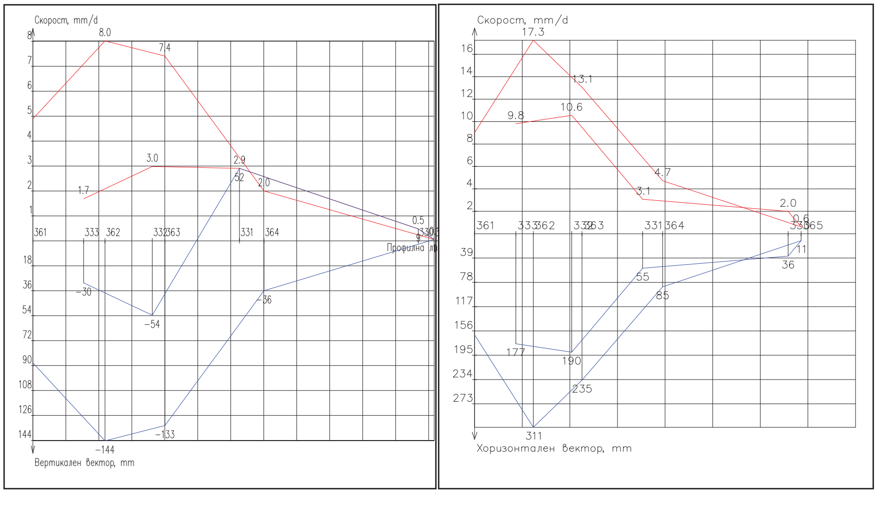
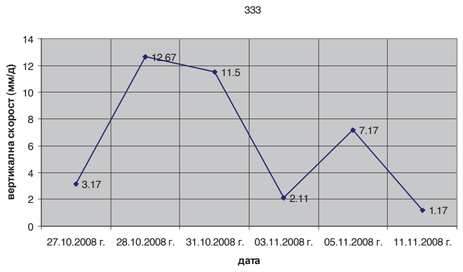


Figure 5. Vertical and horizantal displacements and velocities.



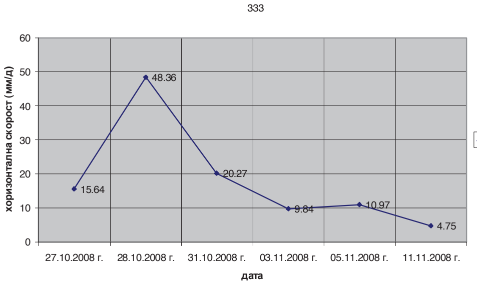


Figure 6. Velocity of the vertical and the horizontal displacements in the time of one benchmark.

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3rd Balkan Mining Congress, BALKANMINE 2009, 1-3 October, İzmir-TURKEY, © 2009, ISBN 978-9944-89-782-2

Evaluation Of Underground High Pressure Gas Storage Plants In Granite Rock

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ABSTRACT The concept of storing natural gas in underground geologic formations arose from the need to supply gas to consumers during periods of high seasonal demand and to protect gas supply against disruptions. The main principle of the Lined Rock Cavern (LRC) concept is to store gas at high pressures at relatively shallow depths (100-200m). The hard rock (granite etc.) absorbs the pressure load and the lining ensures gas tightness.

The objective of the paper is to evaluate high pressure gas storage rock caverns in massive granite rock with regard to different operating conditions. In this context, firstly, underground gas storage applications were summarized. Then, information was given on rock caverns plants for underground natural gas storage purposes. Finally, numerical analyses of high pressure gas storage caverns in granite rock were evaluated.

1 INTRODUCTION

1.1 Underground Gas Storage Types

The concept of storing natural gas in underground geologic formations arose from the need to supply gas to consumers during periods of high seasonal demand and to protect gas supply against disruptions. An underground gas storage (UGS) facility is capable of injecting and withdrawing gas. Depleted gas/oil reservoirs, aquifers, and solution mined salt caverns are the three main types of underground natural gas storage in use today. Gas storage in underground rock caverns will become an alternative storage method, especially in regions where depleted reservoirs, aquifers, and salt deposits are not available.

1.2 Storage Type Characteristics

Each storage type has its own physical characteristics and economics (site preparation, maintenance costs, working gas capacity, deliverability rates, cycling capability etc.), which govern its suitability to particular applications. Two of the most important characteristics of an underground storage reservoir are its capacity to hold natural gas for future use and the rate at which gas inventory can be withdrawn–its deliverability rate. Storage capacity is defined as total ability of a storage facility to provide working gas volume, deliverability and injection rate. Deliverability is most often expressed as a measure of the amount of gas that can be withdrawn from a storage facility on a daily basis. Injection capacity is the complement of the deliverability rate, it is the amount of gas that can be injected into a storage facility on a daily basis. Working gas volume is defined as the volume of gas in a storage above the required level of cushion gas volume, which can be withdrawn/injected with subsurface and surface facilities subject to legal and technical limitations (pressure, flow rate etc.). Depending on storage type and local site conditions the working gas volume may be cycled more than once a year (IGU 2006).

Depleted reservoirs and aquifers are characterized by large storage capacities and high cushion gas requirements. The reservoirs are typically cycled once annually and are used to meet base load demand. Salt caverns provide very high withdrawal and injection rates relative to their working gas capacity. Cushion gas requirements are relatively low. Salt caverns for gas storage purposes have the ability to perform several withdrawal and injection cycles each year. Salt caverns are typically used to meet peak load demands. Lined rock caverns (LRC) have also the ability to provide very high working gas withdrawal and injection rates with very low cushion gas requirements. Working gas can be cycled many times a year.

1.3 Underground Gas Storage in Turkey

Natural gas consumption as a primary energy source has increased in Turkey. Currently underground gas storage (UGS) plants in two depleted gas fields (Silivri and Degirmenköy) are in operation with a total gas storage capacity of 1.6 billion m^3. A second underground gas storage plant under construction has been developed for the purpose of utilizing salt domes 40 km south of Tuz Golu (Salt Lake) in Sultanhani province. In the scope of Tuz Golu UGS Project, twelve underground gas storage caverns will be build by solution mining controlled leaching of the extensive underground rock salt formation. Each cavern will be developed at the depths of 1.100-1.400 m, which have a geometrical volume of approximately 630.000 m^3. The total working gas volume will be about 1 billion m^3 in twelve caverns (Özarslan et al. 2007, BOTAS 2009).

Gas storage projects in rock caverns may become attractive as the demand for natural gas peak load capacity will grow. Locations of igneous and metamorphic hard rock (granite, gneiss) outcrops for potential rock cavern gas storage sites in Turkey are given in Figure 1.

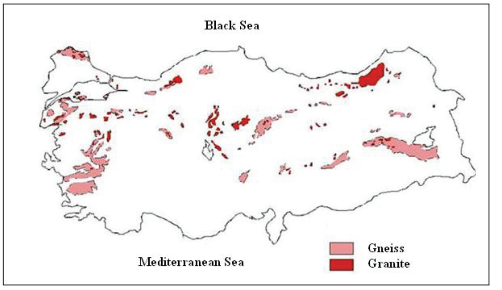


Figure 1. Locations of hard rock outcrops in Turkey (MTA 2008).

2 GAS STORAGE IN ROCK CAVERNS

In underground rock caverns two methods are available to store natural gas safely in rock mass structure. The unlined rock cavern method use groundwater to maintain gas-tightness according to water sealing method, and the lined rock cavern (LRC) method keeps the cavern gas-tight by providing lining on the cavern surface.

2.1 Unlined Rock Caverns

Unlined rock caverns have been used for decades to store a wide range of low vapor pressure products such as crude oil. Since the host rock is never completely impervious, the cavern must be sited deep enough to make sure that the water hydraulic pressure in the fractures of the rock around the cavern is always higher than the pressure of the product stored in the cavern. Sometimes the water pressure is maintained by injecting water into the rock through horizontal water holes above the cavern (water curtain) in order to maintain a stable groundwater level (DOE 1999).

2.2 Lined Rock Caverns

The lined rock cavern (LRC) concept is a technology for underground storage of natural gas at high pressure at relatively shallow depths (100-200 m). The main idea is to store the compressed gas in rock caverns lined with a thin impermeable liner and to let the surrounding rock mass carry the load. A typical LRC storage plant is composed of two main parts, the below ground facility with gas storage caverns and the above ground facility for handling of the gas (Fig. 2). The caverns are connected to the above ground facility via pipes placed in vertical shafts. The below ground facility consists of one or more storage caverns, a vertical shaft for each cavern, and a system of tunnels connecting the caverns with the ground surface (Johansson 2003).

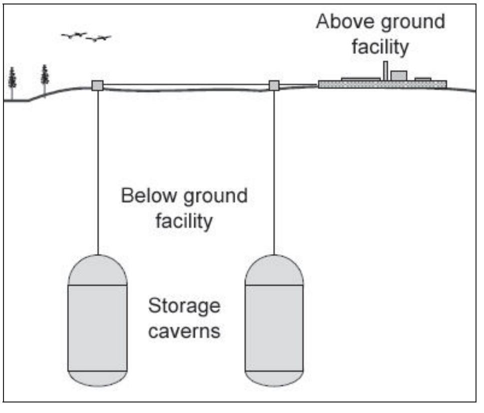


Figure 2. LRC underground gas storage concept (Johansson 2003).

The main principle of an LRC gas storage is that the load from the high gas pressure inside the cavern is taken by the surrounding rock mass. The lining is only to make the cavern absolutely gas tight and carry only a negligible part of the gas pressure. Another characteristic of the LRC concept is that the gas pressure is higher than the in situ rock stress magnitudes, which implies that a jointed rock mass, in case of a gas leak, would be permeable and could not hinder gas escape (Johansson 2003).

The LRC concept is based on the interaction of some main elements, as shown in Figure 3. The role of the steel liner is to achieve gas tightness. It is also able to bridge minor cracks in the concrete. A sliding layer is generally placed between the steel lining and the concrete to reduce the friction and to supply corrosion protection to the steel. The concrete layer is an intermediate layer and is located between the steel liner and the rock. Its main purpose is to transmit the gas pressure in the cavern to the rock and to distribute the deformations uniformly. At the same time, it serves as a smooth base for the steel liner. A welded mesh reinforcement is normally placed in the concrete lining to distribute the tangential strain into many small concrete cracks. A layer of special low strength permeable shotcrete is placed closest to the rock surface. The purpose of the shotcrete is to protect the drainage system, to improve the hydraulic contact to the drainage system and to reduce the interlocking between the concrete lining and the rock surface. The drainage system consists of perforated drainpipes. It allows the water pressure against the steel liner to be lowered when the rock cavern is under construction or the internal pressure in the cavern is low, thus avoiding the risk of liner buckling (DOE 1999, Johansson 2003).

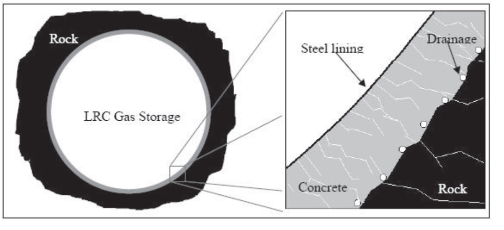


Figure 3. LRC wall design elements (DOE 1999).

The surrounding rock mass support the gas pressure load on the wall, thus acting as a pressure absorber. From the structural stability, the lining is not designed to carry primary loads, as the concrete layer supports the lining, which in turn, is supported by the surrounding rock mass. The steel lining should, however, be able to withstand the stress and strain caused by the deformations of the rock cavern wall (DOE 1999).

The maximum operating gas pressure allowed for the storage depends on the lining characteristics and the geological conditions. It must be set at a value such that the induced deformation in the rock mass and the concrete layer does not result in the strain of the steel lining in excess of its capacity for deformation. The quality of the steel lining is of primary importance for a successful LRC. To make an LRC storage facility economically feasible, the storage pressure must be relatively high (DOE 1999). The LRC concept has been first developed in Sweden (Marion & Mansson 2003, Glamheden and Curtis 2006). Operational and pilot LRC gas storage plants are summarized in Table 1. Typical specifications and dimensions of a commercial LRC gas storage plant are given in Table 2.

Table 1. LRC gas storage plants.

Storage Plant	Country	Plant Type
Skallen	Sweden	Commercial
Grangesberg	Sweden	Pilot
Kamiako	Japan	Pilot

Table 2. Commercial LRC gas storage plant characteristics (LRC 2008).

Total gas volume	20-25 MNm^3
Diameter	40 m
Height	100 m
Max. Gas Pressure	23 MPa (230 bar)
Rock cover	130 m
Injection	20 days
Withdrawal	10 days

The increase of the gas pressure will cause an expansion of the rock cavern due to deformation of the surrounding rock mass. The cavern wall will follow the rock as it expands and the steel lining will be strained. The cavern wall will be subjected to both compressive forces directed perpendicular to the wall, and tensile forces directed tangential to the wall. The resistance of the wall to the tangential forces is very limited. The concrete lining will crack but be held together by the reinforcement. The compressive forces acting on the concrete lining are supported by the rock mass (Johansson 2003).

The most important properties of the rock mass are the modulus of deformation and the compressive strength. For the concrete used in the lining, the important limit is the compressive strength that should be above the maximum gas pressure. The tensile strength of the concrete does not impose any demand, since there is no way to hinder cracking of the concrete considering the general expansion of the cavern. The steel lining consists of steel plates and welds between them. The main properties for steel are the modulus of elasticity, the yield limit, the ultimate strain capacity and the fatigue properties. The fatigue load, influenced by the strain range and the number of cycles, must be below the fatigue capacity of the steel liner (Johannson 2003).

The maximum cavern gas pressure in a LRC storage plant can be expected to be in the range of 15-30 MPa. Minimum gas pressure during normal operation depends on the groundwater pressure surrounding the cavern, the pipeline pressure and is probably in the order of 2-4 MPa. The number of pressure cycles will generally be 10-20 per year. Thus, a plant life of 50 years mean that the total number of load cycles is expected to be 500-1000. The temperature in the cavern will fall during a withdrawal. The temperature variations inside the cavern can be reduced through the use of gas circulation heating/cooling systems (DOE 1999, Johansson 2003). With the circulation system, gas inside the cavern will be cooled during the injection and heated during the withdrawal to improve the working gas amount and to lower the temperature variations inside the cavern (DOE 1999, LRC 2008).

2.3 Evaluation of LRC Storage Sites

Regarding regional geology many potential LRC sites exist in Turkey and Balkan region. Pipeline access and ease of permitting will also play important roles in site suitability analyses.

For the first evaluation of potential LRC storage sites widely used rock mass classification systems (RMR, Q, GSI, RMi) are valuable. As the rock mass rating often is known from other underground excavations in many candidate areas, a first estimate of the suitability of the intended location may be possible without any significant site investigations. Classification systems can also be used for the estimation of the local rock mass properties. They are all more or less based on the rock mass classification systems (Johansson 2003).

Failure of underground openings in hard rocks is generally a function of the in situ stress magnitudes and the characteristics of the rock mass, i.e. the intact rock strength and the fracture network. At low in situ stress magnitudes, the failure process is controlled by the continuity and distribution of the natural fractures in the rock mass (Hoek et al. 1995, Martin et al. 1999). Low in situ stress is defined as the ratio of the maximum far-field stress (Po) to the unconfined compressive strength (σ_c) lower than 0.15. Schematic illustration of opening instability as a function of RMR at low in situ stress levels is presented in Figure 4. Massive unweathered hard rock mass (granite, gneiss etc.) conditions (RMR>75) are favorable rock mass conditions for underground LRC gas storage plants.

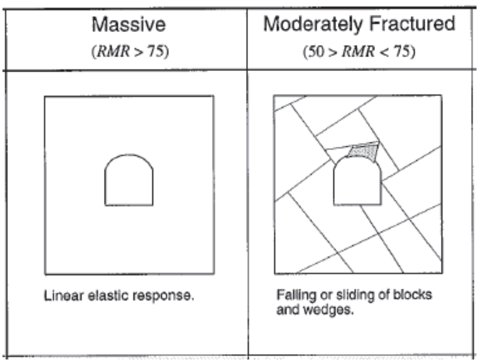


Figure 4. Opening stability at low in situ stress levels (Martin et al. 1999).

Equation 1 can be used to estimate the value of GSI from Bieniawski's RMR_{89} classification (Bieniawski 1989, Hoek et al. 1995, Hoek & Diederichs 2006). In this case a value of 15 is assigned to the completely dry groundwater rating and the adjustment for joint orientation is set to favorable.

$$GSI = RMR_{89}-5 \quad \text{or} \quad RMR_{89} = GSI + 5 \quad (1)$$

3 ESTIMATION OF GRANITE ROCK PROPERTIES

For most hard igneous and metamorphic rocks failure can be classified as brittle, which implies a sudden reduction in strength when a limiting stress level is exceeded. Intact and massive granite rock show brittle strength characteristics, elastic and generally isotropic behaviour (Edelbro 2003).

Important rock properties in relation to the LRC concept are the modulus of deformation (Young's modulus E) and uniaxial compressive strength (UCS) of the investigated rock. Theoretically, the best way of assessing these properties would be to perform in-situ tests in a scale relevant for the problem, involving a sufficiently large volume of the rock mass. Such large tests are, if not impossible, very expensive and complicated. Instead, rock mass classification and empirical relationships can be used for the estimation of the relevant rock mass properties.

Rock material properties (E, UCS, etc.) can be determined by rock mechanics laboratory tests suggested by ISRM and ASTM. Rock material properties of a typical granite rock were determined from samples taken from a granite mine in Yaylak, Turkey. Laboratory experiments were conducted on core specimens of NX size, 54 mm. The experiments were carried out in accordance with the methods suggested by ISRM to determine the physical and mechanical properties of granite rock, including unit weight (γ), uniaxial compressive strength (σ_{ci}) and Brazilian tensile strength (σ_t). Average values of physical and mechanical properties of intact Yaylak granite are given in Table 3.

Table 3. Physical and mechanical properties of Yaylak granite rock material.

Unit weight	σ_{ci}	σ_t
26 kN/m^3	148.3 MPa	11.8 MPa

Using the modulus ratio MR proposed by Deere (1968) and modified by Palmström and Singh (2001), it is possible to estimate the intact modulus from Equation 2. This relationship is useful when no direct values of the intact modulus (E_i) are available (Hoek & Diederichs 2006). Calculated deformation modules for granite rock material are given in Table 4. In this study, the average value of 63 GPa was used for further calculations.

$$E_i = MR \times \sigma_{ci} \quad (2)$$

Table 4. Calculated deformation modules for granite rock material.

MR*	σ_{ci} (MPa)	E_i (GPa)
300 (min)	148.3	44.5
425 (aver.)	148.3	**63.0**
550 (max)	148.3	81.5

* proposed values for granite rock

Typical ranges of Poisson's ratios for various rock types have been published by Gercek (2007). Poisson's ratio of granite rocks are generally in a range of 0.1-0.35. In this study, Poisson ratio of massive granite rock was accepted as 0.25.

The type of granite rock mass considered in this study is massive in situ rock with few widely spaced discontinuities. Lower RMR value of massive granite rock was taken as 75. The calculated GSI value (Eq. 1) for massive granite is 70.

Marinos and Hoek (2000) proposed common GSI ranges for typical granite rock mass conditions between 90 and 50. The suggested GSI values range for massive granite rock conditions are between 70 and 90. The low limit GSI value of massive granite rock with good surface conditions (very rough, fresh unweathered surfaces) can be accepted as 70. GSI values close to 100 correspond to excellent quality rock masses, for which the rock mass strength is equal to the intact rock strength.

The in situ deformation modulus of a rock mass is an important parameter in numerical analyses and in the interpretation of monitored deformation around underground openings. Since this parameter is very difficult and expensive to determine in the field, several attempts have been made to develop methods for estimating its value. Several authors have proposed empirical relationships for estimating the value of an isotropic rock mass deformation modulus on the basis of classification schemes such as the Rock Mass Rating (RMR), the Tunnelling Quality Index (Q) and the Geological Strength Index (GSI). Hoek and Diederichs (2006) stated that while most of these equations give reasonable fits to the field data, all of the exponential equations give poor estimates of the deformation modulus for massive rock. In this study, deformation modulus of massive granite rock was determined by the empirical relationship proposed by Hoek and Diederichs (2006).

$$E_m = E_i (0.2 + \frac{1\text{-}D/2}{1+e^{((60+15D-GSI)/11}} \quad (3)$$

Using average MR value, uniaxial compressive strength value (σ_{ci}) and GSI value of 70 (RMR 75) the rock mass deformation modulus (E_m) of massive granite was calculated (Tab. 5).

Table 5. Rock mass deformation modulus of massive granite rock.

GSI	D*	E_i	E_m
70	0	63.0 GPa	44.9 GPa

*D=0 undisturbed conditions

The compressive strength of a rock mass is difficult to estimate (Edelbro 2003). The estimation of the rock mass strength can be done by empirical relationships, mainly based on σ_{ci}, and RMR, GSI value parameters. A review of widely used equations was published by Genis et al. 2007. Calculated rock mass strengths σ_{cm} for a massive granite rock with RMR rating value of 75 or GSI 70 are given in Table 6. The estimated strength values represents lower limits of a massive rock mass. Palmström (2000) recommends a uniaxial compressive strength of rock mass (σ_{cm}) for massive rock nearly equal to half of the intact rock strength (σ_{ci}), taken scale effect into account (Eq. 4). The tensile strength of discontinuities and rock masses is generally assumed to be zero.

$$\sigma_{cm} \approx 0.5\ \sigma_{ci} \ \text{(massive rock)} \quad (4)$$

Table 6. Rock mass strength estimation for massive granite rock.

Researcher(s)	Empirical Relationship	σ_{cm}*
Hoek & Brown (1980)	$\sigma_{cm}=\sigma_{ci}\sqrt{e^{\frac{(RMR-100)}{9}}}$ (MPa)	37,0
Ramamurthy (1986)	$\sigma_{cm}=\sigma_{ci}e^{\frac{(RMR-100)}{18.75}}$ (MPa)	39,1
Sheorey (1997)	$\sigma_{cm}=\sigma_{ci}e^{\frac{(RMR-100)}{20}}$ (MPa)	42,5
Trueman (1998)	$\sigma_{cm}=0.5e^{0.06RMR}$ (MPa)	45,0
	Average (MPa)	40,9

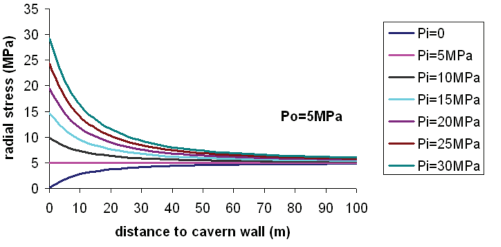
*σ_{ci} =148,3 MPa, 70 GSI = 75 RMR

In the analyses the calculated average σ_{cm} value 41 MPa (Tab. 6) was accepted as minimum and the σ_{cm} value 74 MPa calculated from Equation 4 as the maximum uniaxial compressive strength of the examined massive granite rock mass.

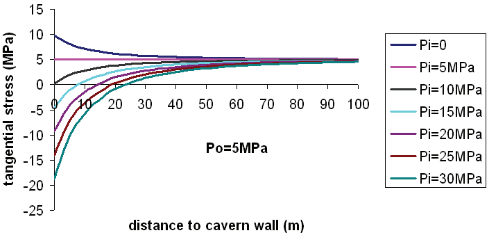
4 NUMERICAL ANALYSES

The type of granite rock mass considered in this study was selected as massive. In massive rock conditions at low in-situ stress levels the medium can be considered to behave as an isotropic elastic continuum. PHASE2D finite element program (Rocscience 2007) was used for numerical elastic analyses of the investigated underground openings. Elastic deformation modulus (E_m) 44.9 GPa and Poisson ratio (ν) 0.25 was used as elastic input parameters for massive granite rock. For assessment of stress induced instabilities analysis results were compared with strength values for massive granite rock.

Firstly, the problem of stresses and deformations around a representative underground cylindrical or spherical opening, with 40 m diameter, was investigated. The openings are assumed to be subjected to varying internal pressure (Pi) and a far-field constant hydrostatic stress (Po) of 5 MPa. The following examples (Figs. 5-6) are intended to illustrate the effect of the internal gas pressure, on elastic stresses around the openings. With respect to the hydrostatic far-field stress, the tangential stresses are equal in the case of spherical openings while, in the case of cylindrical openings, the out-of plane stress, is an intermediate principal stress.

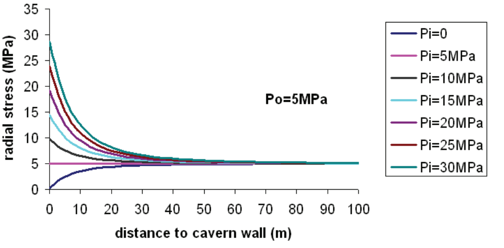


a- radial stress

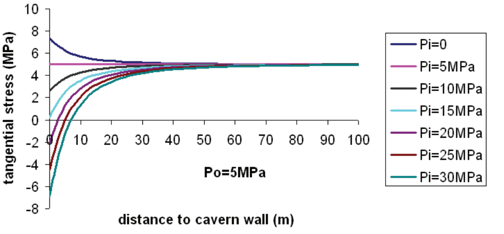


b- tangential stress

Figure 5. Stress distribution of a cylindrical opening.



a- radial stress



b- tangential stress

Figure 6. Stress distribution of a sphere opening.

As expected, the resulting radial and tangential stresses showed lower values for sphere openings than cylindrical openings. After a critical internal pressure value (Pi>2Po for cylindrical and Pi>3Po for spherical openings), tensile (-) tangential stresses occurred around the openings. When increasing the internal gas pressure (i.e. Pi>Po) the tangential direction converts to minimum principal stress direction and the radial direction converts to maximum principal stress direction. At high gas pressure conditions, the internal pressure is the major principal stress at the rock cavern surface wall. Also, for the considered massive granite rock, elastic displacements depends on the internal gas pressure. Comparison of radial displacements (Ur) of a sphere and cylindrical opening at the cavern wall for various internal pressure conditions is given in Table 7. Elastic displacement values are nearly half for a sphere opening at same conditions. Negative (-) values represents inward closure and positive (+) outward extension displacements of the rock cavern wall. After a critical internal pressure of Pi>Po outward displacements occur around the openings, which represents a special rock mechanics case. Due to the symmetry of the problem, the tangential components of displacements are everywhere zero and the radial components at the cavern wall represents maximum displacements.

Table 7. Comparison of max. radial displacements at the cavern wall.

Gas pressure Pi (MPa)	Ur (mm) Sphere	Ur (mm) Cylinder
0	-1.332	-2.616
5	0	0
10	+1.332	+2.616
15	+2.664	+5.233
20	+3.996	+7.850
25	+5.328	+10.466
30	+6.661	+13.083

D=40m, Po=5 MPa, E_m=44.9 GPa, ν=0.25

PHASE2D finite element program was also used for numerical axisymmetric analyses of a typical underground gas storage rock cavern subjected to varying internal pressure loads. Axisymmetric modeling allows to analyze a 3-D excavation which is rotationally symmetric about an axis. The input is 2-dimensional, but the analysis results apply to the 3-dimensional problem. Because of the symmetry, only half of the problem needs to be defined. The representative 3-D dimensional opening shape was selected as a half-sphere roof with cylindrical side wall and bottom with rounded corners. The diameter of the opening was selected as 40 m and total height as 80 m, similar to typical LRC dimensions. In the analyses no blasting and lining effects were considered. The rock surface is assumed to be smooth for uniform distribution of internal gas pressure. Detailed analyses for various opening shapes, far-field stress, gas and rock mass conditions were carried out by Tascakmak (2008).

In the study, isotropic (hydrostatic) in situ far-field constant stress conditions of 5 MPa was considered according to an average depth of 200 m from surface to cavern bottom. The axisymmetric finite mesh is given in Figure 7, which also shows the boundary conditions. The rock cavern opening is modeled in the middle of a rectangular domain with sides about 10 times the opening width.

Example distribution of outward displacements for an internal gas pressure situation of 25 MPa is given in Figure 8. Large displacements occurred at the bottom and side wall of the cavern. Maximum displacement values (Ur) at the middle point of the symmetric cavern bottom are given for various internal gas pressure conditions in Table 8.

Zero displacement occurs when the internal pressure equals the hydrostatic stress, a maximum inward displacement when the pressure equals zero and a maximum outward displacement when the max. internal gas pressure is 30 MPa. Inward and outward displacements resulting from internal gas pressure can cause instability of the lining if the pressure is inadequate or to high.

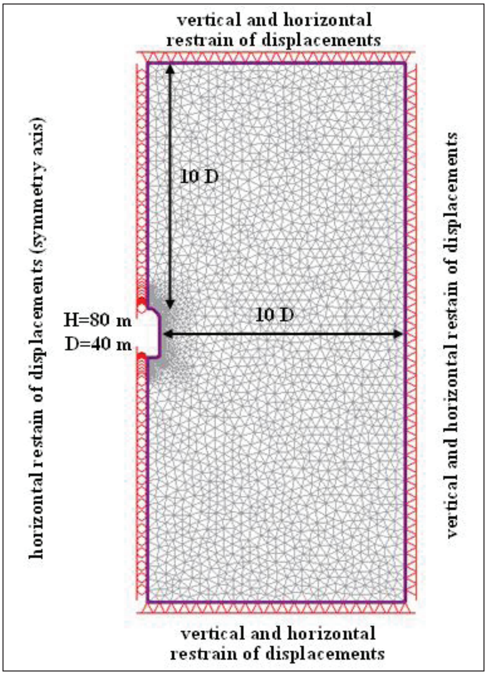


Figure 7. Axisymmetric finite element mesh and boundary conditions.

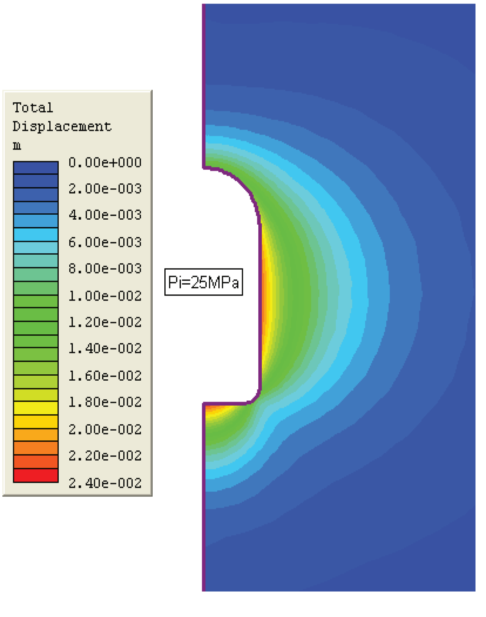


Figure 8. Example distribution of outward displacements for Pi 25 MPa.

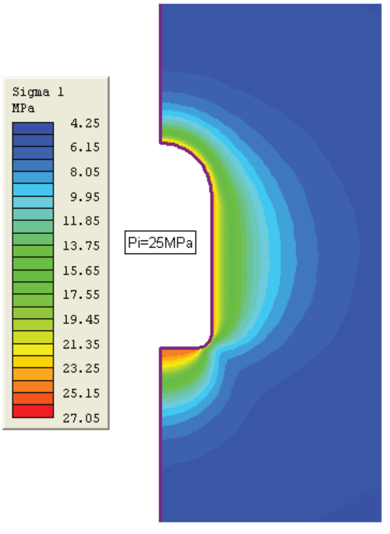
Table 8. Max. radial displacements at the cavern bottom.

Pi (MPa)	Ur (mm)	displ. direction
0	-2.32	inward (-)
5	0	no displac.
10	+2.32	outward (+)
15	+4.64	outward (+)
20	+6.96	outward (+)
25	+9.28	outward (+)
30	+11.60	outward (+)

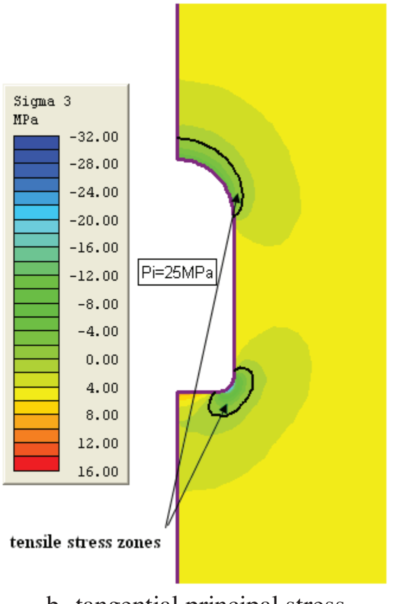
Po=5 MPa, E_m=44.9 GPa, ν=0.25

When internal pressure is applied to an underground storage plant the stresses in the vicinity of the opening are re-distributed. Distribution of radial, tangential and circumferential stress values are given in Figure 9. The example analysis results represent internal gas pressure (Pi) of 25 MPa and massive granite rock conditions. This figure shows that the redistribution of stresses is concentrated in the rock very close to the opening. At high internal pressure conditions maximum compressive radial stresses surrounding the cavern are equal to active gas pressure values. Also, under high gas pressure conditions (i.e. Pi>15 MPa) tangential and circumferential tensile stress zones occur around the opening (Tascakmak 2008). High stress concentrations occur especially at distances close to the cavern wall.

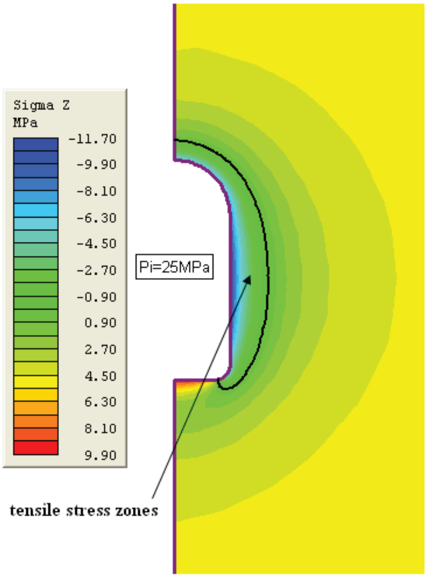
An important factor for stability assessment of gas storage rock cavern is the compressive rock mass strength. For massive granite rock nearly elastic conditions will prevail until the strength of the massive rock mass is exceeded. At the cavern surrounding wall rock failure occurs when gas pressure load exceeds the strength of the rock mass. For high gas pressure conditions, the internal pressure is the major principal stress at the rock cavern surface wall. The maximum principal stresses around the cavern are lower than the estimated compressive strength values of the considered massive granite rock. Tensile minimum principal stresses surrounding the cavern may be higher than intact tensile strength of granite rock. If discontinuities exist in the rock surface tangential tensile stresses will cause opening of radial joints. Tensile failure of the rock mass surrounding the opening will not significantly affect the opening stability since massive granite rock has considerably high compressive strength.



a- radial principal stress



b- tangential principal stress



c- circumferential principal stress

Figure 9. Example distribution of radial, tangential and circumferential stresses.

Fundamental functions of the maximum and minimum internal gas pressure values are to control the inward and outward rock displacements to prevent hazardous strain and stress in the steel and concrete lining of the storage cavern. Therefore, it is important to estimate deformations values for various operational conditions (gas pressure, insitu stress, rock properties etc.) around rock caverns for gas storage purposes.

5 CONCLUSIONS

In this study, an evaluation of underground high pressure gas storage plants in hard rock mass conditions was attempted. The paper deals mainly with the concept, design and conditions of lined rock caverns used for the storage of gas under high pressures. Numerical modeling was also utilized to evaluate the cavern rock response under various gas pressure conditions. Typical rock properties for numerical analyses were obtained from laboratory tests, empirical relationships and rock mass classification systems. Based on the results of numerical analyses for the proposed cavern geometry and with considered massive granite rock properties, it can be concluded that stability and safety of a large silo shaped gas storage cavern can be achieved with only light rock support when the rock mass is of good quality. Gas storage in underground rock caverns may become an alternative storage method, especially in regions where depleted reservoirs, aquifers, and salt deposits are not available. A number of issues is still needed to be clarified before widespread industrial utilization of the storage method. These issues include detailed assessments of the technical, economical and legal feasibility.

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Application of the Umbrella Arch Method in İstanbul (Turkey) Metro Project

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ABSTRACT The subway tunnels of İstanbul Metro Project are driven under the urban areas of the city. The risk of damage on the existing surface and in the underground structures due to the deformations in the tunnel may arise in this situation. Although, major parts of İstanbul Metro tunnels have been excavated with New Austrian Tunnelling Method (NATM), it had shown a bad performance in low or non-cohesive soils. The Umbrella Arch (UA) method was started to be used firstly in Güngören Formation having a dominant lithology of clay and then in Thrace Formation including intercalated sandstone, claystone and shale. The results of UA application become preferable especially in the grounds having high deformation risk and it also causes UA to be thought as an alternative for other grounds having same features. In this study, the application of the UA method was investigated in İstanbul Metro tunnels regarding to the behaviour of ground and also controlled if UA arrives aimed target or not.

1 INTRODUCTION

Depending on the developments of Science and Technology, there have been important improvements in the conventional tunnelling methods. Especially, for the poor ground conditions, a certain number of new techniques named as pre-vault, pre-lining techniques have been developed. These techniques are based on forming a pre-supporting cover to front part of upper side of tunnel before excavating on the face of tunnel and then going forward by excavating underside of the cover. Pre-supporting cover could be made by using umbrella pipes, jet-grouting columns or lined pre-arch. These techniques have been used firstly in Paris Metro (Denek, 2003).

The subway tunnels of İstanbul Metro Project are driven under excess urbanization areas of the city. This may cause severe risks of damage on existing surface and underground structures. Major part of İstanbul Metro Tunnels has been excavated with (NATM) New Austrian Tunnelling Method. In the method, the deformations formed after excavations are carried on both main rock and support elements. However, this method does not have a good performance in low or non-cohesive soils (Kurt et al., 2002).

In some parts of the tunnels, appreciable increases in tunnel and surface deformations were recorded during the excavations. Consequently, the Umbrella Arch (UA) method developed in Italy was brought to application in İstanbul metro tunnels. The UA method was applied firstly in Güngören Formation having a dominant lithology of clay and secondly in Thrace Formation including intercalation sandstone, claystone and shale (Kurt et al., 2002).

Applying of UA gives good results especially in the grounds having high deformation risk and also it is taken into consideration as an alternative for other grounds having same features. In this study, the application of UA is examined in İstanbul Metro tunnels and the results are evaluated regarding to the ground movements.

2 İSTANBUL METRO PROJECT

The gradually growing population, increasing migration and the necessity of transportation increase the necessity of shuttle transportation. It is aimed to solve this problem by the metro project that is presently being constructed (Vardar, 2002).

İstanbul Metro has a complex structure having deep excavations, transportation tubes, ventilation-energy-maintenance rooms and large underground rock structures having various sizes and shapes (Vardar, 2002).

Initially, İstanbul Metro was projected as a line along 4.Levent, Şişli, Taksim, Şişhane and Topkapı that have 16.3 km length with 13 stations. However, during excavation, some modifications were made in the Project due to the changes in residential-social development of the city (Denek, 2003).

Today, the total length of 4.Levent-Taksim tunnels that have been joined in the transportation system including 2 tubes, 6 stations, cut and cover structures is 17000 m. The stations are placed in Osmanbey, Şişli, Gayrettepe, Levent and 4.Levent. The Gayrettepe and Levent Stations are cut and cover structures while Taksim, Osmanbey, Şişli, 4.Levent are platform tunnels (Köksal, 1999).

The second step of the Metro Project, named as Yenikapı-Taksim Metro Line, is under construction with two parts, the Taksim-Unkapanı (3922 m) and the Yenikapı-Unkapanı (6511 m) lines. These tunnel constructions were planned as two lines and have average depth of 14-42 m. Besides, the construction of Yenikapı-Aksaray light rail route construction having 721 m length that connects light rail system to Yenikapı station is also continuing in the scope of this tender (Denek, 2003).

2.1 Application of the Umbrella Arch Method in İstanbul Metro Tunnels

Pre-arching techniques and face bolting application have been started to apply firstly in San Vitale Tunnel having 4,2 km length that is on Caserta-Foggia railway route in Italy with the name of Umbrella Arch, Umbrella Vault (Leca et al., 2003).

This tunnel having 150 m. depth was excavated in poor ground conditions containing silty clay, clay-marn, limestone and sand (Vardar, 2002).

In Turkey, Umbrella Arch Method is applied firstly in Yenikapı Tunnels that is a part of İstanbul Metro Project's second step, because of the occurance of the high deformation risks in some parts of the tunnels that are excavated by using New Austrian Tunnelling Method and taken good results. Depending on the good results taken in Yenikapı Tunnel, the method is also applied in Taksim tunnels and tunnel part under France Palace successfully (Kurt et al., 2002).

By means of Umbrella Arch Method, in the tunnels excavated in poor ground conditions where NATM could not be applied successfully, the tunnel performance data (tunnel and surface safety, productivity, excavating rate etc.) could be carried to a higher level (Kurt et al., 2002).

The Method of which its' sufficiency has been proved in many different projects, is also applied in many Tunnel Projects in Japan (Kurt et al., 2002).

The Method's most important disadvantage with respect to the NATM is that it has lower excavation rate and higher cost price. A part of supporting elements used in Umbrella Method is the same with Classical Method supporting elements (Rock Bolting, Wire mash, steel ribs, shotcrete etc.). However, there can be some different points in application, for instance; Rock bolts application. In the Classical method, Rock bolts are applied homogeneously between the steel ribs along the tunnel arch, but in the Umbrella Method, rock bolts could not be applied to the part of arch that contains Umbrella pipes. Another example is steel ribs application. In the classical method, all steel ribs have the same size for the same type of tunnels, but in the Umbrella method, Steel ribs' sizes are systematically changed according to the consideration of the project in one Umbrella Arch excavation period (Denek, 2003).

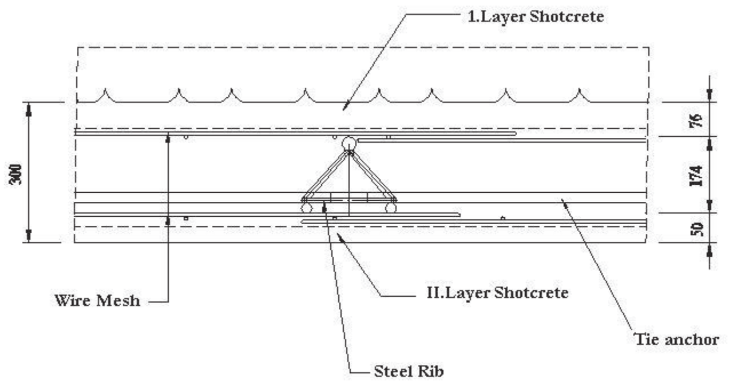


Figure 1. The details of excavation, steel rib, wire mesh and shotcrete.

2.1.1 Excavation

In the Umbrella Method, at the start, in order to be able to place umbrella pipes, widening excavations are made. By these excavations, it is aimed to widen 50 cm. outside of tunnel's cross-section in 3 m tunnel advance. In the poor ground conditions, excavations are made in three steps. In the rock media it could be made in two steps (Denek, 2003).

In Umbrella Method, all excavations are restricted with one Umbrella period. In Istanbul Metro Project, one Umbrella period is 6m. Excavations are made in three steps each one 0, 60 m by taking into consideration the widening. The widening in each advance steps are 5cm on the upper side 3 cm at the zero altitude towards to the sides. These widening at the end of the Umbrella period reach 50 cm on the upper side and 30cm at the zero altitude (Kurt et al., 2002).

In Umbrella technique, excavations are made by hydraulic impact hammers as in NATM and mucking after the excavations are made by wheeled hydraulic loaders. During the excavation, face bolts are broken and the face is smoothed for an easy working (Denek, 2003).

2.1.2 Face bolts

In the tunnels having bigger cross-sectional area that are excavated in soft grounds in order to protect durability of the face of the tunnel, a new supporting element named as face bolting has been started to be used. In the application of supporting element, at the face of tunnel a hole is drilled having beforehand determined diameter and length, after that suitable sized anchor having the same length with the hole is placed and the hole is filled by cement injection completely (Kurt et al., 2002).

Face bolting application has been started to be used after the 80's in the tunnels having bigger cross-sectional area (bigger than 100 m^2) in Italy (Leca et al., 2003).

Face Bolts used in the Umbrella Technique are the supporting elements that are inspected the surface settlements (Kurt et al., 2002).

In the tunnels that are driven in clayey units, fifty per cent of surface settlements are occurred before tunnel comes to the measurement station. In order to explain this condition, the toothpaste tube example can be given as the followings: when the tube with a cover is compressed, there would be no change on its shape, but if it is compressed when it is open, there would be a deformation as much as the amount of the toothpaste that goes out from the tube. Similarly in tunnels, because of the ground stress, there would be some deformations at the surface as much as the amount of material at the tunnel face that looses its resistance to loadings (Kurt et al., 2002).

In order to prevent this condition, the internal parameters of the ground at the face should be changed positively. In the umbrella arch method, to realize this, a supporting element is developed namely ‘face bolt’ that is not used in NATM (Denek, 2003). With this aim, various numbers of bore holes having 12 m length, 10° inclination and 130 mm diameter (ϕ), are drilled and placed on the inner side of the holes ϕ 14mm. ST-3 equipments. After these workings, the holes are filled up with the cement injection (Denek, 2003).

The half length of this columns are breaked out during the excavations, and the other part of the columns are used as overlaps to the second umbrella step’s face bolts (Denek, 2003).

Figure 2. The appearence of tunnel face that is placed Umbrella Pipes and Face Bolts.

2.1.3 Umbrella pipes

Another supporting element of the Umbrella Arch method is ‘Umbrella Pipes’. Umbrella pipes are placed at the upper side section of tunnel surrounding with 30 cm space (Kurt et al., 2002).

By placing, the steel pipes having 114 mm diameter to the holes having 130 mm diameter with 6°- 8° inclination, a pre-arch is formed at the upper section of excavation area. The pipes that are placed have 9 m length and 3 m overlap length (Kurt et al., 2002).

After placing the umbrella pipes, the injection pipes and recycling pipes are located in the inner side of umbrella pipes. After closing the umbrella pipes with a muff, the surroundings of holes and pipes plastered for preventing injection leakages (Fig. 3) and the cement injection is started. The injection is applied until the cement-water mixture comes back from recycling pipe. By this operation, it is supplied that both the inside and outside of pipes are filled with cement injection (Denek, 2003).

Umbrella pipes have two important functions. The first function of the pipes is to work as a beam that resist to ground stresses and to prevent collapses during the excavation step. The second important function of the pipes is to spread ground loads homogeneously to support shell at the supported sections of tunnels (Kurt et al., 2002).

A tunnel face with umbrella pipes appearance and three dimensional appearance of a tunnel with umbrella pipes are shown in Figures 3 and 4 respectively.

Figure 3. An appearance of the metro tunnel face with umbrella pipes.

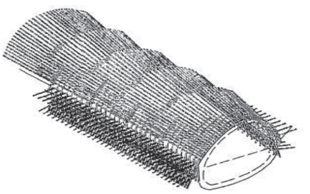


Figure 4. Three dimensional appearance of a tunnel with umbrella pipes and rock bolts.

3 OBSERVATIONS AND MEASUREMENTS BETWEEN 4+676,00-4+824,80 KM OF YENIKAPI LINE 2 UNKAPANI METRO TUNNEL TUNNEL SECTION

3.1 Methodology of the Works

During two years, on a selected tunnel route, researching, observations and measurements were made. At the start of the work, the tunnel that would be observed was determined, at the second step, in order to measure the convergence of the tunnel, fourteen stations were placed in different points in the tunnel and tunnel convergences were measured and recorded. At the station points, some geotechnical observations were made and also samples were taken from the tunnel face to determine geomechanical properties of the tunnel medium. After these works, all data collected were evaluated and the results were given at the end with suggestions. The methodology of the work is shown in Figure 5 schematically.

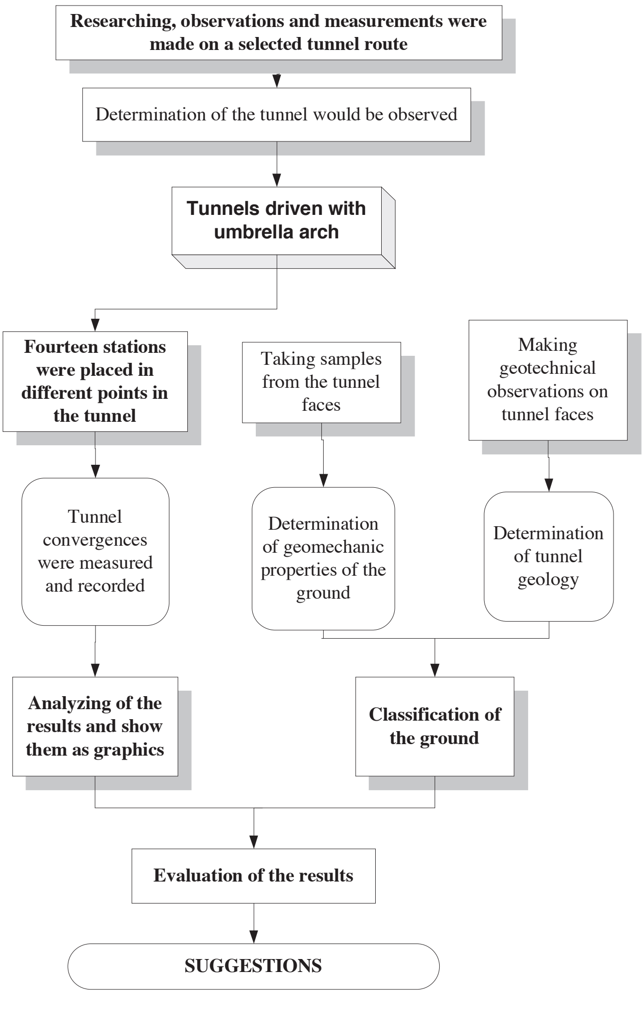


Figure 5. Methodology followed in this study.

3.2 The land observations on 4+676,00-4+824,80 km of Yenikapı Line 2 Unkapanı tunnel section.

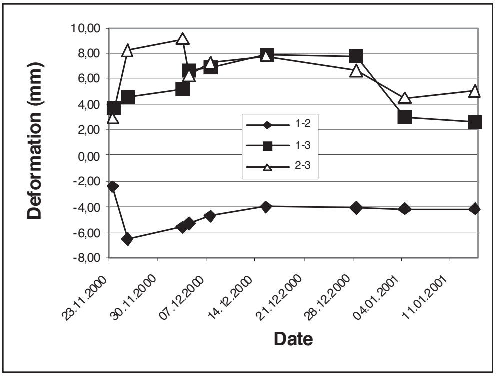
4+676,00-4+824,80 km of Yenikapı Line 2 Unkapanı (YH_2U) tunnel section is driven by using Umbrella Arch Method in poor ground conditions named in literature as Güngören Formation that contains clayey units having plastic feature and higher swelling potential.

In order to measure the deformations formed, 14 numbers of measurement stations are established and started to take measurement data orderly. At this section, also, geomechanical features of tunnel face are designated and the tunnel media is classified by making some observations on the face and taking some samples that characterize the face, and testing these samples in laboratory.

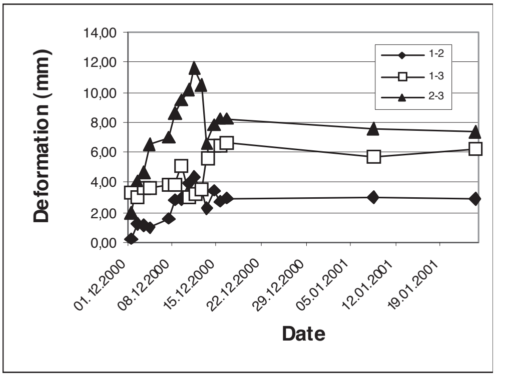
The support system that is applied on this section, the graphs that show deformations progress with time and the results of the face observations are given in Table I and Figure 6-9 respectively.

Table I. The support system that is applied between 4+672,40-4+786,40. Km of Yenikapı Line-2 Unkapanı tunnel section.

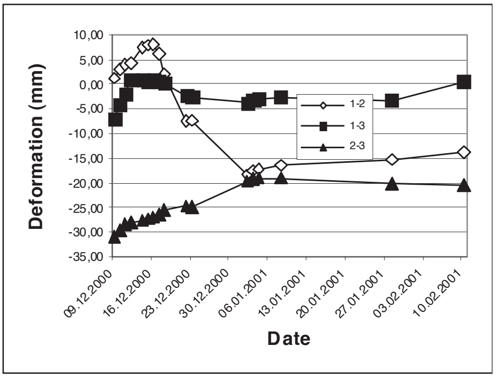
<table>
<tr><td rowspan="3">Umbrella Arch Preparation</td><td colspan="4">Tunnel Name: YH2U Support Type: A4
Bench: Upper Bench (*), Lower Bench ( ) Shotcrete thickness: 25cm</td></tr>
<tr><td>Pipe bolt
Type:$1^{1/2}$" inject.
length: 3m
number: 40</td><td>Bolt
type: SDB
length: 6m
number: 4/4</td><td>Two steel ribs distance: 0.60m.</td><td>First km:$4+672^{40}$
Last km: $4+675^{40}$</td></tr>
<tr><td colspan="4">Explanations: 1) For applying umbrella arch, 3m preparation advancing were made and five steel ribs were placed. At the end of advancing last steel rib were widened at the side 35cm and at the top 50cm, 2) Before excavations 40 numbers pipe bolts having $1^{1/2"}$ diameter were placed then one time the same numbers pipe bolts were placed for each two steel ribs 3) After preparation excavations, the face of tunnel were flattened and covered with wire mesh and shotcrete. And then 16 numbers face bolts were placed having 140mm diameter and 12m length, 4) The lower bench were advanced to the face as soon as possible.</td></tr>
<tr><td rowspan="3">UY-1 Station</td><td colspan="4">Tunnel Name: YH2U Support Type: A4
Bench: Upper Bench (*), Lower Bench (*) Shotcrete thickness:30cm</td></tr>
<tr><td>Umbrella Pipe
diameter: 114 mm
pipe length: 6m
number: 24</td><td>Bolt
type: SDB
length: 6m
number: 4/4</td><td>Two steel ribs distance: 0.60 m.</td><td>First km: $4+675^{40}$
Last km: $4+678^{40}$</td></tr>
<tr><td colspan="4">Explanations: 1) Excavations were made in three steps. The distance between two excavation steps is 2,4m, 2) At each advancing step, the face were covered with protection shotcrete having 5cm thickness, 3) Totally 12 numbers face bolt holes were drilled and steel equipments having ϕ 14 mm diameter and the same length with the hole were placed for face boltings.</td></tr>
<tr><td rowspan="3">UY-2,3,4,5,6,7,8,9,10,11,12 Stations</td><td colspan="4">Tunnel Name: YH2U Support Type: A4
Bench: Upper Bench (*), Lower Bench (*) Shotcrete thicness:30cm</td></tr>
<tr><td>Umbrella Pipe
diameter: 114 mm
pipe length: 9m
number: 24</td><td>Bolt
type: SDB
length: 6/3m
number: 4/4</td><td>Two steel ribs distance: 0.60 m.</td><td>First km: $4+768^{40}$
Last km: $4+786^{40}$</td></tr>
<tr><td colspan="4">Explanations:1)Umbrella pipes and face bolts were applied one times on each 6m, 2) SDB bolts were drilled for each steel support distance four numbers that two of them were drilled 6m length on +2.00 altitude , others were drilled 3m length on 0.00 altitude, 3) Totally 16 numbers face bolt holes were drilled and steel equipments having ϕ 14 mm diameter and the same length with the hole were placed for face boltings, 4) For each advancing step protection shotcrete were applied to the tunnel face, 5) Q335/335 type double layer wire mesh were used.</td></tr>
<tr><td rowspan="3">UY-13 Station</td><td colspan="4">Tunnel Name: YH2U Support Type: A4
Bench: Upper Bench (*), Lower Bench (*) Shotcrete thickness: 30cm</td></tr>
<tr><td>Umbrella Pipe
diameter: 114 mm
pipe length: 9m
number: 24</td><td>Bolt
type: SDB
length: 6/3m
number: 4/4</td><td>Two steel ribs distance: 0.60 m.</td><td>First km: $4+768^{40}$
Last km:$4+786^{40}$</td></tr>
<tr><td colspan="4">Explanations: 1) Umbrella pipes and face bolts were applied one times on each 6m 2) SDB bolts were drilled for each steel support distance four numbers that two of them were drilled 6m length on +2.00 altitude, others were drilled 3m length on 0.00 altitudes. 3) Totally 12 numbers face bolt holes were drilled and steel equipments having ϕ 14 mm diameter and the same length with the hole were placed for face boltings. 4) For each advancing step protection shotcrete were applied to the tunnel face 5) Q335/335 type double layer wire mesh were used.</td></tr>
</table>



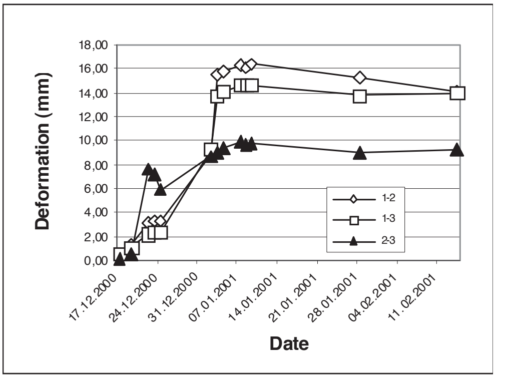
UY-1 Station-Km 4+676,00



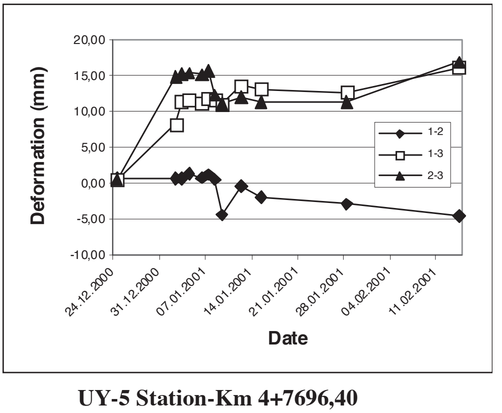
UY-2 Station-Km 4+680,20



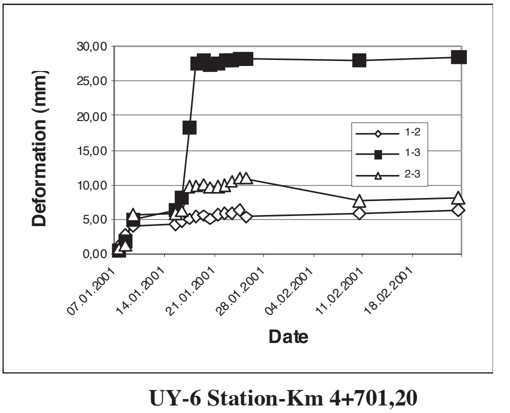
UY-3 Station-Km 4+685,00



UY-4 Station-Km 4+691,60

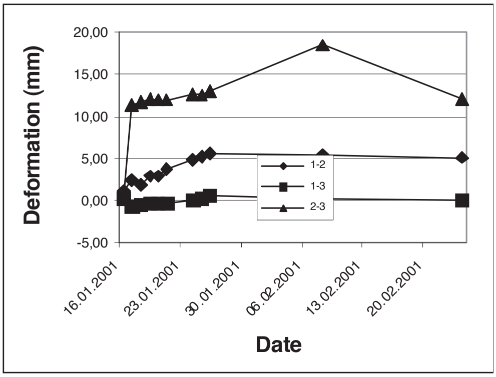


UY-5 Station-Km 4+7696,40

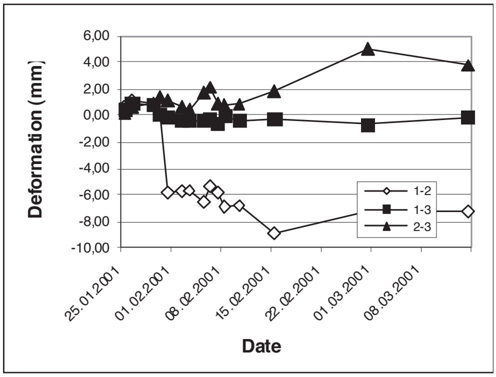


UY-6 Station-Km 4+701,20

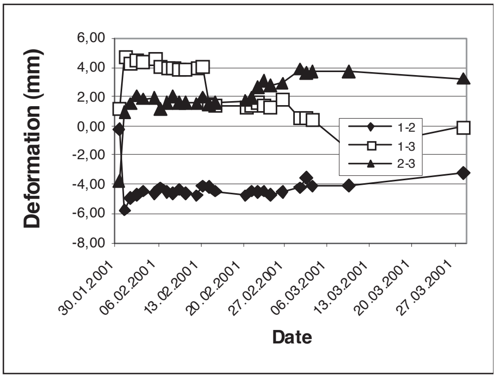
Figure 6.The progress of deformations with time measured on 4+676,00-4+824,80. km (UY-1,2,3,4,5,6,7,8,9,10,11,12,13,14 stations).



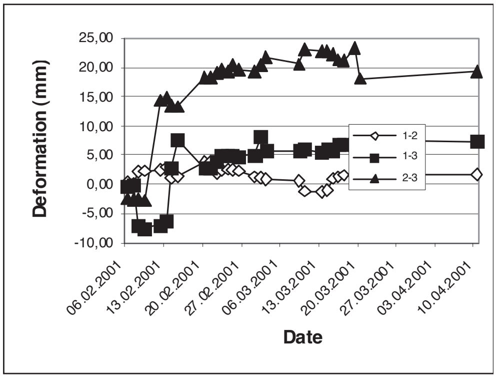
UY-7 Station-Km 4+707,20



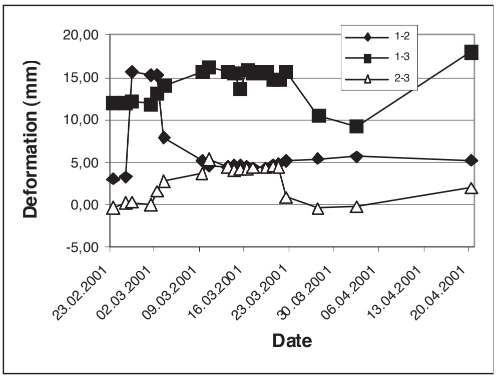
UY-8 Station-Km 4+715,00



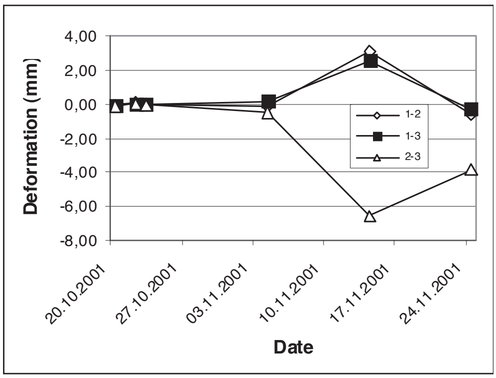
UY-9 Station-Km 4+721,00



UY-10 Station-Km 4+727,00

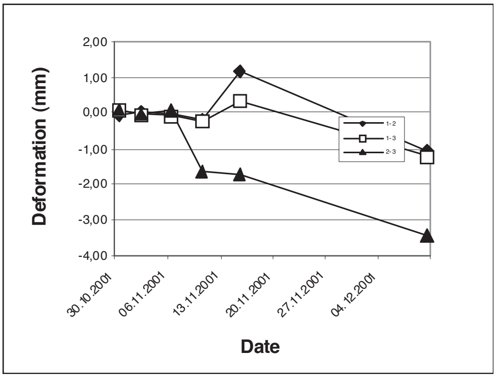


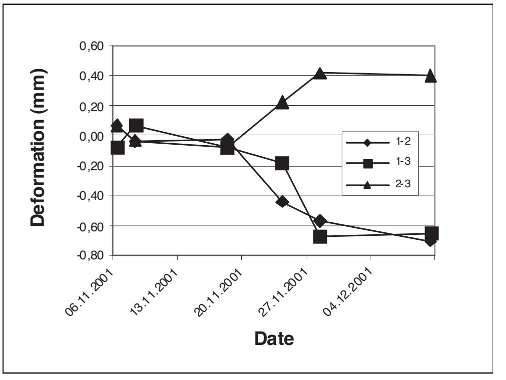
UY-11 Station-Km 4+733,00



UY-12 Station-Km 4+750,80

Figure 7.The progress of deformations with time measured on 4+676,00-4+824,80. km (UY-1,2,3,4,5,6,7,8,9,10,11,12,13,14 stations) .





UY-13 Station-Km 4+775,45 **UY-14 Station-Km 4+788,05**

Figure 8. The progress of deformations with time that is measured on 4+676,00-4+824,80. km (UY-1,2,3,4,5,6,7,8,9,10,11,12,13,14 stations).

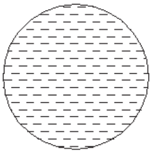
<table>
<tr><td rowspan="5"></td><td>Name: Yenikapı shaft Line 2 Unkapanı Tunnel</td><td>Date: 23/11/2000</td></tr>
<tr><td colspan="2">Km: 4+676,00-4+824,80</td></tr>
<tr><td colspan="2">Formation: Güngören Formation</td></tr>
<tr><td colspan="2">Ground characteristics: Güngören Formation contains clayey units having greeny gray color, higher plastic and swelling potantial.</td></tr>
<tr><td colspan="2">Soil class: CH (clay)</td></tr>
</table>

Figure 9. The face observation results on 4+676,00-4+824,80. km of YH_2U (yenikapı line 2 unkapanı) tunnel section.

4 CONCLUSIONS

All results of measurements and observations are given below;

1) The deformations measured on this section of YH2U tunnel where Umbrella Method is applied are generally between 0-10 mm. The highest deformation values recorded from these stations are 28, 38 mm and 30, 91 mm. It has been thought that these extreme values may cause some errors at the applying step of Umbrella Techniques or there may be a serious ground movement that arises from soil characteristics. In this case, some parameters on support system (on shotcrete thickness, number of umbrella pipes, face bolts, and their overlap length etc.) should be changed.
2) Tunnel inside deformation has not advanced to the project upper deformation limit that has value of 50 mm. On this section the highest deformation values are recorded at UY-6 station as 28, 38 mm, and at UY-3 station as 30, 91 mm.
3) Generally the recorded deformations do not have a character that shows a serious change with time. The deformations that arise with time and serious ground settlements are prevented by forming a strong arch with umbrella pipes, filling all cavities with care over the steel ribs and cutting down ground movements from face direction with face bolts. This result also shows that umbrella pipes spread ground loadings homogeneously.

5 SUGGESTIONS

New metro projects have been developed in Istanbul regarding the increasing needs of the city. Considering that difficulties of working under poor ground conditions are to be met in the future, there will be much more to do relating with the issue.

From this point of view, the main subjects that should be researched in the future and the suggestions about the method are given below:

1) Especially in tunnels that are driven under urbanization areas, in making engineering designs and in choosing of tunnelling methods to be applied, it is very important to know the engineering features and behaviours of tunnel media ground.

To realize a successful work in tunnel design and construction process, it is very important to make geotechnical researches in detail during the preparation step of the project.

2) The tunnels that are driven under excess urbanization areas should be thought differently from the tunnels that are constructed outside of urbanization areas and at the stage of tunnel design; this distinction should be taken under consideration.

3) In the tunnels that would be open with Umbrella technique, by using ground the monitoring systems should be used. All the ground movements should be observed carefully and the number of face bolts and umbrella pipes, overlaps lengths should be determined according to the ground movement characteristics.

4) At the tunnels that are driven especially in soft grounds that have higher deformation risk, the ring of tunnel should be completed in short time. To supply this, lower bench supports should come near to upper bench supports as soon as possible.

5) At the application of Umbrella method the inclinations of the Umbrella pipes should be suitable to Project values. If it is higher than the Project value there will be cavities between steel ribs and the pipes, this situation may cause ground settlements over the support shell. If the inclinations of pipes are lower than the Project value then at the stage of supporting there will be many problems. In the both situations the duty of the pipes will be canceled.

6) At the application of face bolts, the injection work should be made carefully, the holes should be completely filled with injection mixture. If it is thought that some amount of mixture would leak to natural cavities the injection work should be made one more time for the same hole until receiving would stop. At the holes that have ground water flow, in order to prevent ground water flow, the mixture should be injected with suitable density and pressure.

7) By using all results on this scope of research, at the shallow tunnels that are constructed under urbanization areas if there is necessity to work under poor ground conditions in order to supply construction safety, the Umbrella Method is recommended.

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Examination of Drilling Rate Index (DRI) of Rocks

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ABSTRACT Selecting machinery and equipment without examining the physical, mechanical and chemical properties of the rock may cause dramatic problems during working. Therefore, in drilling and blasting excavation, it is very important to find the rock properties before starting operation. In this study, ten different rock samples including marble and magmatic rocks were studied. Physical and mechanical properties of these samples were determined. From after them, in order to determine the drilling rate index (DRI) of the samples, brittleness test (S_{20}) and miniature drilling test (Sievers'J) were carried out. And also, specific destruction energy value was calculated from the stress-strain curves. DRI values and specific destruction energy results were evaluated and strong correlations between DRI and specific destruction energy was found.

1 INTRODUCTION

Drillability can be defined as drilling a rock in a certain time by a drilling tip. In other words, it is the ease of drilling a rock mass. Drilling speed is measured as the length of advance of excavation equipment within the rock mass in a time unit. Drilling rate index and drilling speed can be assumed to originating from the same concept. Drilling rate index is defined as low or high and drilling speed is defined as fast or slow. Drillability is dependent on a lot of geological (rock type and rock mass) and technical parameters. Thuro (1997) stated that type of drilling rig and impact powers of the rock drill influence the drilling rates. Also, rock material properties (physical, mineralogical and mechanical) and rock mass properties including jointing, anisotropy, cement type and weathering influence the drillability (Thuro, 1997).

Factors affecting drillability is shown in Figure 1.

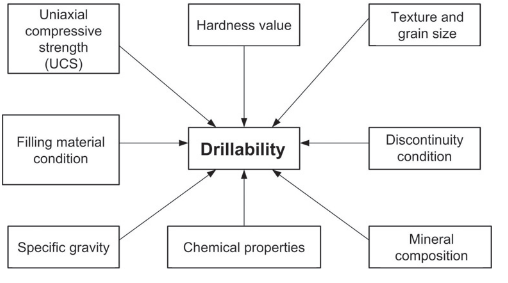


Figure 1. Factors affecting drillability

In conventional drill and blast excavation problem have occurred connected to the accurate prediction of drillability in hard rock. The drillability is a standard factor for the progress of excavation works. The estimation drillability in predicted rock conditions might bear an extensive risk of cost. Apart from conventional mechanical rock properties (compressive, tensile strength, young modulus) a new property for toughness/brittleness referring to drillability has been introduced: the specific destruction work Wz. This new method makes it possible to understand better the connection between drilling velocity and the main mechanical rock character (Thuro, 1997).

2. DRILLING RATE INDEX (DRI)

Drilling rate index is one used to determine rock drillibality. DRI is a measure of hardness or ease of rock drillibality. This index is developed by the Norwegian Institute of Technology and SINTEF in the 1960s. It has become a standard experiment in 1988 mainly used in Scandinavian countries (Yaralı and Soyer, 2008).

Drilling rate index is developed by Olsen and Blindheim in 1970 according to the results of two basic laboratory tests which are brittleness test (S_{20}) and Sievers' J test.

2.1 Brittleness Test

Brittleness test is developed by N. Von Matern and A. Hjelmer from Switzerland in 1943. In the following years, the experiment is modified for many different purposes. The principal flow diagram of the experiment is given in Figure 2.

The samples to be used in brittleness test are crushed by passing through a jaw crusher of 13.6 mm jaw width. They are classified by the help of 16 and 11.2 mm sieves. The pieces between -16 mm +11.2 mm are sent to brittleness test.

The volume of sample between -16 mm +11.2 mm equivalent to the volume of a material with 2.65 gr/cm^3 density and 500 gram mass is placed in the mortar of the test equipment. The 14 kg impact weight falls on to the sample in the mortar from 25 cm height 20 times (Figure 2). After this operation, the sample in the mortar passes again through the 11.2 mm sieve and the weight of the material passing through the sieve is recorded.

The proportion of the material which passes through the sieve (after operation) to the sample in the mortar (before operation) gives the brittleness of the rock sample. This operation should be repeated at lest three times with three different samples (Dahl, 2003).

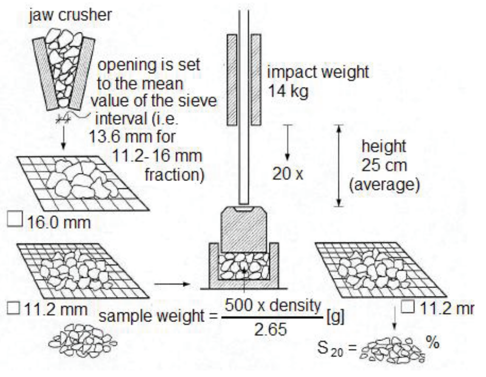


Figure 2. Principal flow diagram of brittleness test (Dahl, 2003).

2.2 Miniature Drilling (Sievers' J) Test

Miniature drilling experiment, which is a measure of surface hardness of rocks or the resistance they show to drilling, is developed by H. Sievers in 1950s. In this experiment, tungsten carbide tips with 8.5 mm diameter and 99^o top angle are used. As seen in Figure 3 and 4, the sample placed in the miniature drilling machine is set on the drilling tip with a 20 kg mass on top and then left free. Drilling operation is started on the 4 to 8 points previously marked. The miniature drilling experiment value is the depth of the hole after 200 turns of the miniature bits. Average miniature drilling experiment value is the average of the values obtained from the 4 to 8 holes (Dahl et al., 2003). Sievers' miniature drilling value (S_J) is found by averaging the depths of holes measured after one minute of operation of tips with 200 cycles per minute (Figure 3).

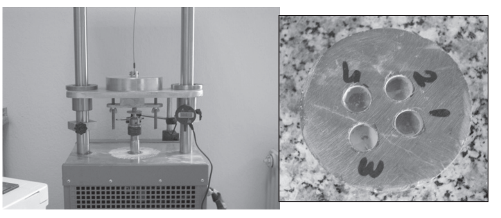


Figure 3. Miniature Drilling Device (Sievers' J) and a tested sample.

2.3 Determination of DRI

Drilling rate index value is found by the intersection of brittleness test result (S_{20}) and Sievers'J (Sj) value found by Sievers'J miniature drilling test in the diagram in Figure 4.

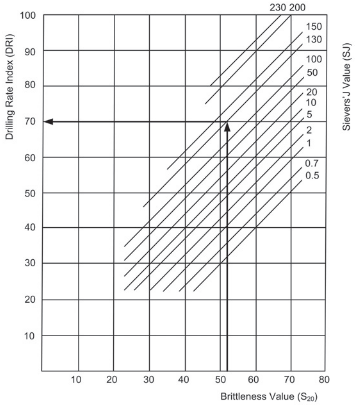


Figure 4. Diagram of DRI (Dahl, 2003).

3 LABORATORY STUDIES

Deformation tests are made on the cylindrical and cubic samples from various regions in order to find their specific energy values. After that, drilling rate index tests are carried out and the results are compared with the specific energy.

3.1 Calculation of the Destruction Specific Energy

Destruction specific energy is a new definiton for rock toughness. This new term is another reference for the drillability. In Figure 5, destruction specific energy (SEdes) is shown. Destruction specific energy is a measurement for quantity of energy requirement for destruction of a rock sample, in other words, the amount of energy for building new surfaces or cracks (Thuro, 1997). This value is the area below the stress – strain line and this value aims for better understanding of the relation between drilling, cutting performance and the main mechanical rock character (Atici and Ersoy, 2009).

Stress-strain experiments are made on rock from various regions according to ISRM 1981.

Table 1. Definition of sample used in the experiments.

Sample location	Sample definition	Sample form
Mugla - Fethiye	Fethiye Beige	Cube– Core
Mugla - Fethiye	Fethiye Brown	Cube
Mugla - Fethiye	Fethiye Rose	Cube
Gumushane	Green Basalt	Core
Izmir-Foca	Basalt	Core
Izmir-Bergama	Granite	Core
Afyon	Marble	Cube
Yatagan	Marble	Cube
Belevi	Marble	Cube
Ege	Marble	Cube

Stress-Strain curves of samples are developed according to the results of the deformation experiments. The area under the deformation curve gives the specific energy amount of the rock to deform. As shown in Figure 5, after developing the deformation curves of all samples, their specific deformation energies are calculated.

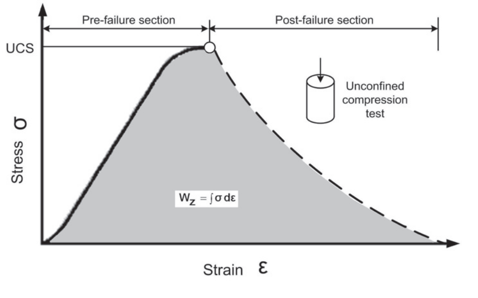


Figure 5. Calculation of specific destruction energy from stress-strain curve.

Specific destruction energy is found by calculating the area under the deformation curve (Tale 2, Figure 6). Atici and Ersoy (2008) calculated the specific destruction energy by definite integral method.

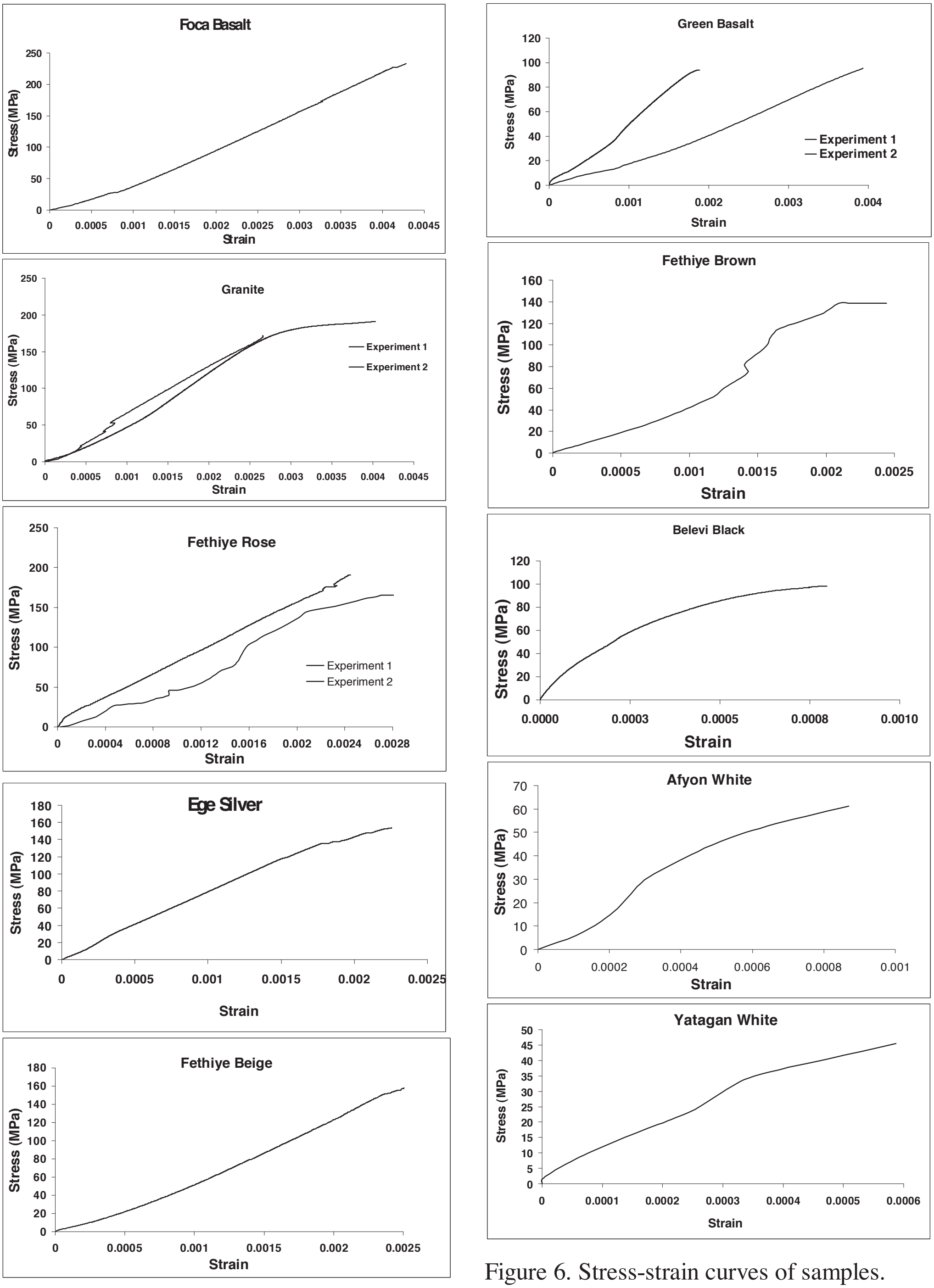


Figure 6. Stress-strain curves of samples.

Table 2. Specific Destruction Energy Values.

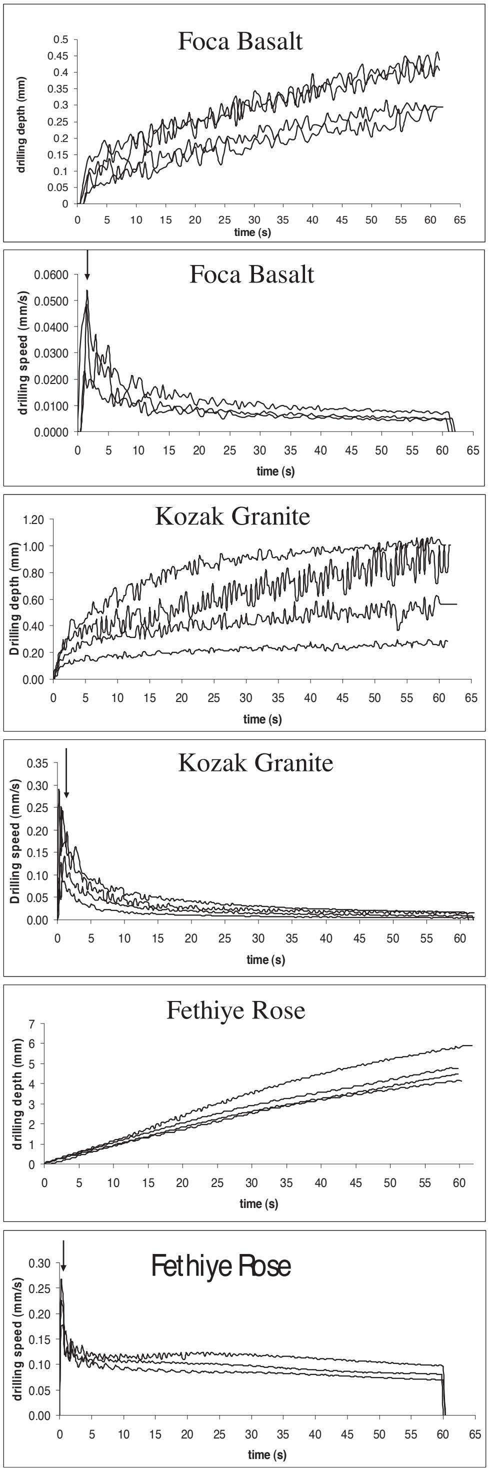
Sample Definition	Specific destruction energy (kj/m^3)
Foca Basalt	458
Granite	338
Fethiye Rose	240
Ege Silver	193
Fethiye Beige	182
Green Basalt	170
Fethiye Brown	170
Belevi Black	55
Afyon White	31
Yatagan White	15

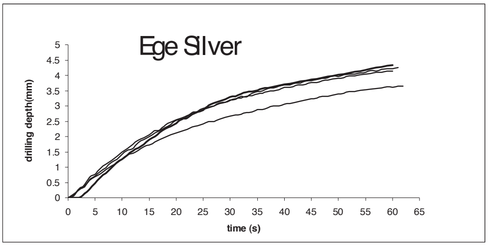
3.2 Determination of DRI

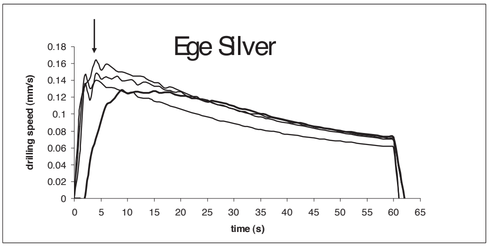
In order to determine DRI, first brittleness test, then miniature drilling test is carried out. The two values obtained from these tests are marked on the DRI graphic (Figure 4) and point of intersection gives the DRI value of the sample. The brittleness and drilling test results are given In Table 3. In Figure 7, the penetration curves and drilling velocity are shown.

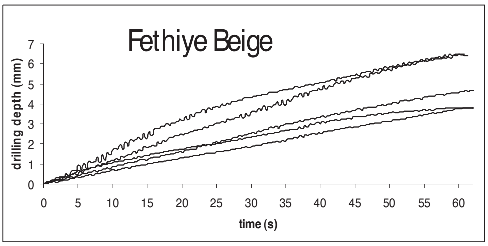
Table 3. DRI values of samples.

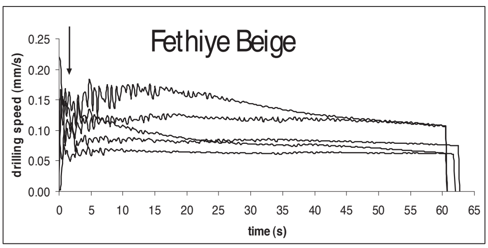
Sample definition	**Brittleness value (S_{20}, %)**	**Miniature Drilling value (Sievers' J, mm)**	**DRI**
Foca Basalt	30.02	0.38	**26**
Granite	50.32	0.72	**48**
Green Basalt	43.86	8.57	**54**
Belevi Black	44.91	4.10	**54**
Ege Silver	46.85	5.23	**56**
Fethiye Beige	50.43	5.02	**58.50**
Fethiye Rose	51.52	4.81	**60**
Fethiye Brown	53.64	5.54	**62**
Afyon White	54.38	5.25	**65**
Yatagan White	58.44	8.26	**67**

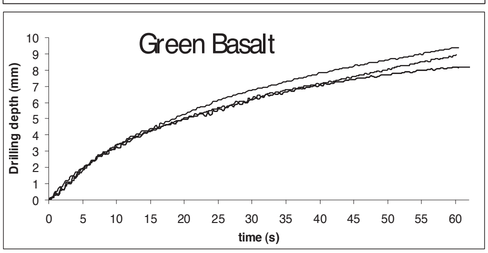


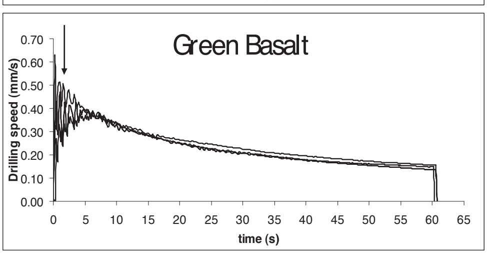

Ege SIlver
drilling depth(mm)
time (s)

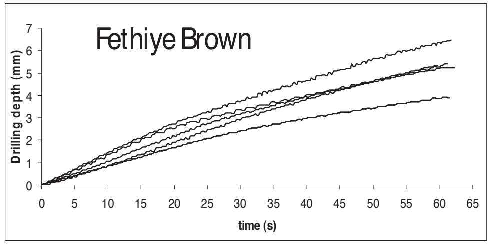

Ege SIlver
drilling speed (mm/s)
time (s)

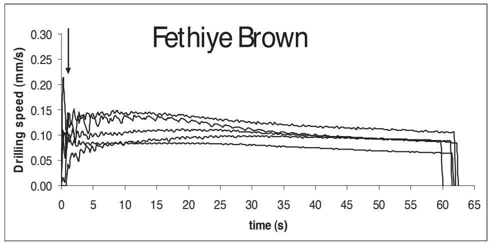

Fethiye Beige
drilling depth (mm)
time (s)

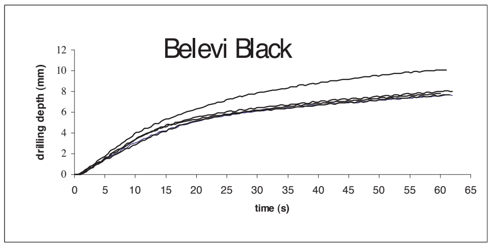

Fethiye Beige
drilling speed (mm/s)
time (s)

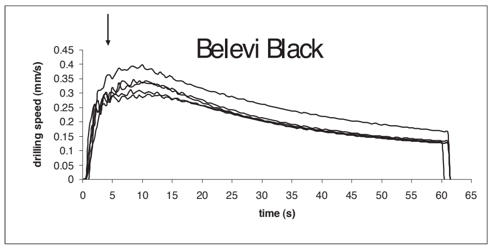

Green Basalt
Drilling depth (mm)
time (s)

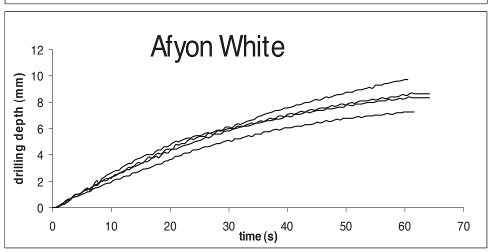

Green Basalt
Drilling speed (mm/s)
time (s)

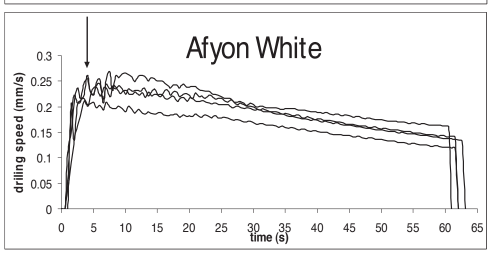

Fethiye Brown
Drilling depth (mm)
time (s)


Fethiye Brown
Drilling speed (mm/s)
time (s)


Belevi Black
drilling depth (mm)
time (s)


Belevi Black
drilling speed (mm/s)
time (s)


Afyon White
drilling depth (mm)
time (s)


Afyon White
drilling speed (mm/s)
time (s)

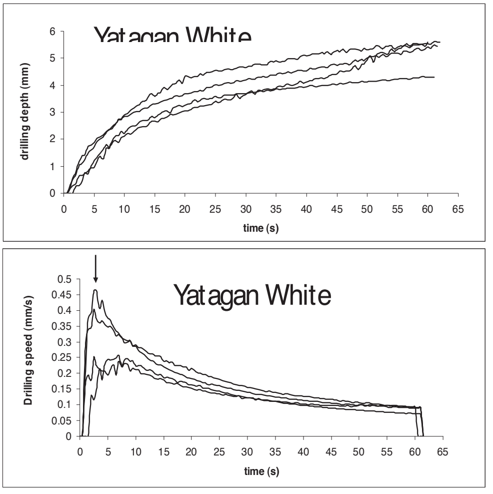


Figure 7. Penetration and drilling velocity curves of samples.

4 EVALUATION OF THE RESULTS

Drilling depth and speed curves are examined in detail. It is observed that the maximum drilling velocity is achieved in the first five seconds in all samples. All drilling depth curves change smoothly (linear) except for Foca basalt and Granite curves. Foca basalt and granite curves are variable.

The relationship of DRI and specific destruction energy values is evaluated by regression as polynomial, linear and exponential. The R^2 values are 85%, 73%, and 69%, respectively. This means that polynomial regression best describes the relation between DRI and specific destruction energy.

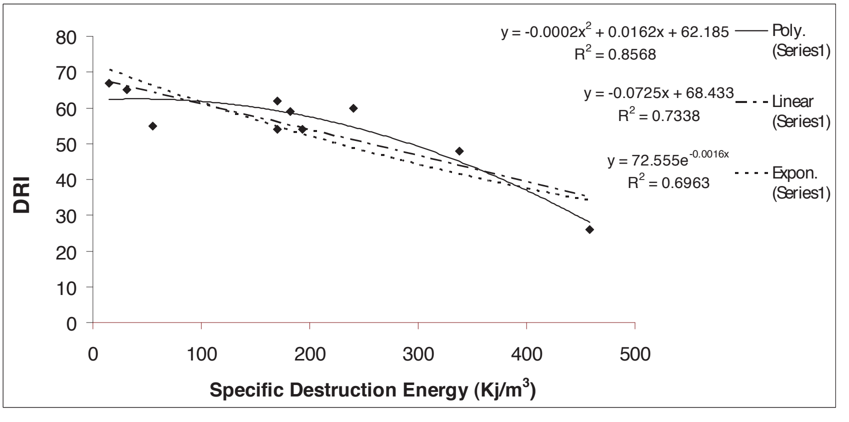


Figure 8. Connection between DRI and specific destruction energy.

5 CONCLUSIONS

DRI curves of marbles change linear by time. Drilling speed curves of all samples increase to maximum value in the first five seconds. And then, drilling speed curves decrease and reach a steady level. Most wear of drilling bits occur in the first a few seconds, as previously reported by Dahl et al. (2003).

DRI values and specific destruction energy results were evaluated and strong correlations between them was found. Relationship of DRI values with the specific destruction energy values is evaluated by regression as polynomial, linear and exponential. The R^2 values are 85%, 73%, and 69% respectively. Polynomial regression describes well the relationship between DRI and specific destruction energy.

ACKNOWLEDGEMENTS

Authors thanks to Ass. Prof. Dr. Olgay Yarali for his help, Ass. Prof. Dr. Zeki Karaca for providing the test samples and SINTEF which is the developer of the original test instruments.

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Türkiye'deki Bazı Doğaltaş Ocaklarında Zincirli Kollu Kesici Uygulamaları ve Ocak Üretimine Etkileri

Chain Saw Machine Applications at Some Natural Stone Quarries in Turkey and its Effects on Quarry Productivity

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ÖZET Doğal taş blok üretimi, ocaklarda genellikle elmas tel kesme yöntemi ile gerçekleştirilmektedir. Özellikle az çatlaklı ve masif olarak nitelendirilebilecek doğal taş ocaklarında bu üretim yöntemine bir alternatif de zincirli kollu kesicilerdir. Zincirli kollu kesiciler ile daha düşük basamak yüksekliklerinde, ocak kayaç yapısı izin verdiği sürece sayalama işlemine gerek kalmaksızın blok üretimi yapmak mümkün olabilmektedir. Bunun yanında yumuşak ve orta sert olarak nitelendirilebilecek, süreksizlik içeren doğal taş ocaklarında blok üretiminde elmas tel kesme ile birlikte kullanımı da mümkündür. Bu çalışmada, Türkiye'nin farklı bölgelerinde bulunan doğaltaş ocaklarındaki elmas tel kesme ve zincirli kollu kesicilerle blok çıkartma yöntemleri, ocak tasarım parametreleri değerlendirilmiştir. Zincirli kollu kesicilerin farklı doğaltaş ocaklarındaki performansları belirlenmiştir. Blok kazanım oranları zincirli kollu kesicilerin kullanımı ile %15-20'lerden %75'e kadar çıktığı görülmüştür.

ABSTRACT Production of the natural stones is generally carried out by diamond wire cutting method in quarries. Chain saw machine is an alternate to this method in the quarries that can be defined especially as massive or containing less discontinuities. Block production without any consequent dimensioning is possible in the lower bench height by chain was machines in suitable quarries. In addition, the machine can be used together with diamond wire cutting method in soft or medium hard discontinuities bearing quarries. In this study, block production with chain saw machine and quarry design parameters were investigated in different quarries located in different regions of Turkey. The cutting performance of the chain saws in different quarries were determined. Block recovery ratio was observed as increasing from 15-20% to 75% by using chain saw machines.

1 GİRİŞ

Doğal taş endüstrisinde ekonomik zorunluluklar ve teknolojideki gelişmeler üreticileri daha yüksek kesme hızları ve düşük maliyetlerde üretim yapılmasına imkan veren yöntem ve makinaların arayışına zorlamaktadır. İlk olarak 1928 yılında yeraltı kömür işletmelerinde "potkabaç" adıyla kömür kazısında kullanılmaya başlanan zincirli kollu kesiciler, burada kömür damarının tabanında ve kömür içinde alt kesme yapılıp serbest bir yüzey oluşturularak kömürün kolay sökümünü sağlamaktaydı. 1980'li yıllara kadar bu şekilde kullanılan zincirli kollu kesiciler gelişmiş kazıcı ve yükleyicilerin kömür üretiminde kullanılmasıyla doğal taş blok üretimine kaydırılmıştır. Zincirli kollu kesicilerin ocaklarda kullanımı blok üretim miktarının artmasını, verimliliğin yükselmesini ve blokların şekil-boyut özelliklerinin iyileşmesini sağlamıştır.

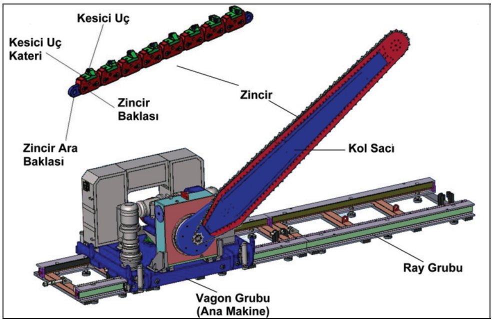
Zincirli kollu kesiciler, ocak jeolojik yapısının uygun olduğu durumlarda tek başına blok üretiminde kullanılırken, genelde süreksizlik içeren doğal taş ocaklarında elmas tel kesme ile birlikte kullanılmaktadır. Elmas tel kesme makinalarının yanına sadece bir adet zincirli kollu kesici ilavesi ile ocak genel veriminin %20 arttığı gözlenmiştir (Çopur vd., 2006). Blok verimliliği elmas tel kesme yönteminin uygulandığı ocaklarda %10-15 seviyelerinden, ocak basamak tasarımı değiştirilerek zincirli kollu kesicilerin kullanılmasıyla %75-80'lere kadar çıkabilmektedir.

Bu çalışmada zincirli kollu kesicilerin Türkiye'de bulunan farklı ocaklardaki kullanım şekilleri ve ocak verimliliğine etkileri araştırılmıştır.

2 ZİNCİRLİ KOLLU KESİCİLERİN YAPISI

Zincirli kollu kesici makinalar, çeşitli firmalarca çok değişik tip ve kapasitelerde imal edilmektedir. Kayaç yapıları farklı olduğundan kesmede kullanılan makinalar ve özellikle kesici uçların yapısı da büyük farklılık göstermektedir. Zincirli kollu kesiciler doğal taş ocaklarında ana kütleden blok kesiminde, ayrılan blokların sayalanmasında ve galeri açmada kullanılmaktadırlar. İkinci ve üçüncü tipi paletli imal edilerek hareket kolaylığı sağlanmıştır. Bu çalışmada konu edilen zincirli kollu kesiciler dağ kesimi diye adlandırılan ana kütleden blok kesiminde kullanılan makinalardır (Şekil 1).

Bu makinaların toplam ağırlığı yaklaşık 2-10 ton arasında olup, elektrik enerjisiyle çalışan üç adet motoru mevcuttur. Bu motorlardan birincisi kesici zincir sisteminin bağlı olduğu kolu hareket ettirmekte, ikinci motor kesici zincir sistemini çevirmek için, üçüncü motor da makinanın ray üzerinde ileriye geriye hareketini sağlamaktadır (Kulaksız, 2007). Zincirli kollu kesicilerin temel bileşenleri; ana makine, ray sistemi, kesme kolu (jib)-kesici zincir tertibatı ve kesici uçlar olarak sıralanabilir.



Şekil 1. Zincirli kollu kesicilerin temel bileşenleri (Demirel, 2008).

Ray Sistemi: Sökülüp uç uca eklenebilir olması nedeniyle fazlaca uzun değillerdir. Kesim biten taraftan sökülen parça, ilerleme yönünde monte edilir. Bir ocak için iki tane 3 metrelik, bir tane 2 metrelik ve bir tane 1 metrelik olmak üzere toplam 9 metre ray yeterli olmaktadır. Yeni gelişmelerle hidrolik sabitleyiciler ve/veya kendiliğinden hareketli raylar kullanılmaktadır.

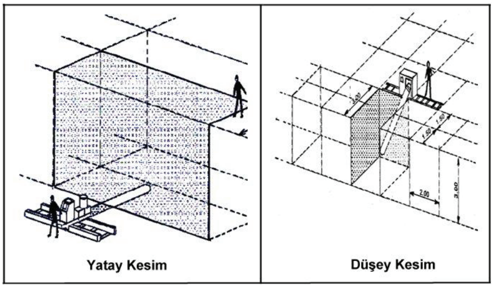
Ana Makina: Ray üzerinde hareket eden, elektrik motorlarını, hidrolik güç ünitelerini, kontrol ve kumanda cihazlarını ve kesme kolunu taşıyan kısma ana makina adı verilir. Bu ana makina sağlam bir şasi üzerinde oturmaktadır. Motor gücü 25-75 HP arasında değişmektedir.

Kesme Kolu (Jib) ve Kesici Zincir Tertibatı: Zincirli kollu kesicilerde kesme kolu hidrolik güç kaynağı ile kolayca yatay ve düşey kesme pozisyonlarına getirilebilir. Böylece aynı makina ile hem yatay hem de düşey kesim yapılabilir. Özel alaşımlı saçtan imal edilmiş kol makinaya kolayca takılıp sökülebilir. Zincirli kollu kesiciler kol uzunluğuna bağlı olarak 2 m-6 m arasında değişen derinliklerde kesim yapabilirler.

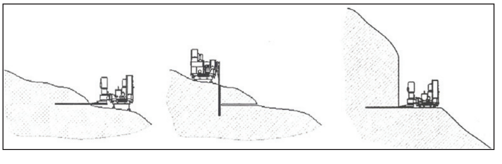
Kol üzerinde dönen zincir tertibatı üzerine kesici uçları taşıyan 7'li ve 11'li kesme takımı bulunmaktadır. Bu 7'li ve 11'li kesme takımı belirli bir dizilişle yer alır ve üzerinde kesici uçlar vidalanır. Kesici uçlar genelde Tungsten Karbitten yapılır ve doğal taşların kesiminde bu uçların özel kesme açıları bulunmaktadır. Bu özel yapı onlarla hızlı kesme ve uzun süre dayanma imkanı sağlar. Bunların dört kenarı da kesicidir. Bir yüzey körlenince kolayca diğer yüzey devreye alınır. Kesme işlemi susuz gerçekleştirildiğinden -15^0 ile 40^0 arasındaki hava sıcaklığında üretim yapmak mümkündür.

Zincirin kesimini kolaylaştıran aşındırıcılar genellikle tungsten karbit ve daha sert doğal taşlarda ise strapax (polikristalin elmas) 'tır.

Kollu kesiciler ocakta genellikle Şekil 2'deki gibi düşey ve yatay kesimlerde kullanıldığı gibi, yeni ocak ağzı açımında Şekil 3'deki gibi ve serbest yüzey bulunmayan taban hazırlık kesimlerinde Şekil 4'deki gibi kullanılmaktadır.

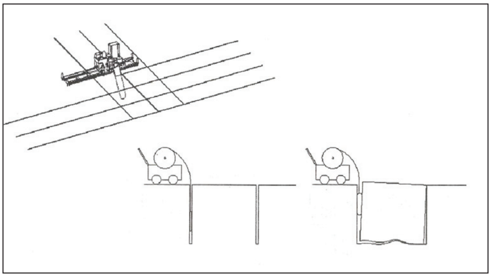


Şekil 2. Zincirli kollu kesiciler ile gerçekleştirilen yatay ve düşey kesimler (Kulaksız, 2007).



Şekil 3. Yeni ocak ağzı açımında zincirli kollu kesici kullanımı (Fantini, 2005).

Kesim işlemi, önceden belirlenmiş plan çerçevesinde hareket eden kolun kaya içinde ilerlemesi ve dönmesi sayesinde gerçekleşir. Karbonatlı ve sert kayaçlarda kolaylıkla kesim imkânı sağlamaktadır.



Şekil 4. Taban hazırlık kesimlerinde zincirli kollu kesici kullanımı (Fantini, 2005).

3 FARKLI OCAKLARDA ZİNCİRLİ KOLLU KESİCİ UYGULAMALARI

Zincirli kollu kesiciler ocak tasarımı, doğal taş içerisindeki süreksizlikler ve kayacın sertlik ve aşındırıcılığına bağlı olarak farklı şekillerde kullanılmaktadırlar. Burada Türkiye'deki değişik yapıdaki doğal taş ocaklarında kullanılan zincirli kollu kesicilerin özellikleri, performans parametreleri ve ocaklardaki kullanım şekilleri hakkında bilgi verilmektedir.

3.1 Eskişehir-Sivrihisar Bej Ocak İşletmesi

Bu ocakta basamak yüksekliği 7 m olarak seçilmiş ve zincirli kollu kesicinin kullanılmasından sonra da bu yükseklik değiştirilmemiştir. Zincirli kollu kesiciler ocakta sadece alt kesimlerde kullanılmaktadır, düşey kesimler yine elmas tel kesme ile gerçekleştirilmektedir. Sivrihisar Bej ocağında kullanılan zincirli kollu kesicinin teknik özellikleri Çizelge 1'de verilmiştir.

Ocakta yapılan ölçümlerde zincirli kollu kesicilerin kesme performansları incelenmiş ve sonuçlar genel olarak Çizelge 2'de verilmiştir.

Kollu kesiciler ocakta delik sayısını azaltmak ve üretim hızını artırmak için kullanıma alınmış olup, sadece alt kesimde kullanılmaktadır.

Çizelge 1. Eskişehir-Sivrihisar Bej ocağında kullanılan kollu kesicinin teknik özellikleri.

Zincirli kollu kesici marka ve modeli	Garrone MCRH 340
Ağırlık	6500 kg
Toplam kurulu güç	53 kW
Maksimum kesme derinliği	4,4 m
Kesme genişliği	42 mm
Hidrolik tank kapasitesi	220 lt
Kesme kolu dönüş açısı	360^0
Zincir dönüş hızı	1,8 m/s
Kol ilerleme hızı	0-20 cm/dk
Kesici kol uzunluğu	3,4 m
Kullanılan keski türü	11'li set
Kullanılan keski sayısı	110 keski
Keski markası	Garrone

Çizelge 2. Sivrihisar Bej ocağında kullanılan kollu kesicinin çalışma ve performans parametreleri.

Kesme derinliği	3,4 m
Kesici kolun ilerleme hızı	1,5-2 m/sa
Kesme miktarı	5-6,5 m^2/sa
Hidrolik tüketimi	0,11 lt/sa
Keski aşınması	140 m tek köşe için (560 m tüm keski için)
Gres yağı tüketimi	0,4 kg/sa

Zincirli kollu kesici ile üretilen blokların kullanım oranı %63-75 arasında değişmektedir. Bu makinelerin kullanılmasıyla ocakta blok verimliliğinde 4 kat artış üretim hızında ise iki kat artış sağlanmıştır. Blok devirme işleminin önüne geçildiğinden hafriyat işlemleri azalmış ve enerji ve işçilikte önemli tasarruflar (% 30'a yakın) sağlanmıştır.

3.2 Burdur Bej Ocak İşletmesi

Bu ocakta basamak yüksekliği 8 m olarak seçilmiş ve zincirli kollu kesicinin kullanılmasından sonra da bu yükseklik değiştirilmemiştir. Ocağın genel görünümü Şekil 5'de verilmiştir. Zincirli kollu kesiciler ocakta genellikle alt kesimlerde kullanılmaktadır (Şekil 6), düşey kesimler yine elmas tel kesme ile gerçekleştirilmektedir.

Şekil 5. Burdur bej ocağının genel görünümü.

Şekil 6. Zincirli kollu kesicinin taban kesimi sırasındaki görünümü.

Burdur Bej ocağında kullanılan zincirli kollu kesicinin teknik özellikleri Çizelge 3'de verilmiştir.

Çizelge 3. Burdur Bej ocağında kullanılan kollu kesicinin teknik özellikleri.

Zincirli kollu kesici marka ve modeli	Garrone MCRH 340
Ağırlık	6500 kg
Toplam kurulu güç	53 kW
Maksimum kesme derinliği	4,4 m
Kesme genişliği	42 mm
Hidrolik tank kapasitesi	220 lt
Kesme kolu dönüş açısı	360^0
Zincir dönüş hızı	1,8 m/s (maksimum)
Kol ilerleme hızı	0-20 cm/dk
Kesici kol uzunluğu	3,5 m
Kullanılan keski türü	11'li set
Kullanılan keski sayısı	112 keski
Keski markası	Böhler

Ocakta yapılan ölçümlerde zincirli kollu kesicilerin kesme performansları incelenmiş ve sonuçlar genel olarak Çizelge 4'de verilmiştir.

Zincirli kollu kesici kullanımı ile blok verimliliğinde %60 artış sağlanmıştır.

Çizelge 4. Burdur Bej ocağında kullanılan kollu kesicinin çalışma ve performans parametreleri.

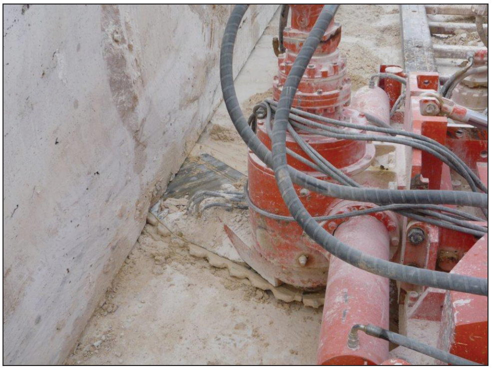
Kesme derinliği	3,35 m
Kolun ilerleme hızı	1,1 m/sa
Kesme miktarı	3,68 m^2/sa
Hidrolik tüketimi	0,17 lt/sa
Keski aşınması	120 m tek köşe için (480 m tül tüm keski için)
Gres yağı tüketimi	0,8 kg/sa

3.3 Manisa-Alaşehir Traverten Ocak İşletmesi

Yüksekliği 7,5-8 m arasında üç basamaktan oluşan bu ocakta zincirli kollu kesici sadece taban kesimlerinde, ocak genelinde ise elmas tel kesme yöntemiyle birlikte kombine bir şekilde kullanılmaktadır. Ocağın genel görünümü Şekil 7'de verilmiştir. Zincirli kollu kesiciler ocakta genellikle alt kesimlerde kullanılmaktadır (Şekil 8), düşey kesimler yine elmas tel kesme ile gerçekleştirilmektedir.

Manisa-Alaşehir ocağında kullanılan zincirli kollu kesicinin teknik özellikleri Çizelge 5'de verilmiştir.

Şekil 7. Manisa-Alaşehir Traverten ocağının genel görünümü.



Şekil 8. Ocakta zincirli kollu kesici ile yapılan taban kesimi.

Çizelge 5. Manisa-Alaşehir traverten ocağında kullanılan kollu kesicinin teknik özellikleri.

Zincirli kollu kesici marka ve modeli	Fantini MOD 50.81 /RA.TC
Ağırlık	5200 kg
Toplam kurulu güç	50 kW
Kesme genişliği	38 mm
Kesme kolu uzunluğu	3,4 m (maksimum) 3,2 m (kullanılan)
Hidrolik tank kapasitesi	400 lt
Kesme kolu dönüş açısı	180^0
Zincir dönüş hızı	0,7 m/s (maksimum)
Kol ilerleme hızı	0-13 cm/dk
Kullanılan keski türü	11'li set
Kullanılan keski sayısı	67
Keski markası	Böhler

Ocakta yapılan ölçümlerde zincirli kollu kesicilerin kesme performansları incelenmiş ve sonuçlar genel olarak Çizelge 6'da verilmiştir.

Çizelge 6. Manisa-Alaşehir traverten ocağında kullanılan kollu kesicinin çalışma ve performans parametreleri.

Kesme derinliği	3,15 m
Kesici kolun ilerleme hızı	3 m/sa
Kesme miktarı	9,45 m^2/sa
Hidrolik tüketimi	1-1,3 lt/ay
Keski aşınması	
Gres yağı tüketimi	120 kg/ay

Zincirli kollu kesici kullanımı ile blok verimliliğinde, üretim hızında artış sağlanmıştır.

3.4 Denizli-Zeyelli Mevkii Traverten Ocak İşletmesi

Ocakta basamak yüksekliği zincirli kollu kesici kullanımı ile birlikte 10 m'den kol uzunluğu ile kesim yapılabilecek uzunluk olan 3 m'ye düşürülmüştür. Ocağın genel görünümü Şekil 9'da verilmiştir.

Şekil 9. Denizli traverten ocağının genel görünümü.

Denizli traverten ocağında kullanılan zincirli kollu kesicinin teknik özellikleri Çizelge 7'de verilmiştir.

Çizelge 7. Denizli traverten ocağında kullanılan kollu kesicinin teknik özellikleri.

Zincirli kollu kesici marka ve modeli	Fantini MOD 50.81/P
Ağırlık	5200 kg
Toplam kurulu güç	50 kW
Maksimum kesme derinliği	3,4 m
Kesme genişliği	38 mm
Hidrolik tank kapasitesi	400 lt
Kesme kolu dönüş açısı	180^0
Zincir dönüş hızı	0,7 m/s (maksimum)
Kol ilerleme hızı	0-13 cm/dk
Kesici kol uzunluğu	3,4 m
Kullanılan keski türü	11'li set
Kullanılan keski sayısı	103
Keski markası	Sandvik

Ocakta yapılan ölçümlerde zincirli kollu kesicilerin kesme performansları incelenmiş ve sonuçlar genel olarak Çizelge 8'de verilmiştir.

Çizelge 8. Kollu kesicinin çalışma ve performans parametreleri.

Kesme derinliği	3 m
Kesici kolun ilerleme hızı	3 m/sa
Kesme miktarı	9 m^2/sa
Hidrolik tüketimi	0,16 lt/sa
Keski aşınması	175 m tek köşe için (700 m tül tüm keski için)
Gres yağı tüketimi	1,8 kg/sa

Zincirli kollu kesici ocakta alt kesimlerde kullanılmakta düşey kesimler elmas tel kesme ile gerçekleştirilmektedir. Zincirli kollu kesici kullanımı ile blok verimliliği %65 den %90'a ulaşmıştır. Aylık üretilen blok miktarında %70 oranında artış sağlanırken işçilikte %25 oranında tasarruf elde edilmiştir.

3.5 Denizli-Kaklık Traverten Ocak İşletmesi

Demmer A.Ş.'ye ait olan ocakta basamak yüksekliği ilk olarak 12 m olarak seçilmiş, blok devrilmesi sırasında tabaka düzlemleri boyunca oluşan kırılmalar dolayısıyla blok kaybı yüksek olunca 8 m'ye kadar düşürülmüştür. Blok devrilmesi sırasında oluşan kırılmalara bir örnek Şekil 10'da verilmektedir (Demirel, 2008).

Şekil 10. Yüksek basamak kesiminde blok devrilmesi sırasında oluşan kırılmalar.

Zincirli kollu kesiciler öncelikle ocakta taban kesiminde kullanılmaya başlanmış düşey kesimler elmas tel kesme ile gerçekleştirilmiştir. Daha sonra basamak yüksekliği 3 m'ye indirilerek düşey kesimler de zincirli kollu kesici ile yapılmaya başlanmıştır. Ocağın eski üretim yapılan kesiminin görünümü Şekil 11'de, düşük basamak yüksekliğinde üretim yapılan yeni bölgelerin genel görünümü ise Şekil 12'de verilmektedir.

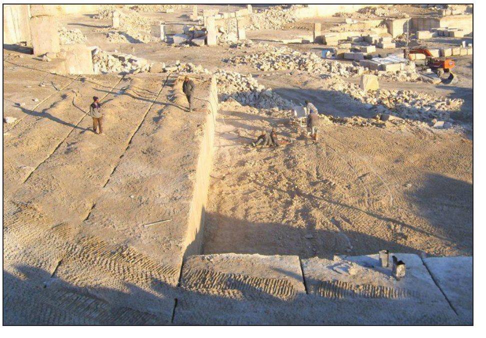
Şekil 11. Denizli traverten ocağının eski işletilen yüksek basamaklı kesiminin genel görünümü.

Şekil 12. Denizli traverten ocağının şu an işletilen alçak basamaklı kesiminin genel görünümü.

Şu an ocakta taban kesimleri ve düşey kesimler zincirli kollu kesiciler ile gerçekleştirilmektedir. Kesim işlemini yapan makinanın ve kesim sahasının görünümü Şekil 13-14'de verilmektedir.

Denizli traverten ocağında kullanılan zincirli kollu kesicinin teknik özellikleri Çizelge 9'de verilmiştir.

Şekil 13. Ocakta zincirli kollu kesici ile yapılan düşey kesim.



Şekil 14. Denizli traverten ocağının şu an işletilen kesim sahasından bir görünüm.

Çizelge 9. Denizli traverten ocağında kullanılan kollu kesicinin teknik özellikleri

Zincirli kollu kesici marka ve modeli	Demmak DZK-3400
Ağırlık	8100 kg
Toplam kurulu güç	45 kW
Maksimum kesme derinliği	3,4 m
Kesme genişliği	42 mm
Hidrolik tank kapasitesi	204 lt
Kesme kolu dönüş açısı	180^0
Zincir dönüş hızı	0,7 m/s
Kol ilerleme hızı	0-18 cm/dk
Kesici kol uzunluğu	3,4 m
Kullanılan keski türü	7'li set
Kullanılan keski sayısı	87
Keski markası	Böhler

Ocakta yapılan ölçümlerde zincirli kollu kesicilerin kesme performansları incelenmiş ve sonuçlar genel olarak Çizelge 10'da verilmiştir.

Çizelge 10. Denizli traverten ocağında kullanılan kesicinin çalışma ve performans parametreleri.

Kesme derinliği	3 m
Kolun ilerleme hızı	3 m/sa
Kesme miktarı	9 m^2/sa
Hidrolik tüketimi	0,15 lt/sa
Keski aşınması	250 m tül tek köşe için (1000 m tül tüm keski için)
Gres yağı tüketimi	1,3 kg/sa

Hammadde zayiatının çok az olması ve devirme işleminde kullanılan yastıklama tepeciklerine ihtiyaç duyulmaması sayesinde hafriyat işlemine gerek yoktur. Dolayısıyla işgücü ve enerji tasarrufu çok yüksektir. Zincirli kollu kesiciler ile blok üretiminde tel kesme makinasında görülen tel kopması gibi insan sağlığına zarar verecek durumlar görülmez. Bloklar yerinde ebatlandığı için blok üzerinde parçalanmalar olmaz ve çalışanları etkileyecek herhangi bir tehlikeli durumla karşılaşılmamaktadır. Zincirli kollu kesme makinasında kullanılan ray yollarının birbiri ardınca montajı ve demontajı, kesici uçların montajı ve demontajı çok kolaydır. Tel kesme makinasında olduğu beklemelere yol açmamaktadır.

Zincirli kollu kesicilerin kullanımı ile üretim hızı tel kesme makinası odaklı sisteme göre 2 kat, hammadde verimliliği ise 4 kat artmıştır. Bununla birlikte tüm bu işlemler %30 daha az işçi kullanılarak yapılmaktadır.

4 SONUÇLAR VE TARTIŞMA

Türkiye'de bulunan doğaltaş ocaklarında zincirli kollu kesicilerin kullanımı sıklıkla elmas tel kesme yöntemiyle birlikte kombine kullanım şeklindedir. Halihazırda uygulanan elmas tel kesme uygulamasına zincirli kollu kesicinin ilavesi işçilik giderlerinde düşüş sağlarken blok kazanım oranında ve üretim hızında ciddi artışlar sağlamaktadır. Zincirli kollu kesiciler Türkiye'deki ocaklarda genellikle taban kesimlerinde kullanılarak tel geçirmek için gereken delik sayısını azaltarak zamandan ve işçilikten tasarruf sağlarlar. Bu durumda düşey kesimler elmas tel kesme ile gerçekleştirilmektedir.

Özellikle fazla çatlak vb. süreksizlik içeren bej tipi doğal taş ocaklarında istenen ekonomik boyutta blok elde edebilmek için kollu kesici ile yapılan taban kesim derinliği 3,5 m ile sınırlı olduğundan 7m üzerinde basamak yüksekliklerinde çalışmak zorunlu hale gelmektedir. Devrilen bloklar sayalama ile istenen boyutlarda hazırlanmaktadır. Süreksizlik içeren ocaklarda ekonomik blokların zincirli kollu kesicilerle üretimi için daha uzun kol uzunluğuna sahip (>5 m) makinelerin kullanımı gerekmektedir.

Doğal taş ocaklarında zincirli kollu kesici ile bej türü doğal taşlarda 4-5 m^2/sa, mermerlerde 6-7 m^2/sa, traverten türü doğal taşlarda ise ortalama 10 m^2/sa üretim hızına ulaşılmaktadır.

Zincirli kollu kesicilerin kullanımı ile düzgün duvarlı, kademesiz, standart boyutlarda blok üretimi mümkün olurken yeni ocak açımında üçgen girme zorunluluğunu ortadan kaldırdığı için zamandan ve işçilikten önemli oranda tasarruf sağlanır. Elmas tel kesme yönteminin uygulandığı ocaklarda kollu kesicilerin sisteme ilavesiyle birim üretimde işçilikte %25-30 oranında azalma, blok kazanım oranında 1,5-4 kat, üretim hızında 2 kat artış görülmüştür. Özellikle az çatlak içeren sadece tabaka düzlemlerinin bulunduğu traverten ocaklarında doğrudan satışa hazır blokların üretilmesi mümkün olmaktadır. Bu tip ocaklarda blok kazanım oranında ve üretim hızlarında ciddi artışlar görülmüştür. Blok kazanım oranında % 80'lere ulaşılmıştır.

TEŞEKKÜR

Yazarlar, bu çalışmanın gerçekleştirilmesindeki değerli katkılarından dolayı Sayın Şuayp Demirel'e Demmer A.Ş. ve Demmak A.Ş.'ye, Kur Mermer ve Granit A.Ş.'ye, EGE Doğaltaş ve Traverten San.Tic.Ltd.Şti'ye ve Turan Bekişoğlu Mermer San.Tic.Ltd.Şti'ye teşekkür ederler.

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3rd Balkan Mining Congress, BALKANMINE 2009, 1-3 October, İzmir-TURKEY, © 2009, ISBN 978-9944-89-782-2

Smooth Blasting an Effective Method for Rocks Blasting with Explosives

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ABSTRACT This paper deals with the problem of damage minimization in remaining rock during breaking with explosives, using the smooth blasting technique.

The results are based upon observations and measurements, not only from our practice in mines, but also from other works which are important for designing and application of the qualitative improvements in mining openings construction.

Particular attention was paid to the determination of essential blasting parameters during the breaking process. This is related to the necessary energy transmitted in the rock mass and its effects by the explosive detonation.

1 INTRODUCTION

Stability of underground openings, as a permanent request in increasing the security for all the service time and to decrease unit costs, is realized, among others, from smooth blasting technique. This technique is used as an effective method to decrease the damage of the rock mass in underground opening contour.

Common objective of the cautious blasting is a better distribution of the explosive energy, generated in the rock mass by the explosive detonation, as to reduce stressing, fracturing and back break of the remaining rock.

2 MAIN INFLUENTIAL FACTORS

In rock fall practice by explosives, a special role plays blasting technique in respect to internal essential characteristic of explosives, that representative from necessary blasting energy effective which is transmitted to rock environment.

It is a recognized fact that in a process of excessively complicated blasting, the orientation of energy for detachment, breaking and displacement of rock mass has different and evident effects.

The interaction between explosives characteristics and rock environment it is very important. In this framework, the interpretation for parameters estimation of the energy transmitted to rock environment supports the appropriate explosives choice.

Observations and analyses of explosives application practice in underground excavations of Albanian mining industry indicate that on blasting effectiveness the main influential factors are:

1. Explosives (density, detonate velocity).
2. Explosives lading (diameter of cartridge, diameter of hole, lading ratio, sort of inciting impulse and place).
3. Rocks (density, propagate velocity of seismic waves, energy absorbent, resistance in pressure, resistance in traction, structure, etc.).
4. Dispersion of holes (distance between holes line and free surface (**V**), distance

between holes **(E)** and ratio between **V** and **E**).

3 ROCK MASS DAMAGE

Explosive falling process is associated with radial and transversal fractures of neighbor rock mass, therefore is not difficult to observe that damages and rocks cutting propagate beyond the designed size and shape of the opening.

The contour of underground opening damages, mainly in faces and roof, because in these places it is more probable and easy for the rock to fall down, which is associated with unnecessary work for lading, transportation, maintenance, stability insurance, etc.

The drilling and blasting works take, sometimes, up to 50% of advancing cycle time of some underground openings, sensible influenced in shape and dimensions of their transversal section in mine.

Direct observations and measurements in advancing process of some mining openings in Fe–Ni mines (+500 and +530 gallery Prrenjas, +380 and +440 gallery Bushtrica) have shown emphasized differences of designed and actual cross section owing to excessive explosives quantity and some other factors during ordinary drilling and blasting application (Fig. 1: a, b).

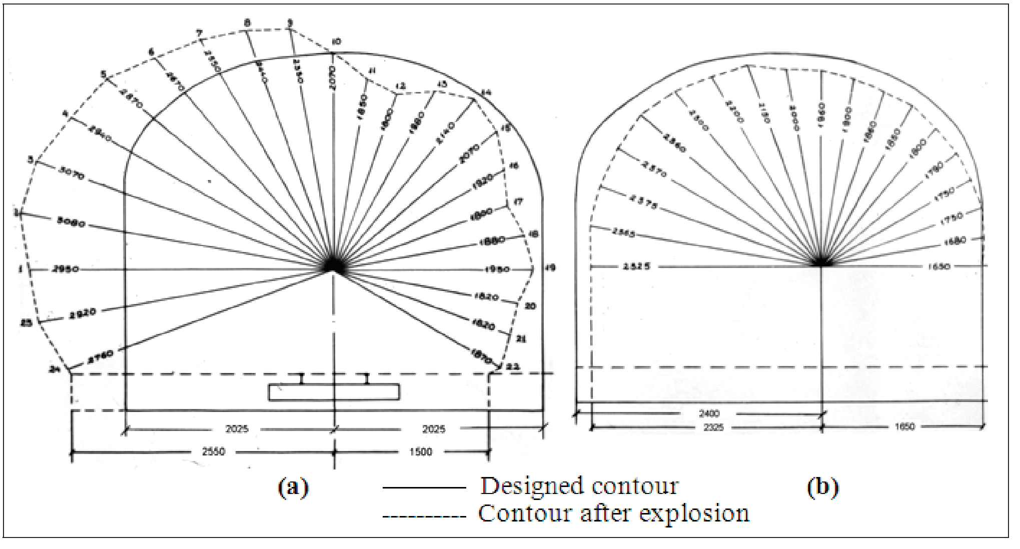

Figure 1. View of cross sections: (a) usual blasting and (b) smooth blasting.

In some sectors of Prrenjas and Bushtrica mines, values of excessive fallen material or over-digging, for ordinary drilling and blasting method, reach from 15% to 45%. Also, during construction process of tunnels Nr. 1 and Nr. 2 Klos–Bulqiza, are observed differences between designed cross section and practical one: in over-digging from +2.7% up to +18.9% and from +1.25 % to +22.4, and in sub-digging from 4.5% to 19.9% and from 2.79% to 22.5%, respectively.

Above mentioned data show that:

1. The differences between real shape and cross section size in comparison with designed ones are consequences of mistakes to order perforation execution process like placing way, depth and contouring of explosives lading, inciting kind of detonation as well as the explosive consumption.

2. A low attention is shown to take in consideration the reciprocal influence of factors like: number, depth and diameter of holes, explosive types, primary cutting method, presence or no of the air intervals into explosives lading, the order and explosion manner.
3. It is missing the evidence which will evaluate the productivity of perforation–explosion works in accord with conditions changes during mining openings advance.

4 THE DETERMINATION OF BLASTING PARAMETERS ACCORDING TO USED ENERGY

A lot of studies in some of the most developed countries have shown that when the explosive characteristics are conformable with those of rocky environment than in the rock mass is transmitted the maximal value of the energy (Berta, 1983 & 1985; Berta & Piovano, 1980**).** For that reason in composition of explosion patterns the determination of essential blasting parameters must be in function with the rock mass environment and the effects of used energy, as follow:

a) The useful energy for breaking (ε_b):

$$\varepsilon_b = \eta_b \varepsilon_t \text{, J or MJ} \tag{1}$$

where;
η_b : the breaking efficiency of explosive charge (accepted 0.15)
ε_t : the energy transmitted to the rock mass from explosion (J or MJ)

b) The energy transmitted to the rock mass (ε_t):

$$\varepsilon_t = \eta_1 \eta_2 \varepsilon Q \tag{2}$$

where;
η_1 is the coefficient that expresses the ratio between explosives characteristics and rock mass ones.
η_2 is the coefficient that expresses the ratio between the holes and explosive cartridge diameter:

$$\eta_1 = 1 - \frac{\left(\rho_{LP} D - \rho_{sh} V\right)^2}{\left(\rho_{LP} D + \rho_{sh} V\right)^2} \tag{3}$$

where;
ρ_{LP} : volumetric mass of explosive (kg/m^3)
D : explosive detonation velocity (m/s)
ρ_{sh} : volumetric mass of rock (kg/m^3)
V : seismic velocity of rock (m/s)

$$\eta_2 = \frac{1}{e^{\frac{\phi_b}{\phi_{ng}}} - (e-1)} \tag{4}$$

where;
φ_b : holes diameter (m)
φ_{ng} : cartridge diameter (m)
e : 2.72

while ($\varepsilon \cdot Q$) represents potential energy developed by explosion and in particular:
ε : specific energy of explosive (MJ/kg or J/kg)
Q : explosive quantity (kg)

According to equations 1 and 2, the breaking energy is:

$$\varepsilon_c = \eta_c \varepsilon_t = \eta_1 \eta_2 \varepsilon Q \tag{5}$$

The energy of specific surface (ε_{ss})

$$\varepsilon_{ss} = \frac{\varepsilon_b}{V_{sh} \cdot S} \tag{6}$$

where;
V_{sh} : fallen rock volume ($S_h\, l_p$ = m^3)
S : formation new surface referred to volume unit, expressed by:

$$S = \frac{64}{D_M}, \quad m^2/m^3 \tag{7}$$

D_M : maximal dimension of fall (m) that is accepted according desirous degree of breaking, usually from 0.15 to 0.25.

c) The specific explosive consumption (C):

$$C = \frac{Q}{V_{sh}} \tag{8}$$

From equations 5 and 6, have:

$$C = \frac{64}{D_M} \cdot \frac{\varepsilon_{ss}}{\eta_c \cdot \eta_1 \cdot \eta_2 \cdot \varepsilon} \tag{9}$$

d) The distance between holes and free surface, or the distance between the hole rows (V), it is given by:

$$V = \phi_{ng} \cdot \sqrt{\frac{\rho_{LP} \cdot \pi}{4 \cdot C}} \tag{10}$$

e) The distance between holes, inside rows (E), is given by:

$$E = 0.8\,V \tag{11}$$

In a correct delineation of the opening cross section and especially the faces and the roof from smooth blasting, a significant effect has the behavior manner of cutting from hole to hole, therefore the determinations must be supported in equilibrium equation between pushing action of gases on hole faces (P_s) and tensile strength of rock, or rock mass (σ_t), as follows:

$$\left(\frac{\phi_{ng}}{\phi_b}\right)^2 \cdot \frac{L_{ng}}{L_b} \cdot \Delta_{LP} \cdot P_s \cdot \frac{\phi_b}{E_{ngk} - \phi_b} = \sigma_t \tag{12}$$

The solution of equation (12) enables the determination of distance between contour holes in (m), as:

$$E_{ngk} = \phi_b + \left[\left(\frac{\phi_{ng}}{\phi_b}\right)^2 \cdot \frac{\Delta_{LP} \cdot P_s \cdot \phi_b}{\sigma_t}\right] \cdot \frac{L_{ng}}{L_b} \tag{13}$$

where;
ϕ_{ng} : explosive cartridge diameter, (m)
ϕ_b : hole diameter (m)
L_{ng} : explosive cartridge length (m)
L_b : hole length (m)
Δ_{LP} : explosive specific density
E_{ng} : distance between contour holes (m)
σ_t : tensile strength of rock (MPa or Pa)

In further determinations will take place the spreading of holes in cross section according to the role they will play in each group (with determined number and loading). With special importance are and other indexes as hole using coefficient ($\eta = L_p/L_b$), fallen rock volume ($V_{sh} = S_h\ L_p$, in m^3), blasting caps consumption ($C_{det} = N_b/V_{sh}$, Caps/m^3), drilling coefficient ($F_{sh} = L_{shp}/V_{sh}$, m/m^3).

The data received from experiments in above mentioned mining openings showed that the success of rock blasting method is related also with two other factors, such as:

First, the recommended explosive charges for immediate explosion must be, in quantity, around 3 to 5 kg for each group.

Second, the delay interval between explosion groups (cutting, fall and contour) must be within the limits of 25 to 100 ms in order that vibrations level to decrease in minimum limits, perhaps less than 10 mm/s.

5 THE POSSIBILITY OF DAMAGE MINIMISATION IN ROCK BY SMOOTH BLASTING

During charge explosion the adjacent rock mass and rock environment is damaged, but exist the possibility to reduce the undesirable effects by taking more and continuous care in projected phase and especially during application one, in conformity with real conditions of mine.

In real conditions of mining openings where we performed the observations and measurements, the elimination of over or sub excavation was achieved by smooth blasting, taking care also to a careful and technical discipline to accomplish drilling and blasting operations which evidently improved the quality of mining openings construction.

In “smooth blasting” method the contour holes charges incited at the end of blasting process, on purpose to use the free surface preliminary created from previous charges explosions (Fig. 2).

The larger sizes of rock pieces, falling from contour zone, comparable with those from previous charges, are because of a smaller distance (E_c) between contour holes than the distance between contour holes raw and previous one (V_c),that is to say $E_c<V_c$ (Fig. 2&3). The tests of smooth blasting to the faces of some mining openings are accomplished by contour cutting after blasting of center part, using extended charges with and without air intervals in the

presence of wood slats as screen of detonations energy. Blasting of explosive charges accomplished by fuse and electric method and electric caps with milisecond slowness (25–30 ms) and reverse initiation (Fig. 4).

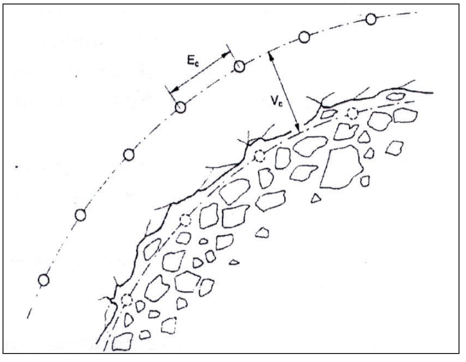

Figure 2. Free surface with distance (V_c) from contour mine holes created from previous charges explosion and (E_c) the distance between contour holes.

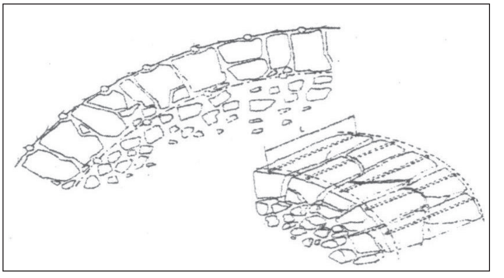

Figure 3. View of rock pieces falling from contour charges explosion.

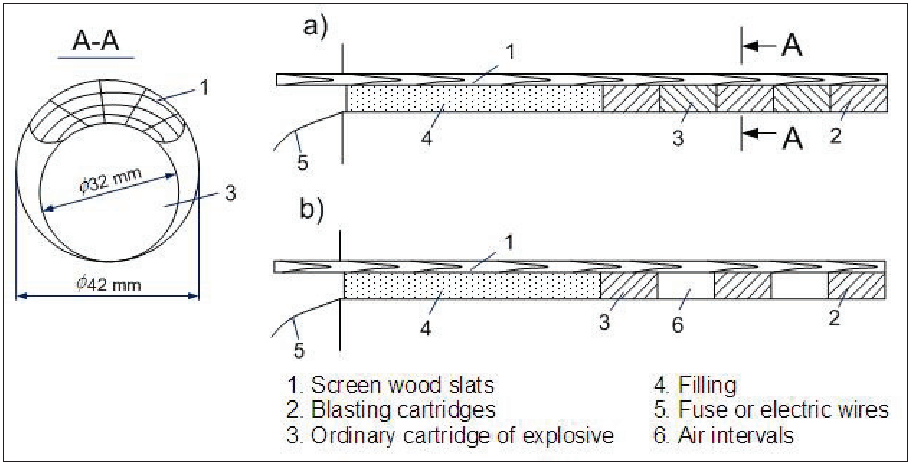

Figure 4. View of explosive charges construction of contour holes for smooth blasting:

a) Extended charges – column without air intervals,
b) Extended charges – column with air intervals.

Taking in consideration during calculations: the distance between holes, the number of contour holes, the distance between holes and free surface (minimal resistance line), the concentration of explosive charges, the explosive type and the incitement kind of detonation, etc., influenced evidently the real cross section of mining openings to approach the design one (in average values from 3 to 5% over excavation) and for a more regular shape (Fig. 5).

6 COMPARABLE VALUES OF ROCK BLASTING INDEXES

Some of the rock blasting indexes during the application of usual and smooth blasting in the same condition of rock environment represented in Table 1, show that the best values are for smooth blasting which enables also more work safety (Nako, 1985 & 1987) because the danger of rock blocks detachment from faces and roof of mining openings is decreased.

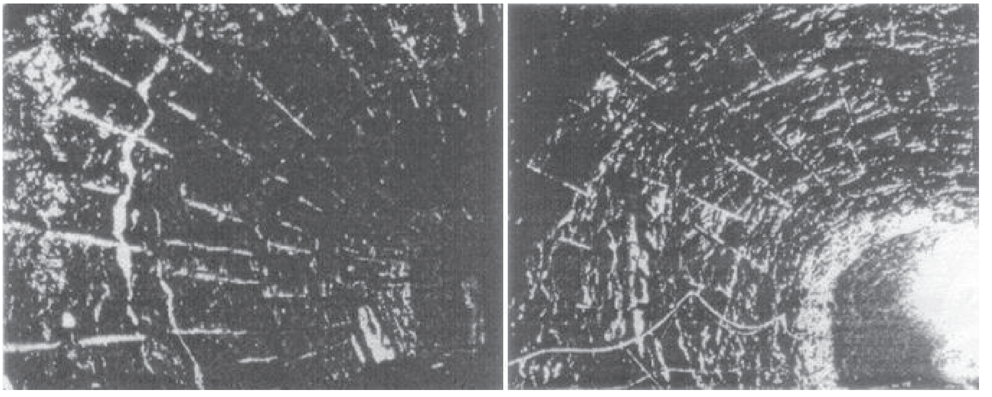

Figure 5. View of tracks of perforated holes after smooth blasting.

Table 1. Indexes of rock blasting for usual and smooth blasting.

Index Designation	Usual blasting	Smooth blasting
Depth of holes (m)	1.2	1.4-1.5
Incitement method	right	reverse
The coefficient of explosive lading	0.55-0.6	0.46-0.5
Specific consumption for f=6-7, kg/m^3	2	1.4-1.5
Specific consumption for f=8-10, kg/m^3	2.7-3.23	1.9-2.4
Holes utilization coefficient for f=6-7	0.8	0.96
Holes utilization coefficient for f=8-10	0.65	0.87-0.9

Factual data related with the minimum reduce of fallen material in limits of projected contour borders as well as the minor damages of rock mass, through a smooth surface with lower explosive consumption and with better technical and economical indexes are weighty reasons for smooth blasting efficiency which must be used more widely in our mines.

7 CONCLUSIONS

In designing and application process of rock blasting by smooth blasting method visible influence have reciprocal relations between rock environment and explosive charge characteristics which are represented in useful and useless effects.

Smooth blasting enables the rock mass damages reduction and an increase security and stability of mining openings contours as well as better technical and economical indexes by improved utilization of released energy during blasting process.

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Controlled Blasting with Demolition Agent

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ABSTRACT Over the last decades of last century and this years the interest in using non-explosive demolition has increased again. Several companies started producing various agents with various names and similar characteristics, but with the same conditions of use.

Unlike demolition by blasting, this method is environmentally friendly. The operation takes place quietly, with no seismic affects, dust, gases and no flying rocks. Rock cutting is done as needed and in the desired scale, with-out affecting the other rock mass or surrounding structures. In non-explosive blasting there are no work breaks, no evacuation of workers and machinery, and special safety at work, special requirements in traffic or other safety measures. These advantages have raised the issue with experts in the field of use and improvement of various agents or slurries for controlled cutting and demolishing of rock masses and structures. The first part of this paper points out the principles in the use of such agents, some properties and several examples for practical use (photos) in open pits for breaking of hard rock masses and demolishing concrete elements and construction facilities.

1 INTRODUCTION

NERS is an agent – a powder mixture composed of several chemical compounds with pronounced alkali properties and their expanding while mixing with water. The mixture possesses very high expansive power with high strain under pressure of over 50 MPa and expansive operating capacity of 18 000 lb/inch^3.

These values vary based on the company that has produced the agent.

The major component part comprising NERS are iron oxides (2.5%), silicium (5.1%), aluminum (1.6%) and calcium (89.6%).

Its specific weight, compared with that of water, is 3.2 times higher, melting point amounts to 1 000^{o}C, it is easily dissolved in water, with no odor and pronounced grey colour.

The time limit for use is 6 months, and when properly stored and kept with no moisture, it can be used for one year or more.

The consequences of inadequate use are the same as quicklime or cement. When operating, rubber gloves, goggles and dust proof mask are recommended.

1.1 Conditions for Use

Because of its expansive characteristics, it is used in controlled cutting, breaking and demolishing of AB logs and slabs, large stones, marbles, granites, limestone or any other material (fig. 1). It is simple to use and process. It is mixed with water and then holes, natural cracks or slits are filled.

NERS is a non-explosive agent and is safer than other explosives, does not cause quakes, vibrations, there is no rock flying and does not produce toxic gases. It achieves good results and is cost effective compared with other methods. There is no need of special permit for operation. Only equipment and experience are needed.

This is so called environmentally friendly product with MSDS factor of safety.

Breaking, demolishing and cutting are done according to a schedule of drill holes with earlier defined parameters regarding distance, the diameter and amount of slurry to be used.

The operation in cutting lamellae is done safely, simply and easily taking in consideration the controlled expansive breaking. NERS can also be used to make very thin slabs or onyx blocks, marble, granite and other kinds of stone suitable for processing.

Figure 1. Results of the use of NERS in granite blocks.

It can be used in areas where explosives cannot be used. Cleaning up the site is safer, faster and easier if one respects the rules for work environment and the surrounding. The impact of these products to the surrounding is obvious: there is less material to throw away, no cleaning is required, and after use there is no smoke or any chemicals left.

All this makes it possible NERS to be safe to work in closed premises, where it does not collect dust and the possibility of dust contamination is excluded.

When the use of explosive is required, it can lower operation costs for breaking and at the same time increase safety since it is used for cracking of weaker structures. It allows the use of less explosive and provides safer breaking or fall of the structure.

NERS is used for: Breaking of various rocks (marble, granite, onyx, limestone), demolishing of massive reinforced concrete structures, demolishing of bases for placing of heavy machinery, demolishing of supports, logs, walls or slabs in bridges, excavation in tunneling, underground demolishing, various concrete structures, shafts, pools etc.

1.1.1 Manner of use

NERS mixture is poured into earlier prepared holes according to schedule and depending on the general condition of the rock or the structure (location, physico-mechanical and structural characteristics).

The technique of performance, the number and manner of drilling the holes is conditioned by the characteristics of the material to be obtained or finalized, the appearance and shape of free (open) surfaces, the desired shape and the scale of block excavation for further processing.

NERS packing can be different depending on the producer. Most often it is packed in polyethylene sacks of 20 kilos each. Another way of packing for delivery is in waterproof boxes the size 15” x 11”, each box containing 4 plastic packages of 5 kilos each.

Drilling

Depth of drill holes should amount to 80 – 90% of the block thickness, lamella or concrete element. The distance between holes should be determined based on resistance strength to the cutting of rock, concrete etc.

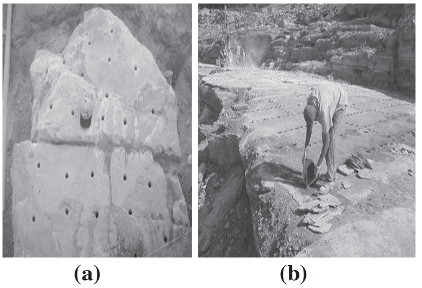
Holes are drilled by a drilling hammer or other drilling tools used for rocks and concrete. It is recommended the holes to be the diameter of 30 – 40 mm, depending on the type of NERS, to maximum of 50 mm (fig. 3).

In the first use of the technique, experimental drilling series of various diameters and distance are done.

Determination of optimal diameter and distance between the holes is done based on the quality of cut surfaces that have been obtained and their general appearance.

In practice, the distance between holes amounts to 10 – 30 cm depending on the drilling diameter, NERS type and the material drilled.

In enforced concrete the distance amounts to 20 cm, in hard compact rocks it amounts to 15 – 20 cm, whereas in rock of primary, visible direction of cutting the distance can amount to 30 cm or more.



(a) (b)

Figure 3. Use of NERS in given conditions, (a) drilling a hole, (b) filling of hole.

1.1.2 Preparing NERS for filling

In a large vessel the powdered agent is mixed with 30 – 40% water so that the slurry obtained is suitable for filling and handling. The ratio between water and powdered NERS is 1:3.

For obtaining better coherence of the slurry, a mixer, concrete cement or shovels can be used. For example for a sack of 5 kg NERS (11 lb.) 1.5 l (0.39 gallons) of common water is added. It is mixed as long as the mixture of water and powder becomes slurry.

Before filling starts, holes should be cleaned up from dust or small pieces by a blower. NERS slurry should be filled within 10 to 15 min after mixing. This means that its properties are valid for a certain period of time. This is one of the advantages of the slurry since this space period allows additional preparations and safety in all further steps.

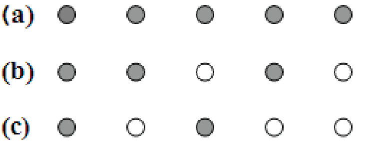


Figure 4. Geometry of possible patterns of holes and manner of filling; (a) all holes are filled; (b), (c) empty holes are left as buffer zone.

Owing to the chemical composition and the amount of certain component parts this period differs depending on the type of NERS.

For some NERS types this space period can last up to 1 hour. After this, it says in the technical characteristics, that the NERS jelly – slurry heats up (its temperature increases), gradually hardens and is in the phase of swelling or its volume increases.

In the three dimensional area in the hole, after this period, it causes significant dynamic strain overcoming boundary strength, first of all that of pressure and extension of the rock or material, causing, in that manner, rock cutting, breaking or other kind of deformation, depending on the hole pattern, parameters of the method and the shape of the rock massif or element.

The holes must not be filled to the top, but 2.5 cm lower than the surface. Before filling, the slurry is mixed and poured into the holes. During filling, no air gaps should be present in the holes. For filling of horizontal holes a grouting pump is used, whereas clay or well-prepared plugs for stopping are required.

The effects can be seen after 30 – 40 minutes, depending on the weather conditions, temperature, humidity and hardness of rocks or concrete.

Cracking of the material can be noticed after 2 hours, and complete action is fulfilled after a period of 24 hours, which is the maximum expanding time.

One package of 20 kilos of powdered NERS is enough to fill a hole pattern of 10 m, the diameter of 38 mm. It should be mentioned that it is not always necessary to fill all holes.

1.2 Effects of NERS Action

After filling, one should not look closely and directly at the holes. The type of NERS should be selected depending on the temperature. For better results it is important to know the temperature of the material and the air. This is so because rock or concrete temperatures can differ from that of the air.

In summer the best time to use NERS is early in the morning or evening when the temperature of the material is low.

Material with high temperature may cause explosion. In summer the hole must be covered with tarp to protect it from sunlight in order to avoid possible explosion. It can be done with tarp or wet hay. In summer, some ice can be put in the material.

In winter NERS is mixed with 1% calcium chloride. If the slurry gets dry and does not have any effects, some more common water is poured into the filled holes to neutralize the action.

Figure 5. NERS effects in different materials and conditions.

2 CONCLUSION

The material for non-explosive controlled breaking, cutting and demolishing for our conditions and use has been translated as NERS (non-explosive demolishing means). In Bulgaria it is known as BRS – a type of Bulad. It is known as NEDA in Great Britain and the USA. In the Republic of Macedonia its use has not been allowed since the material has no permit or license for use (the procedure is in progress).

The high quality and safety in its use speaks for its high efficiency and possibilities for application in industry, urban environments and domestic conditions and, generally, in many difficult and complicated conditions of operation.

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New Ecological Technologies for Hole Blasting in Opencast Mines and Quarries

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ABSTRACT At present, a considerable amount of blasting operations is made in the opencast mines and quarries. As a result considerable gas and dust emissions violently polluting the environment are evolved.

A new technology for hole blasting without the holes to be deepened and in combination with hydro-physical blast is developed in Bulgaria with the purpose of decrease in the harmful influence.

By the research it is found out that the new technology to a considerable extent decreases the outgo of explosive needed for the destruction of $1m^3$ of rock mass and also there is a decrease in the harmful gas and dust emissions. Apart from that, an excellent disintegration of the material and shaping of the low bench are obtained.

The developed new technology is in process of introduction in practice in several quarries in Bulgaria with real ecological and economical effect.

1 INTRODUCTION

Mining works at the opencast mines and quarries are predominantly executed by borehole-blasting operations according to the method of borehole blasting. Different types of explosives are used – mainly roughly-dispersed explosives of the type ANFO, roughly-dispersed TNT-containing ammonites, emulsion and water-saturated explosives. The boreholes are mainly of diameters from 100 to 250mm and even more and the height of the mine benches is from 10 to 20 rarely 30 m. A simultaneous blast covers 1-15 to 50 and even more tons of explosive.

Furthermore, the opencast mines and quarries are usually established at a distance of 1000 to 5000 m aside from urban places and/or other facilities.

High intensity of mining and method applied as well as the organization of the works bring to significant gas-dust emissions into the atmosphere, which heavily contaminate the environment. Important are also the impact of ground vibrations, the detonation stroke and the fly of rock fragments, as well.

The topic of harmful effect of the blasts on the environment has recently got special importance and has attracted attention.

Several series of standards have been issued in the European Union, which aim to provide the requirements to explosives for civil use (Directive 93/15 EEC). One of those standards refers to the method for analyzing and investigating the toxic gas emissions, liberated from explosives, which are allowed for use (EN 13631-16). This standard does not define the norms for emission of orange nitrous gases and the carbon monoxide and carbon dioxide, which should be defined by each country.

An important issue related to the harmful effect of blasting operations is the assessment of rational charge density of blasted massif with explosives or the so called relative consumption of explosives for destruction of 1 m^3 of rock mass. The analysis has revealed that there are many different reasons, for which the rock massifs are usually over-charged with explosives, which brings to higher quantities of toxic gas emissions.

The above necessitated the design of new technologies, including new technologies for hole blasting.

2 NEW TECHNOLOGY FOR HOLE BLASTING

Recently a technique has been developed, which eliminates the sub-grade drilling of boreholes beneath the level of the lower bench and leaving an air gap at the hole bottom of 0.9 m (Chiappetta, 2004). With regard to the above a technology has been developed and a special device for leaving an uncharged hole bottom has been designed, which is called Power Desk™.

The device, designed by Frank Chiappetta is shown in Figure 1.

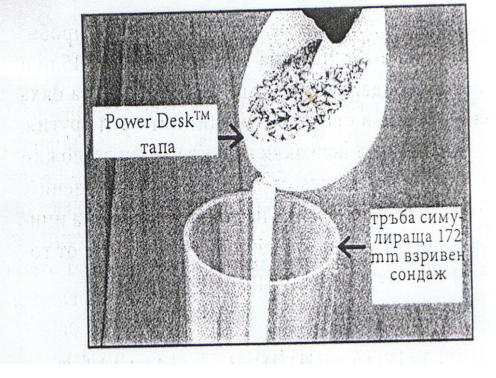

Figure 1. Wooden deck with a plug for providing an air gap, according to Frank Chiappetta.

The investigations carried out in the Republic of Bulgaria, at the "Klokotnitsa quarry, in particular, indicated that the application of the technology and the device are labor-consuming and, practically, their use is very difficult.

Further to systematic research a mobile technology and a special device have been designed for the uncharged part of the borehole. A flexible device is used, which is fixed and fastened at a respective height of the borehole and the charge of the explosive is set above it (Kamburova, 2009)

Parts of the designed device and the approach of plugging it into the boreholes are shown in Figure 2.

Figure 2. Parts of the designed flexible device and the approach of installing it into boreholes.

Series of investigations have been performed by the above technology and the designed device. The investigations comprised blasting of boreholes of 15 m depth, 105 mm diameter, air gap at the borehole bottom of 1 m, while the boreholes are positioned in a grid of 3.5x3.5 m. The investigations revealed the following:

1. The blasted rock material in the field with no sub-grade drilling of boreholes is very well fragmented and it does not differ from rock material from the field, blasted according to the old technology without sub-grade drilling of boreholes of 1m beneath the lower level. The prevailing size of rock fragments is 30-40 cm.
2. The level of the lower bench does not have any rises, it has an appropriate shape, and in addition, its upper layer is not disturbed, which will have a favorable effect on the next drilling.

3. The relative consumption has been reduced with 15–17 % for the explosives and more than 5 % for the drilling, which means significant cost-reduction.
4. The reduced consumption of explosive means significant reduction of toxic gas-dust emissions.

The results obtained provide the reasons that operations at the "Klokotntsa" quarry will be from now on carried out according to the new technology without sub-grade drilling of boreholes and leaving a 1 m air gap at the borehole bottom. Pressure and/or kinetic energy may be used to explain the events at the borehole bottom of the new system. When the explosive detonates in the blasted borehole the released high temperature products direct to the lowest resistance location – the air gap at the borehole bottom. When the initial front of the impact wave strikes the borehole bottom, the velocity of the wave is reduced, it is reflected by the borehole bottom and increases pressure at that point. The secondary strike of the products from blasting adds one more push to the borehole bottom. With regard to that combined effect the pressure P_2 at the borehole bottom may be increased more than twice, compared to the initial pressure P_1, thus generating an air-physical blast. The action of the above air-physical blast is shown in Figure 3.

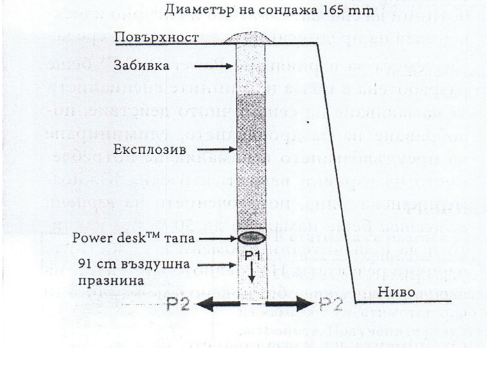

Figure 3. Pressure at the borehole bottom without sub-grade drilling and with an air gap.

The increased pressure is enough to cause splitting and breaking of the borehole bottom into two directions. In fact, the initial energy of the stroke wave and the secondary energy of blasted products are much more effective than the concentrated, continuous cylindrical charge at the borehole bottom, but only when the air gap has a specific length, defined for given conditions and a given blasting setup.

Additional investigations with another carrier of the generated physical blast, namely water, have been performed with the objective of a better reduction of toxic gas-dust emissions.

3 TECHNOLOGY WITHOUT SUB-GRADE DRILLING OF BOREHOLES WITH A HYDRO-PHYSICAL BALST AT THE BOREHOLE BOTTOM

The analyses, carried out to specify the quality of rock massif at the opencast mines and quarries have shown that in many cases operations are performed in partially or completely watered areas. In those cases, the preparation of an air gap at the borehole bottom, according to the new technology, seams difficult. Furthermore, it is well known that water is successfully used as an effective carrier of physical blast. That is the reason for a research on the application of a technology without sub-grade drilling of boreholes in a conjunction with a hydro-physical blast in the 1 m uncharged section at the bottom of the boreholes.

The research was performed at the same "Klokotnitsa" quarry in order to be comparable. The accepted diagonal setup for blasting with a non-electrical technique has been applied.

In order to establish the difference between the air-physical blast and the hydro-physical blast at the borehole bottoms, the blasting field, prepared according to accepted at the "Klokotnitsa" quarry blasting setup, has been divided into two sub-fields, and they have been blasted simultaneously (Figure 4).

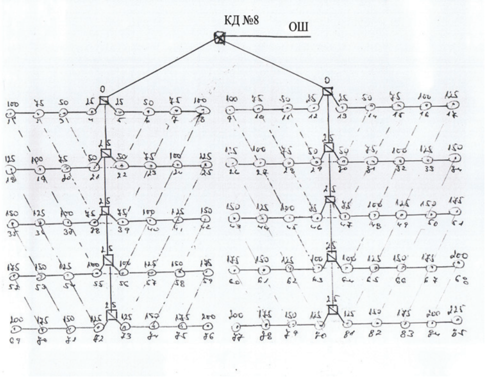

Figure 4. Scheme of the test blasting field at the "Klokotnitsa" quarry.

In the left-hand part of Figure 4 the field has been charged according to the new technology without sub-grade drilling and with an air gap of 1 m, made by a flexible device, according to Fig. 2. In the right-hand side instead of air gap a water cartridge of 100 mm diameter and 1 m length is inserted.

The insertion of the water cartridge at the borehole bottom is shown in Figure 5.

Figure 5. Preparation of uncharged section with a water cartridge and its insertion.

The gas-dust emission, the fragmentation of the rock mass and the level of the lower bench have been monitored during the test blasting.

Figure 6 shows the gas-dust emissions, caused by the blast in the two sub-fields in the beginning and at the end of the process.

Figure 6. Gas-dust emissions, caused by blasting according to a technology without sub-grade drilling. To the left – an air-physical blast, to the right – hydro-physical blast.

It has been established that much smaller gas-dust cloud has been emitted at the sub-field of a hydro- physical blast, compared to the sub-field of the air-physical blast, under all other equal conditions. That effect has been achieved under conditions of equal relative consumption of explosive in both of the test sub-fields.

The fragmented rock mass, obtained after blasting of both sub-fields is shown in Figure 7. It shows that the rock mass is equally well fragmented.

Figure 7. Fragmented rock mass obtained after blasting of the two sub-fields.

The level of the lower bench after excavation of the blasted rock mass is shown in Figure 8.

It shows that the bench is very well prepared and does not have any rises.

Figure 8. Level of the lower benches of the two blasting sub-fields.

4 MAJOR CONCLUSIONS AND FINAL COMMENTS

The following major conclusions may be derived from the research:

1. When borehole blasting is performed at the opencast mines and quarries large quantities of 50 and even more tons of roughly dispersed explosives are blasted simultaneously. With regard to the above significant quantities of toxic gas-dust emissions are liberated in the atmosphere.
2. The issue of the harmful impact of blasting operations on the environment has acquired special importance and attention in the European Union and the other developed countries and it is a subject of new requirements with regards the norms and standards.
3. The design of new technologies for borehole blasting and the reduction of consumption of explosives cause the reduction of toxic gas-dust emissions. Blasting without sub-grade drilling and leaving an air uncharged section of 0.9 – 1.0 m at the borehole bottom is a technology of that kind.
4. The commissioning of the new technology without sub-grade drilling and preparing an air gap by a mobile flexible device has brought to more than 15-17% reduction of the relative consumption of explosives more than 5 % reduction of consumption of boreholes. Good fragmentation of the rock mass has been achieved; the lower bench is undisturbed and has a good form and gas-dust emission have been reduced.
5. The new technology without sub-grade drilling and an air gap at the borehole bottom is difficult for use for partially or completely watered blasting fields. Under the above conditions the effects of hydro-physical blast may be used.
6. The test blasting has indicated that gas-dust emissions are significantly reduced compared to the air-physical blast, when a water cartridge is inserted in the uncharged section of the borehole bottom and all the other conditions are equal. In addition, the rock mass has been well fragmented and the lower bench has a good shape and it is undisturbed.
7. The technology without sub-grade drilling and the use of hydro-physical blast may be successfully applied for blasting of partially and heavily watered blasting fields with an unquestionable environmental effect.

5 CONCLUSION

The research and test blasting has undoubtedly revealed that the hydro-physical blast at the borehole bottom and the use of the new technology without sub-grade drilling is more rational than the air-physical blast according to the same technology. The hydro-physical blast is practical, easily applicable and it can successfully se used in dry blasting fields. When watered boreholes are blasted, the room, required by the hydro-physical blast shall be defined by qualified experts. Water cartridge is not required for that purpose, but water column may be regulated by the flexible balloon, designed for the technology with the air-physical blast.

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Mining Solution for Works on a Pressure Penstock in the Pumped Storage Hydropower Plant Avce- Slovenia

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ABSTRACT Company SENG d.o.o., the member of Holding of Slovenian Power Plants invests in construction of the first pumped storage hydropower plant in Slovenia. The location of pumped storage hydro power plant Avče (Avče PSP) is in western part of Slovenia. The powerhouse is situated on the left bank of the river Soča.

In start of project the open-air penstock from the upper storage reservoir on the hill Kanal peak must go from the point T1 to T11 on the surface to the river Soča. After several months of geological investigations, the open-air penstock had to be rerouted underground between T6 and T11. Several alternatives have been investigated and finally the Investor chose the so-called mining alternative IIA for the various advantages it offers compared to the other ones. It foresees a 188m high Vertical pressure shaft followed by a 385m long sub-horizontal pressure tunnel.

This paper will resume a hard mining works on that vertical shaft in very poor rock and unstable material, the excavation works, the rock support measures, the problems with water flows in the shaft and the ventilation. The part of the works was also a 190m drilling well in diameter 0,5m for a material transport and dewatering.

Because of the problems in the shaft a part of it was done with an Alimak work platform. The works in vertical shaft finished in December 2008. With this engineer article we will present a mining solution that helped that the civil works on the pumped storage hydropower plant project go on.

1 MINING SOLUTION FOR WORKS ON A PRESSURE PENSTOCK IN THE PUMPED STORAGE HDROPOWER PLANT AVČE - SLOVENIA

1.1 Generalities

During periods when the prices of electricity are low (nights and weekends), the pumped storage hydropower plant Avče (Avče PSP) will pump water into the upper storage reservoir; it will generate electricity during periods when electricity prices are high (peak demand).The pumped storage hydro power plant Avče powerhouse is situated on the left bank of the river Soča, downstream from the village of Avče. Basic characteristics:

- Installed capacity in turbine operation 185 MW
- Installed capacity in pump operation 180 MW
- Annual average energy prod.,approx 426 GWh
- Annual average energy consumption, approx.553 GWh
- Power plant utilization rate 0.77

The other objects of pumped storage hydropower plant Avče which we didn't describe in the paper are:

- **Upper storage reservoir**

Maximum headwater level 625 m asl.
Minimum headwater level 597m asl.
Effective storage capacity 2.17 million m^3

- **Lower storage reservoir**

Maximum tail water level, continuous operation 106.0 m asl.
Minimum tail Water level, continuous operation 104.5 m asl
Useful daily reservoir volume 0.42 million m^3

- **Penstock**

Inner diameter 3.3-2.6 m
Total length 1470 m

- **Headrace Tunnel**

Length of approximately 685m

- **Surge Tank**

97m long and a vertical shaft of 37m in

- **Powerhouse shaft**

The Powerhouse shaft is 20,00m wide and 80m deep.

- **Sub-horizontal pressure tunnel**

385m long sub-horizontal tunnel

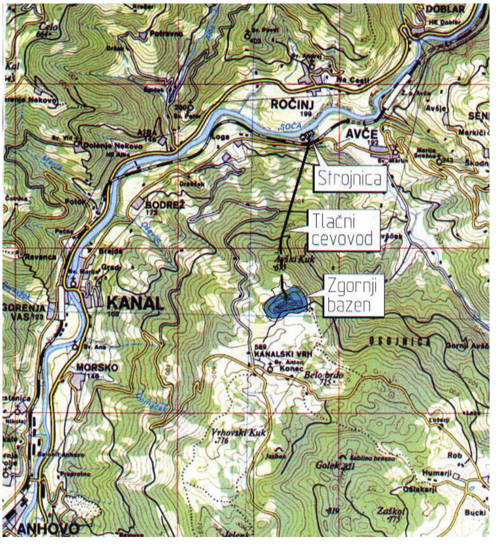

Figure 1. Location of the pumped storage hydropower plant Avče.

1.2 Vertical Pressure Shaft

After several months of geological investigations, the open-air penstock had to be rerouted underground between T6 and T11. Chose mining alternative IIA foresees a 190m high vertical pressure shaft followed by a 385m long sub-horizontal pressure tunnel. The situation is on Figure 2.

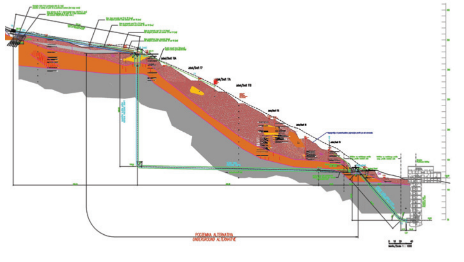

Figure 2. Profile of the underground penstock with a vertical pressure shaft (190m) and horizontal tunnel (385m).

The vertical pressure shaft can be divided into 2 steps: (1) un upper part with a depth inferior to 36m and a diameter of 8m and (2) a lower part with a depth between 36m and 190m with a diameter of 4.00 to 4.40m.

- <u>Upper part (depth < 36m)</u>

Three typical profiles / rock support measures have been developed in function of the expected rock conditions.

Table 1. Type of rock support as for rock quality.

Rock Quality	Rock Support
Weathered rock	Type 1
Creeping soil	Type 2
Unstable material (completely crushed)	Type 3

For each profile, a basic rock support has been pre-defined. It consists of a reinforced shotcrete layer with a corresponding mesh of grouted rock bolts. Some additional measures with steel ribs should be applied when the rock conditions are poor to very poor. The typical profiles are as follows.

Table 2. Support measures in upper part (depth < 36m).

Profiles	Shotcrete thickness (cm)	Grouted Rock Bolts		
		Dia. (mm)	Lengt. (m)	Number of rock bolts (pces)
Type 1	8-10	25	2.00	10
Type 2	20-25 with steel fabric	25	2.00	12
Type 3	20-25 with 1-2 layers steel fabric	25	4.00	Spot rock bolts

Each round consists of excavation work with drilling and blasting and rock support.

Figure 3. Support measures profile Type 1.

- <u>Lower part (depth > 40m to 190m)</u>

The shaft sunked by the application of the »strip and line metod« using hand held drill hammers. Each round consists of excavation work, cleaning of shaft bottom and rock support. Excavation work was done carefully and by the application of smooth blasting methods to minimize rock disturbance.

Four typical profiles / rock support measures have been developed in function of the expected rock conditions.

Table 3. Type of rock support as for rock quality.

Rock Quality	Rock Support
Very good /good rock	Type 1
Fair rock	Type 2
Poor rock	Type 3
Very poor rock	Type 4

For each profile, a basic rock support has been pre-defined. It consists of a reinforced shotcrete layer with a corresponding mesh of grouted rock bolts. Some additional measures with steel ribs should be applied when the rock conditions are poor to very poor. The typical profiles are as follows. The typical profiles are as follows.

Table 4. Support measures on lower part (depth > 36m to 190m).

Profiles	Shotcrete thickness (cm)	Grouted Rock Bolts		
		Diameter (mm)	Length (m)	Number of rock bolts (pces)
Type 1	6-10	25	2.00	8
Type 2	8-10 with steel fabric	25	2.50	8
Type 3	10-15 with 1 layers steel fabric	25	2.50	10
Type 4	15-20 with 2 layers steel fabric	25	3.00	Spot rock bolts

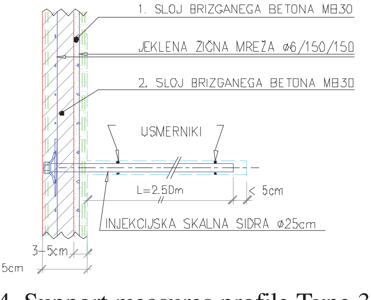

Figure 4. Support measures profile Type 3.

1.2.1 Making an advancewelle, pilot drilling

Work on Vertical pressure shaft started with an advance hole, pilot drilling diameter 444,15mm and reaming to diameter 650mm.

The work stated in 14.11.2007 and finished on 14.03.2008. After the reaming of advance hole the all pilot drilling was protect with a pipe in diameter of 508mm.

Figure 5. Drilling equipment FRASTE FS400 on the location of vertical shaft.

During the drilling we had a lot of problems with a lost of drilling fluid. The geological setting of the area comprises one stratigraphic formation, namely upper cretaceous flysch. However, the flysch itself, defined as cyclic or chaotic undersea deposits of shales, marls, silt/sandstones and limestone's, is a highly heterogeneous sedimentary formation with abrupt changes of litho logy. But we didn't have geological and hydro-geological information about conditions after 50m below surface.

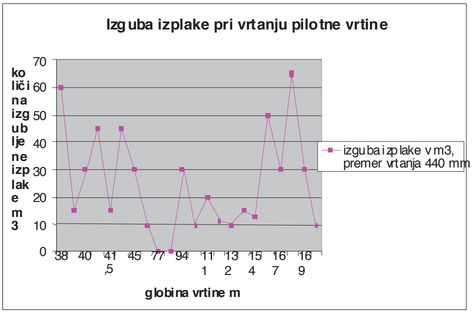

Figure 6. The lost of drilling fluid m^3 during the drilling with diameter 444,15mm on the different depth.

The advance of the pilot drilling hole during the drilling was different. The maximal advance with a diameter 444,15mm was 23m/day with a diameter 650mm was 28m/day. The all advance during the drilling we can see in the Figure 7 and 8.

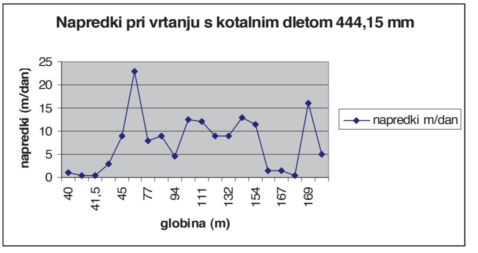

Figure 7. The day advance of the drilling diameter 444,15mm on the different depth.

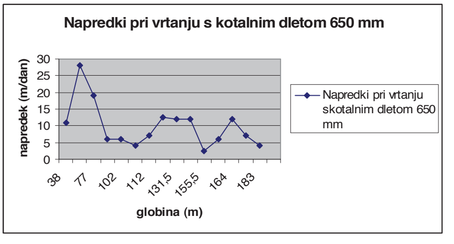

Figure 8. The day advance of the drilling diameter 650mm on the different depth.

The lost of the drilling fluid in the hole we saved with a cement grout and redrilling of the hole. The quantity of the cement grount who was grouted in the hole after the lost of the drilling fluid we can see in the Figure 9.

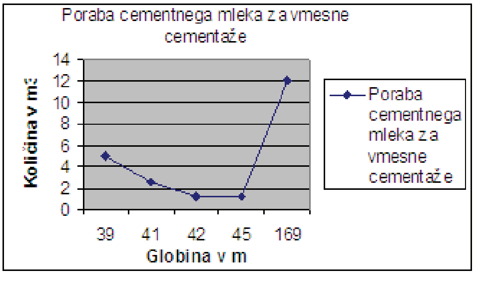

Figure 9. The quantity of the cement grount in m^3 who was grouted in the hole after the lost of drilling fluid in the different depth.

After the reaming of advance hole the all pilot drilling hole was protect with a pipe in diameter of 508mm.

Figure 10. The pipe in the well how it was on the intersection of vertical and horizontal tunnel.

The main duty of the hole was to transport the all excavation material from the bottom of the vertical shaft to the horizontal tunnel, for dewatering and ventilation.

1.2.2 The excavation works in the vertical shaft

The excavation works has been proceed by steps which height was depends on the rock quality. In poor to very poor conditions, the height was reduced while it was increased in far to good rock. In general the excavation height wasn't exceed 2,0m. Each step was consist of a typical excavation round with drilling, mucking, rock support according to the detailed corresponding attached drawings. The rock surface must be fully supported before proceeding with the next excavation step.

In the shaft was typical mining equipment that was used during the excavation works. Some of them we can see in the Figure 11.

As we wrote before the main duty of the hole was to transport the all excavation material from the bottom of the vertical shaft to the horizontal tunnel. But on the 02.04.2008 we got a full stuff and brake off the pipe in the drilling hole. The transport of the all excavation material with a container in vertical way was to slow so we decided to start another excavation from the horizontal tunnel to the vertical with an Alimak platform.

Figure 11. The machine excavation on the bottom of the shaft.

Figure 12. The typical mining equipment in the shaft.

Figure 13. The place of full stuff and brake off the pipe in the drilling hole.

The Alimak platform made a new shaft in diameter 2,0m x 2,0m along the advance hole for the material transport. The working platform Alimak made a 75m of new shaft.

Figure 14. The platform Alimak.

The works on excavation was made with a blasting and no support. On the 07.08.2008 we made a successful breakthrough and connection the both working point.

Figure 15. The platform Alimak and the bottom of the vertical shaft – breakthrough.

After the connection of the both shaft, one made by sunked by the application of the »strip and line metod« using hand held drill hammers in diameter of 4,0m and the other who made by Alimak way in diameter of 2,0m x 2,0m, we expand the last one. The all picture we can see in the Figure 16.

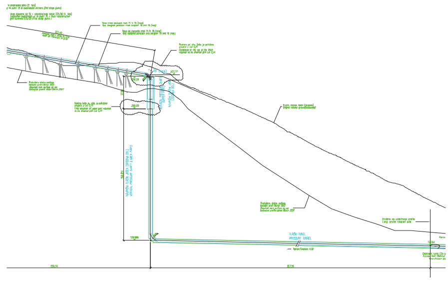

Figure 16. The all profile to vertical pressure shaft and horizontal pressure tunnel in the end of the works.

2 CONCLUSION

As we wrote in the beginning of this article, with this engineer mining works we helped that the civil works on the pumped storage hydropower plant Avče project go on. The represent vertical shaft is also the last shaft in Slovenia who was made after the year 1989. It is also the first shaft who made with a transport drilling hole in Slovenia. We can say now that we got a lot of new experience and a lot of technical establish (for example that the diameter 0,5m of the transport drilling hole was too small). With this paper we represent the basic information about the work in vertical shaft.

The project is still in realization and it will be complete in October 2009.

Estimating the Los Angeles Abrasion Loss of the Aggregates from Bedrock Properties

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ABSTRACT Aggregates are exposed to abrasion due to impact forces during processing, transporting and construction. Los Angeles abrasion (LAA) test is commonly used test for determining the abrasion loss and the relative competence of the rock aggregates. In this study, laboratory tests were carried out to determine the LAA loss values of aggregates produced from 12 different rock types and the physico-mechanical properties of parent rocks, e.g. bulk density, Schmidt Hardness, P-wave velocity, uniaxial compressive strength and tensile strength. Above rock properties were determined through standard testing methods in order to investigate the correlation between LAA and each rock property by using regression analysis. By analysing the correlations between the LAA loss values of the aggregates and pyhsico-mechanical properties of parent rocks, it was found that parent rock properties are good predictors for LAA value of aggregates and can be used for initial evaluation of rock formations as an aggregate source.

1 INTRODUCTION

Aggregates with distinctly different origins are widely used for a variety of construction purposes especially as road stone and in some concrete applications. Since they normally comprise between 75% and 85% by volume of portland cement concrete and asphalt mixes, the quality of aggregates is of considerable importance in determining their suitability for any specific engineering application (Al-Harthi, 2001). It would be damaging to construct houses or bridges or roads with building materials made with weak rock.

The suitability of aggregates for use in a given type of construction is determined by evaluating the material in terms of its physical and mechanical properties. Most specifications for aggregates require the material to be strong (Al-Harthi, 2001). Aggregates used must be tough and abrasion resistant to prevent crushing, degradation and disintegration when stockpiled, fed through an asphalt plant, placed with a paver, compacted with rollers, and subjected to traffic loadings. Aggregates, which do not have adequate toughness and abrasion resistance, may cause construction and performance problems (Wu and Parker, 1998).

Aggregates used in roads or infloor surfaces should be resistant to abrasion. The Los Angeles abrasion test (LAA) (ASTM, 1989) is one that reflects the aggregate resistance to abrasion and fragmentation due to impact and it has a considerable utility in determining the quality of the aggregates for the specification of requirements for their specific usages (Fernlud, 2005). LAA testing requires specialized equipment, large quantities of sized aggregates, time consuming sample preparation and difficult testing procedure. Therefore, reliable estimates of the performance of an aggregate may be made from the properties of the bedrock.

The objective of this study is to establish empirical relations between the Los Angeles abrasion resistance for 100 and 500 revolutions and physical and mechanical properties of rock samples collected from many locations in Turkey. This will be a step in this direction with the object of determining a method of estimating the strength of aggregates by using the simple techniques designed for rock property determination.

2 PREVIOUS WORKS

The predictive capability of the LAA test was considered fair by the researchers, although it is widely utilized. Early development studies, Moavenzadeh and Goetz (1963) observed that gradation is the most important factor for degradation and concluded that, the denser the mix the less the degradation. Aggregates with high Los Angeles values resulted in more degradation than those with low Los Angeles values. Rocks with good interlocking and strong cementation between grains produced less degradation than rocks with loose interlocking and weak cementation. Increase in the magnitude of load or increase in the number of repetitions of load increased degradation. Kazi and Al-Mansour (1980) carried out a study of the abrasion characteristics of crushed-rock aggregates obtained from a wide variety of igneous rocks of volcanic and plutonic origins. The grain size was found to be a significant geological factor controlling the evaluation of abrasion resistance. Fine-grained rocks when compared with coarse-grained rocks having the same porosity were found to be more resistant to wear. Gandhi and Lytton (1984) indicated strong and definitive correlations between the LAA values and field performance about toughness and abrasion resistance of the aggregates.

There is enough evidence in the literature to suggest that the strength of the aggregates are closely related to strength of the parent rock. For instance, Al-Harthi found meaningfull correlation among the LAA value of aggregate, rock compressive strength and point load index of rocks (Al-Harthi, 2001). Strength tests can be carried out directly by the uniaxial compressive strength tests (Hawkes and Mellor, 1970), by an indirect tests such as the point load test (Broch and Franklin, 1972 or by the Schmidt hammer test (Hucka, 1965). Cargill and Shakoor (1990) found strong linear correlations exist between the results of uniaxial compression vs the point load and Schmidt hammer tests, the correlation for the Schmidt hammer being dependent on rock type. Kahraman and Fener (2007) found a good correlation between uniaxial compressive strength and LAA loss.

3 LOS ANGELES ABRASION TESTING METHOD

Several means have been devised for assesing the abrasion resistance of aggregates, the most common method being the LAA test (ASTM, 1989; ISRM, 1981; TSE699, 1987). The Los Angeles test is a measure of degradation of mineral aggregates of standard gradings resulting from a combination of actions including abrasion or attrition, impact, and grinding in a rotating steel drum containing a specified number of steel spheres. The LAA test is widely used as an indicator of the relative quality or competence of mineral aggregates.

The Los Angeles abrasion testing machine consists of a hollow steel cylinder, closed at both ends, having an inside diameter of 710±5 mm and an inside length of 508±5mm (Fig.1).

A graded aggregate sample is placed into this cylinder with a charge consisting of steel spheres and is rotated for a specified number of revolutions from 100 to 500. The interior of the cylinder has a shelf that picks up the sample and charge during each rotation and drops them on the opposite side of the cylinder, subjecting the sample to abrasion or attrition. The cylinder rotates at 30 to 33 rpm and after the prescribed number of revolutions, the machine will automatically be stopped by a counter switch. The test consists of placing an aggregate sample in a steel drum along with 6-12 steel spheres weighing approximately 420 g each and having a diameter of 47 mm. Charge requirements for each sample size of grading are given in Table 1.

Figure 1. Los Angeles Testing Machine.

Table 1. Gradings of test samples.

Grain size class	Steel sphere number	Sieve size (mm) (square openings)		Weight of indicated sizes(g)	Total weight (g)
		Passing	Retained		
A	12	40	25	1250±25	5000±10
		25	20	1250±25	
		20	12.5	1250±10	
		12.5	10	1250±10	
B	11	20	12.5	2500±10	5000±10
		12.5	10	2500±10	
C	8	10	6.3	2500±10	5000±10
		6.3	5	2500±10	
D	6	5	2.5	5000±10	5000±10

Following the completion of the 100 revolutions, the resulting sample is dry sieved over 1.6 mm sieve (G_{100}) to determine the amount of degradation that occured during the test. The whole sample was then placed together with dust again into the cylinder and rotated for another 400 revolutions. The material coarser than 1,6 mm sieve was separated and weighed (G_{500}). The abrasion loss as a percentage of the original mass of the test sample after 100 and 500 revolutions is calculated according to the Eqs. 1 and 2, respectively.

$$K_{100} = \frac{G_0 - G_{100}}{G_0} * 100 \quad (1)$$

$$K_{500} = \frac{G_0 - G_{500}}{G_0} * 100 \quad (2)$$

Where;
K_{100} =Abrasion loss after 100 revolutions (%)
K_{500} =Abrasion loss after 500 revolutions (%)
G_0 = Original sample mass (g)
G_{100} = Sample mass after 100 revolutions (g)
G_{500} = Sample mass after 500 revolutions (g)

4 EXPERIMENTAL WORK

4.1 Sample Description

The samples tested included ignous rock of volcanic origin as well as both metamorphic and sedimentary rocks. The volcanic igneous rock is andesite while the metamorphic rocks are mainly marbles and the sedimentary samples are limestones, travertines and lymras of different grain sizes.

The tests were carried out on 12 different rock samples representing ten different localities reflect the natural distribution in rock properties throughout the country (Table 2).

Table 2. Details on rocks tested.

Rock name	Rock type	Rock class	Location
Tundra Grey	Limestone	Sedimentary	Afyon
La Perla	Limestone (Lymra)	Sedimentary	Demre-Antalya
Gold E	Travertine	Sedimentary	Sivas
Crema Zelve	Marble	Metamorphic	Bilecik
Classic Light	Travertine	Sedimentary	Denizli
Afyon Sugar	Marble	Metamorphic	Iscehisar-Afyon
Isparta Andesite	Andesite	Magmatic	Isparta
Konya Travertine	Travertine	Sedimentary	Karaman-Konya
Bucak Travertine	Travertine	Sedimentary	Bucak-Burdur
Burdur Beige	Limestone	Sedimentary	Yeşilova-Burdur
Burdur Brown	Limestone	Sedimentary	Karamanlı-Burdur
Salome	Marble	Metamorphic	Eskisehir

Extraction of the test specimens was made in one of three different operations. Core drilling operations using an electromechanical coring machine to produce core specimens with specific dimension, cutting operation using diamond saws to obtain cubic samples and crushing operation using jaw crushers to produce particular size fraction of aggregate material. The aggregate samples were produced using jaw crushers in the size fraction of not more than 20 mm not less than 10 mm to meet the specifications issued by the TS 699. Cylindrical cores were sawed by using the cutting machine and finished with No. 120 abrasive. The dimensions of the finished discs were approximately 54 mm in diameter and 35 mm in thickness.

Sample preparation was carried out in accordance with the specifications of the ASTM 1989 and TS 699. Description of the main tests conducted in this research is outlined in the following sections. The samples were initially tested for their physical and mechanical properties such as bulk density, P-wave velocity, Schmidt and Shore hardness, uniaxial compressive strength, indirect tensile strength and point load index and then the fresh rock samples were crushed and tested for LAA value. The grading of aggregate samples was performed by sieving the crushed samples.

4.2 Los Angeles Abrasion Test

LAA values were measured according to TS 699 for both 100 and 500 revolutions together with 12 samples of Grade B (Table 1) were tested using 11 steel spheres. The abrasion values are given in Table 3.

Table 3. LAA value of rocks.

Rock name	K_{100}	K_{500}
Tundra Grey	4.6	25.8
Laperla	7.8	34.3
Gold E	5.8	24.5
Crema Zelve	7.2	33.6
Classic Light	5.6	26.0
Afyon Sugar	6.3	26.7
Andesite	4.8	20.2
Konya Travertine	10.0	40.9
Bucak Travertine	9.0	37.8
Burdur Beige	7.3	23.5
Burdur Brown	5.8	23.9
Salome	7.1	30.7

4.3 Properties of BedRock

4.3.1 Porosity and bulk density

The effective porosity and bulk density of rock samples were determined using saturation and bouyancy techniques, as recommended by ISRM (1981) and TSE699 (1987). All samples were saturated by water immersion for a period of 48 h with periodic agitation to remove trapped air. Later, the samples were transferred underwater to a basket in an immersion bath and their saturated-submerged weights were measured with a scale having 0.01 g accuracy. Then, the surface of the samples was dried with a moist cloth and their saturated-surface-dry weights were measured outside water. Bulk sample volumes were found from weight differences between saturated-surface-dry weight and saturated-submerged weight. The dry mass of samples was determined after oven drying at a temperature of 105 ^oC for a period of at least 24 h. The effective pore volumes were determined from weight difference between saturated-surface-dry weight and dry sample weight. The density of samples was calculated by dividing the dry weight of samples to the bulk volumes; whereas, the effective porosity was found by the ratio of pore volume to bulk sample volume. Average bulk density and porosity values with standard deviations (SD) and coefficient of variations (CoV) for fresh rocks are given in Table 4.

Table 4. Bulk density and effective porosity of rocks.

Rock name	Bulk density (gr/cm^3)			Effective porosity (%)		
	Mean	SD	CoV (%)	Mean	SD	CoV (%)
Tundra Grey	2.635	0.016	0.61	1.842	0.475	25.78
La Perla	2.382	0.008	0.33	8.982	0.221	2.46
Gold E	2.521	0.015	0.61	4.580	0.427	9.32
Crema Zelve	2.690	0.002	0.06	0.386	0.039	9.99
Classic Light	2.458	0.011	0.43	2.139	0.236	11.06
Afyon Sugar	2.702	0.002	0.07	0.257	0.205	6.61
Isparta Andesite	2.280	0.010	0.46	7.765	0.205	6.61
Konya Travertine	2.124	0.050	2.35	6.591	0.435	6.61
Bucak Travertine	2.400	0.015	0.61	2.612	0.196	7.51
Burdur Beige	2.675	0.004	0.14	0.575	0.116	20.26
Burdur Brown	2.689	0.003	0.12	0.342	0.023	6.77
Salome	2.707	0.004	0.13	0.363	0.028	7.84

4.3.2 P-wave velocity

P-wave velocities of samples were measured on cubical shaped rock blocks having 70 mm edge length. In the tests, the PUNDIT instrument and two transducers (a transmitter and a receiver) having a frequency of 54 kHz were used. The direct transmission method which is more sensitive than the other methods was preferred for measurement of P-wave velocities of rocks. The faces of the samples were flattened and smoothed to provide tight contact of transducers with the face of the specimen. The velocity was calculated from the ratio of travel distance to travel time of the P-wave through the rock sample. For each rock type, the measurements were made on eight samples and the average P-wave velocity, standard deviation and coefficient of variation for fresh rocks are given in Table 5.

Table 5. P-wave velocity of rocks.

Rock name	P-wave velocity (m/s)		
	Mean	SD	CoV (%)
Tundra Grey	5900	174	3
La Perla	4295	208	5
Gold E	5075	172	3
Crema Zelve	6218	90	1
Classic Light	4984	216	4
Afyon Sugar	5424	234	4
Isparta Andesite	4745	57	1
Konya Travertine	3730	247	7
Bucak Travertine	4056	216	5
Burdur Beige	5803	140	2
Burdur Brown	6022	231	4
Salome	5589	245	4

4.3.3 Schmidt rebound hardness

Schmidt rebound hardness of rocks was determined by following the testing procedure suggested by ISRM (1981). An L-type Schmidt hammer having an impact energy of 0.74 Nm was used. Cubic block samples having an edge dimension of 70 mm were sawed from the large blocks. The testing side surfaces of samples were smoothened. The Schmidt hammer was held on vertically downward position. 20 impacts were carried out on any rock sample as each test location separated by at least the diameter of the plunger. The upper 10 values from the measured test values for each rock were taken into consideration. The average values of Schmidt hardness for each rock type with standard deviations and coefficient of variations are given in Table 6.

Table 6. Schmidt rebound hardness of rocks.

Rock name	Schmidt hardness		
	Mean	SD	CoV (%)
Tundra Grey	45	3	7
La Perla	32	2	7
Gold E	44	2	5
Crema Zelve	52	1	2
Classic Light	34	2	7
Afyon Sugar	46	1	3
Isparta Andesite	47	3	6
Konya Travertine	27	3	11
Bucak Travertine	35	2	4
Burdur Beige	47	2	3

Burdur Brown	49	1	3
Salome	46	2	5

4.3.4 Shore hardness

The shore hardness of the rocks was determined according to the specifications outlined by the ISRM (1981) by using C-2 model shore hardness testing device. Five specimens of each rock type were prepared and 20 readings were obtained for each specimen. The results from this test with standard deviations and coefficient of variations are given in Table 7.

Table 7. Shore hardness of rocks.

Rock name	Shore hardness		
	Mean	SD	CoV (%)
Tundra Grey	48.500	9.630	19.86
La Perla	29.800	3.440	11.55
Gold E	51.300	3.180	6.2
Crema Zelve	62.900	2.610	4.16
Classic Light	37.000	3.950	9.7
Afyon Sugar	49.350	2.850	5.78
Isparta Andesite	69.750	7.480	10.72
Konya Travertine	37.400	7.230	19.33
Bucak Travertine	38.050	4.640	12.19
Burdur Beige	58.550	3.720	6.35
Burdur Brown	64.900	2.360	3.64
Salome	52.700	1.560	2.96

4.3.5 Uniaxial compressive strength

The uniaxial compression strength was determined following TS recommendations (TSE699, 1987) and tests were carried out on cubical shaped samples having an edge length of 70 mm prepared by sawing from large blocks. The stress rate was within the limits of 1–1.2 MPa/s. The results from this test with standard deviations and coefficient of variations are given in Table 8.

4.3.6 Tensile strength

The indirect test commonly adopted for the tensile strength determination, known as the Brazilian Test, was conducted. Cylindrical rock cores were used for this test in accordance with the ISRM (1981). The diameter of the cores is 54 mm; these cores were cut by diamond saw into thin discs with thickness around 35 mm. The tensile strength of each sample was calculated from the following expression;

$$\sigma_t = \frac{2P}{\pi D t} \tag{3}$$

Where σ_t is the tensile strength (MPa), P is the load at failure (N), D is the diameter of the sample (mm) and t is the thickness of the sample (mm). The results from this test with standard deviations and coefficient of variations are given in Table 9.

Table 8. Uniaxial comp. strength of rocks.

Rock name	Uniaxial comp. strength(MPa)		
	Mean	SD	CoV (%)
Tundra Grey	91	23	25
La Perla	73	5	7
Gold E	84	14	17
Crema Zelve	126	14	11
Classic Light	68	17	24
Afyon Sugar	98	9	9
Isparta Andesite	82	9	25
Konya Travertine	20	9	44
Bucak Travertine	36	9	27
Burdur Beige	100	14	14
Burdur Brown	119	26	22
Salome	70	13	18

Table 9. Tensile strength of rocks.

Rock name	Tensile strength (MPa)		
	Mean	SD	CoV (%)
Tundra Grey	5.73	1.53	27
La Perla	3.90	0.28	7
Gold E	6.41	1.56	24
Crema Zelve	8.23	0.81	10
Classic Light	5.78	1.71	30
Afyon Sugar	6.89	1.69	25
Isparta Andesite	11.42	1.63	14
Konya Travertine	2.52	0.19	8
Bucak Travertine	3.36	0.59	17
Burdur Beige	8.01	0.46	6
Burdur Brown	7.00	1.09	16
Salome	8.18	1.16	14

4.3.7 Point load index

Rock specimens in the form of blocks with depth to width ratio between 0,3 and 1 were broken by application of load through a pair of spherically truncated, conical platens. The point load strength was determined by using the following equation.

$$I_s = \frac{P}{D_e^{\,2}} \tag{4}$$

To convert the values for 50 mm diameter core size, calculated values from Eq. 4 were corrected by using the following equation;

$$I_{s(50)} = \left(\frac{D_e}{50}\right)^{0,45} * I_s \tag{5}$$

Where I_s is the point load strength of the rock (MPa), P is the failure load (N), D_e is the equivalent core diameter (mm). The results from this test with standard deviations and coefficient of variations are given in Table 10.

Table 10. Point load index of rocks.

Rock name	Point load index Mean	SD	CoV (%)
Tundra Grey	3.10	0.58	18.82
La Perla	3.50	0.38	10.86
Gold E	4.16	0.44	10.62
Crema Zelve	4.11	0.36	8.83
Classic Light	4.48	0.51	11.46
Afyon Sugar	3.89	0.39	9.98
Isparta Andesite	7.79	0.74	9.5
Konya Travertine	1.60	0.17	10.82
Bucak Travertine	2.79	0.52	18.57
Burdur Beige	5.57	0.37	6.7
Burdur Brown	4.87	0.31	6.33
Salome	4.06	0.42	10.24

5 RESULTS AND DISCUSSION

LAA values of rocks depending on the rock properties were analyzed using the method of least squares regression. The equation of the best-fit line and the coefficient of determination (R^2) were determined for each regression. In an attempt to establish a more meaningful relationship, LAA values were divided by P-wave velocity (V_p) values since it is strongly dependent on the porosity, density, mineral composition, size and frequency of fractures in the rock structure and indicates weakness of rocks to abrasion. The normalised LAA values were correlated with the bulk density and uniaxial compressive strength, tensile strength, Schmidt and Shore hardness values and point load index. The correlation coefficients and best fit curves were calculated by the least squares curve fit method. The plot of the normalised LAA values as a function of the bulk density, uniaxial compressive strength, tensile strength, Schmidt and Shore hardness values and point load index are shown in Fig. 2-7 respectively.

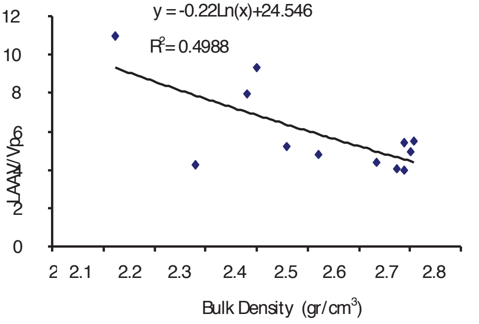

Figure 2. LAAV/Vp vs bulk density

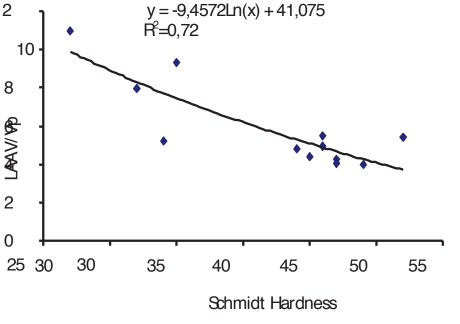

Figure 3. LAAV/Vp vs Schmidt hardness.

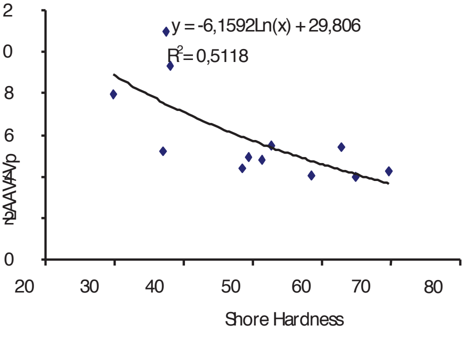

Figure 4. LAAV/Vp vs Shore hardness.

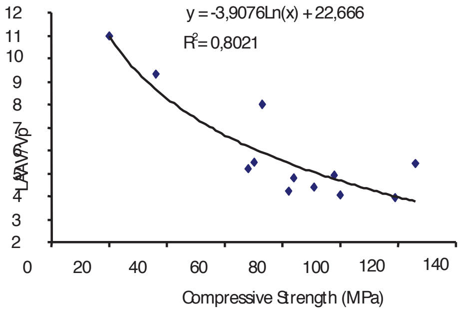

Figure 5. LAAV/Vp vs uniaxial compressive strength.

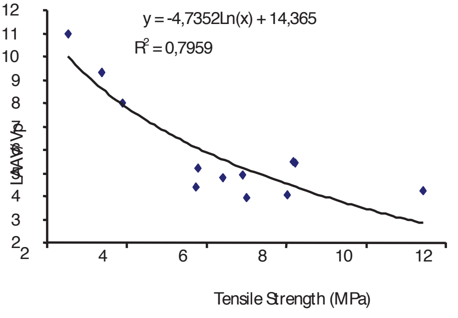

Figure 6. LAAV/Vp vs tensile strength.

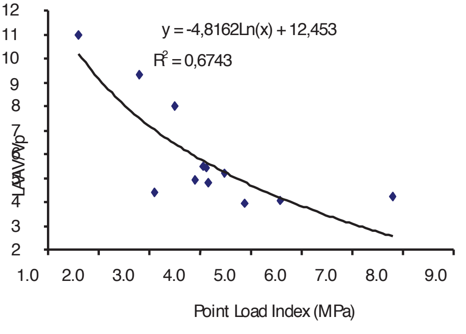

Figure 7. LAAV/Vp vs Point load index.

As can be seen from Figs., the LAA value is inversely related to measured properties and decreases with increase in each different rock property. Among the large number of functions, following logarithmic function found to be providing the best correlation to data is given in Eq. 6.

$$y = a * \ln x + b \qquad (6)$$

Where;
$y = LAA / V_p$
$x =$ Rock property
a, b = Regression coefficients

Determination coefficient (R^2) values larger than ± 0.50 were considered statistically significant at a 99 % confidence level with 10 degrees of freedom (Snedecor, 1989). According to this statistical judgement, compressive strength, tensile strength, Schmidt hardness and point load index could adequately estimate the LAA value of rocks whereas rough estimates could be made with bulk density and shore hardness of rocks. To test the significance of the Eq. 6, analysis of variance for the regression was performed. This test follows an F-distribution with numerator degrees of freedom of 1 and the denominator degrees of freedom of 10 so that the critical region is consist of values exceeding 10.04 for a 99 % level of confidence. If the calculated F-value is greater than the tabulated F-value, we reject the null hypothesis that there is a significant relation between LAA value of rocks and specific rock property. Analysis of variance for the test procedures is given in Table 11. Calculated F-values for predictors of compressive strength, tensile strength, Schmidt hardness and point load index are significantly greater than the tabulated F-value; the null hypothesis is rejected. From this it can be concluded that these properties are significant rock properties to predict the LAA value. On the other side, bulk density and shore hardness are not so significant properties in predicting the LAA value that rough estimates could be made with these rock properties.

Table 11. Variance analysis for the significance of regressions.

Source of variation	Predictor from Eq 1	Sum of squares	Degrees of freedom	F-Test	F-Table (99% confidence level)
Regression	Bulk Density	28.45	1	9.98	10.04
	Compressive Strength	45.68	1	40.59	10.04
	Tensile Strength	45.32	1	39.02	10.04
	Schmidt hardness	41.34	1	26.52	10.04
	Shore hardness	29.15	1	10.49	10.04
	Point load	38.44	1	20.79	10.04
Residual	Bulk Density	28.48	10		
	Compressive Strength	11.25	10		
	Tensile Strength	11.61	10		
	Schmidt hardness	15.58	10		
	Shore hardness	27.78	10		
	Point load	18.48	10		

6 CONCLUSIONS

Laboratory testing was carried out on 12 different rocks to investigate LAA value of each rock. Bulk density, P-wave velocity, uniaxial compressive strength, tensile strength, Schmidt hardness, Shore hardness and point load index values of rocks were also determined by standard testing methods to establish the relationship between LAA value and each rock property. Among the tested rocks, andesite and limestones are more resistant to abrasion than marbles and travertines. Rock properties have certain influence on the abrasion of rocks and could be used to predict LAA value of rocks. It was found that the LAA value of rocks was smallest for the high density, compressive strength, tensile strength, hardness, point load index and P-wave velocity values. Dependence of abrasion characteristics on each rock property investigated by regression analysis showed that high correlations exist between LAAV/Vp and compressive strength, tensile strength, schmidt hardness, point load index.

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Numerical Modeling for the Determination of Umbrella Arch and Face Bolt Effects on the Ground Settlement Induced by Urban Shallow Tunneling

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ABSTRACT Tunnel stability is the common problem in the weak ground condition. Other hand, ground settlement, induced by urban tunneling, is very important matter for the structures and infrastructures around the tunnel effect area in the city. As known, the plastic zone does not occurred properly around the shallow tunnel. So, ground settlement can reached extensive area if not to take measurements. The artificial shell can be formed with the umbrella arch application and the face internal properties can be increased and face movement can be prevented by face bolt applications. Ground settlement values can be reduced by those applications. In this study, the effects of umbrella arch and face bolt application on the ground settlement were investigated with the numerical modeling at the Bornova metro tunnel project, which is in the earthquake zone. The results of the numerical modeling were compared with the field measurements and the satisfaction results were found.

1 INTRODUCTION

Construction of a bored tunnel with a length of 761 m was designed as part of 3.2 km long new light rail system in conjunction with Izmir Metro Project between the Hospital of Ege University and the center of Bornova based on the data obtained from laboratory and field studies. Following conditions were considered in all calculations:

a. State of primary stresses (stresses created by the virgin rock prior to the excavation)

b. State of excavation after completion (unsupported tunnel after the excavation or for excavation in stages, consideration of the behavior of excavated and supported tunnel section for the next stage).

c. Final state of primary support (Final state after the completion of all support requirements along the tunnel line).

The site of investigations is situated in Bornova District of Izmir Province and contains the area between Manisa Junction and center of Bornova as shown in Figure 1.

2 GENERAL GEOLOGICAL PROPERTIES OF INVESTIGATION SITE AND ITS VICINITY

Izmir and its surroundings are geologically composed of aged Bornova Melange (Onargan et. al., 2007). Limestone mega-olistolites which are older than matrix of Bornova Melanges can be found as scattered in Melanges. The limestones mentioned above are also known as Isiklar limestone around Altindag (Onargan et. al., 2007). Bornova Melange is composed of platform type limestone and diabase blocks and pebblestone lens-channel fillings which float in a matrix formed of sequential sandstone and shaled-limestone (Erdoğan, 1990). Neogen aged lacustrine sediments cover the

Bornova Melange angularly inconsonant. Yamanlar volcanics also covers these units inconsonantly. Quarternary aged alluvium inconsonantly lay on all the geological units in the investigation area. Longitudinal crossections.of the area drilled and engineering properties of rock formations recovered from the drillholes can be seen in Figure 2 and Table 1 respectively.

Investigation area is densely populated and is classified as 1st degree earthquake zone. For the project, damage tolerance on nearby structures was kept very low owing to adverse effects of existing damages, which were induced by the recent earthquakes. Geological formations along the tunnel line are composed of heavily altered low strength rocks and contain patchy clay with high plastic property. Certain rock mass values calculated using the data obtained from the drillholes are given in Table 2.

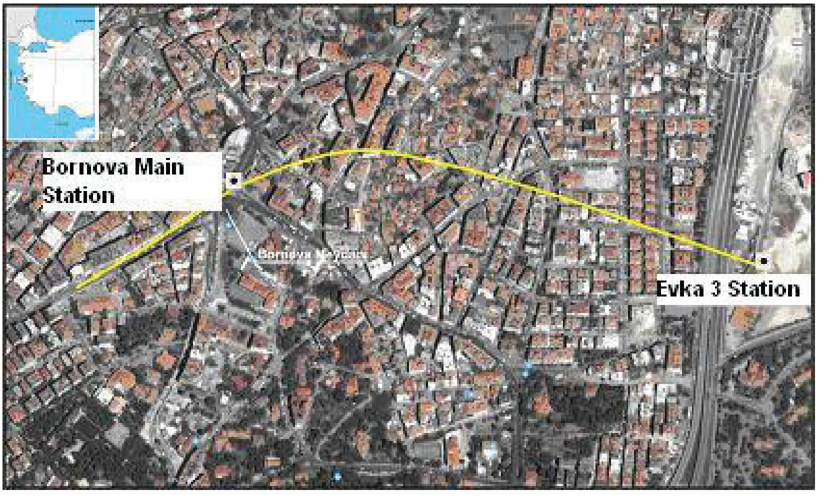


Figure 1. Construction site and the tunnel line.

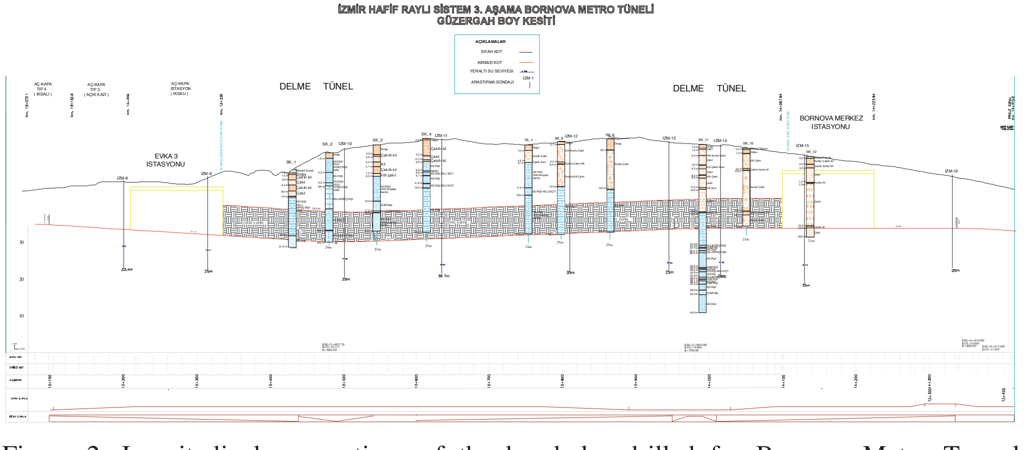


Figure 2. Longitudinal crossections of the boreholes drilled for Bornova Metro Tunnel Project.

Table 1. Engineering properties and the failure criteria for rock formations recovered from the drillholes in the investigation area.

Drillhole No	Depth m	Hoek and Brown Rock Mass Classification System			Hoek and Brown Failure Criteria			Mohr-Coulomb		Rock mass parameters (MPa)		
		GSI	Rock Stability m_i	İntact Elasticity Modulus $E_{i,}$ MPa	m_b	S	a	c MPa	ϕ	σ_τ	$\sigma_{\mu\chi}$	E_m
SK-2	2.5–3.00	19	4	14	0.085	2.04e-5	0.547	0.008	11.24	-0.000218	0.002	0.41
SK-2	7.6–8.00	19	4	14	0.085	2.04e-5	0.547	0.008	10.43	-0.000176	0.002	0.41
SK-2	23.0–24.00	22	4	29	0.024	3.04e-5	0.538	0.004	5.50	-0.000100	0.002	0.92

GSI: Geological Strength Index *ϕ : Internal Friction Angle* *σt : Mass Tensile Strength*
C : Cohesion *σ_{cm} : Mass Uniaxial Compressive Strength* *E_m = Mass Deformation Modulus*

Table 2. Results of rock mass classifications.

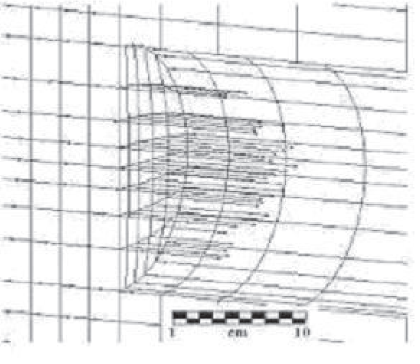
WELLBORE NO	LİTHOLOGICAL DEFINITION	TUNNEL METER	DEPTH m	σ_χ MPa	GSI		RMR		Q	
SK-1	**CLAY WITH PEBBLE** Brown and altered	13+434.93	4.70-5.00	1.35	25	Very weak	23	Weak	1.466	Very weak
SK-1	**CLAYSTONE (Altered)** Brown, altered in carbonate patches, very low strength	13+434.93	14.00 – 14.50	0.18	15	Very weak	15	Very weak	0.0088	Extremely weak

Rock mass parameters indicate that the quality of rock mass along the tunnel route is classified as extremely poor. The project bears a high risk since that the area is densely populated, pre-existing damages on the structures from recent earthquakes and possibility of hazard due to shallow tunneling conditions, in which lack of arching around the tunnel may generate higher lateral stresses than vertical stress causing shear failure on the sidewalls of the tunnel. Therefore, it was aimed to form an artificial arch around the tunnel in order to minimize the effect of tunnel excavation by integrating in the project double row umbrella arch (2 in.-9mm) and face bolts of 12m long, especially starting from the station at Evka 3 location, within a 100 m of which there is a highway with heavy traffic of 10000 vehicles a day. Also, alongside this highway, condition of high rise buildings with pre-existing earthquake damages must be evaluated during the construction of metro tunnel Rock mass parameters indicate that the quality of rock mass along the tunnel route is classified as extremely poor. The project bears a high risk since that the area is densely populated, pre-existing damages on the structures from recent earthquakes and possibility of hazard due to shallow tunneling conditions, in which lack of arching around the tunnel may generate higher lateral stresses than vertical stress causing shear failure on the sidewalls of the tunnel. Therefore, it was aimed to form an artificial arch around the tunnel in order to minimize the effect of tunnel excavation by integrating in the project double row

umbrella arch (2 in.-9mm) and face bolts of 12m long, especially starting from the station at Evka 3 location, within a 100 m of which there is a highway with heavy traffic of 10000 vehicles a day. Also, alongside this highway, condition of high rise buildings with pre-existing earthquake damages must be evaluated during the construction of metro tunnel.

Certain reinforcement systems cannot be literally distinguished. Thus, pipe-arch reinforcement process should be separated from other systems. In this process, the main tunnel is systematically excavated under the protection of previously constructed pipe jackings. The pipe arch process is one of the safest and most efficient ways of constructing large diameter tunnels over short distances. It is likely to encounter various applications of Pipe Arch method in literature (Hoek, 2003; Kim et. al. 2004, Miura, 2003; Gibbs et al. 2002). The pipes can be exercised on the ground by two different methods, being pre-drilling and case-drilling. Both methods possess advantages and disadvantages (Volkmann, 2004).

In shallow urban tunneling by NATM, methods of pipe arch (umbrella arch), jet grout arch, ground dowels, face jet grout column must be exercised extensively in addition to the conventional pre-support systems such as shotcrete, rock bolts, steel mesh. Application of the face bolts reduces the surface settlements by 60%. Face convergence values prior to and after the installation of face dowels can be seen comparatively in Figure 3.



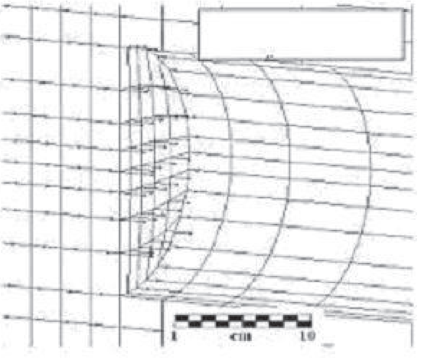


Figure 3. Variations in the convergence values at the excavation face prior to and after the installation of face bolts.

In this study, application of a robust pre-support system has been suggested after the assessment of all possibilities, especially for the location near the highway with heavy traffic and the buildings with pre-existing earthquake damages. Suggested pre-support system is shown in Figure 4.

Finite Elements and Finite Difference methods were both employed in the analyses and determination of pre and final support systems required for the stability of metro tunnel construction for the section of Ege University Hospital and Bornova Center as part of the 3rd stage of Izmir Metro Project. The main objective of the numerical analyses has been to be able to suggest the best construction method (stage construction) and pre-support system that will prevent any possible damage resulting from misapplication of tunneling process. The chart of principles used in numerical modeling is shown in Figure 5.

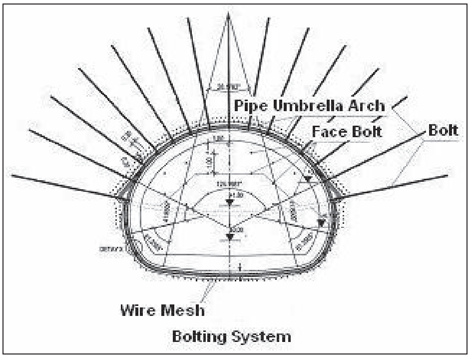


Figure 4. Proposed pre-support system.

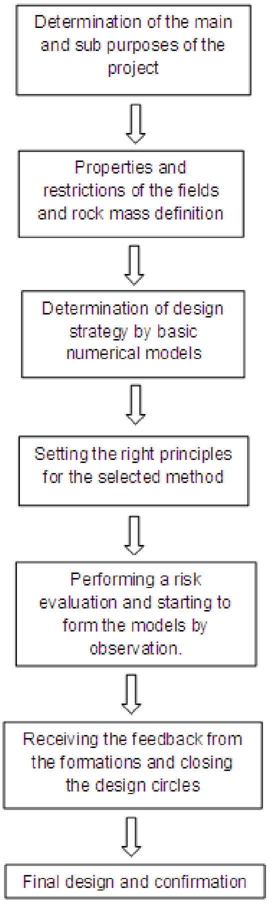


Figure 5. The principles used in numerical modeling.

In the result of assessments performed, properties of materials to be used, members of pre-support system and properties of the concrete for final lining were determined and used as input in the analyses. Material properties used in the analyses will be given separately for each analysis.

Excavation stages and pre-support parameters for the final models formed are as follows:

- Steel props: one HEA 120 large flange
- Shotcrete lining: double layer 30 cm (C20 concrete)
- Steel wire mesh: Double layer (Q221/221)
- Steel pipe arch: 9 m. long and overlapped at each 4.5 m. to the bottom of the tunnel.
- Face bolt: 12 m. long and 14 mm. diameter ribbed steel bar (overlapped at each 6 m)

In numerical modeling, PLAXIS 3D V2 was employed for finite elements analyses. A distributed load of 10 kN/m2 was defined in order to simulate the weight of buildings etc., static (surcharge) and dynamic loads. In the model, excavation stages and the properties of soil and construction materials are given in Figure 6 and Tables 3 and 4 respectively.

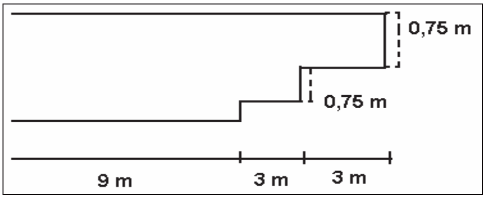


Figure 6. Excavation and pre-support stages.

Table 3. Soil parameters used for portal support class.

		Filling Material (Mohr-Coulomb)	Gravel with clay and sand (Mohr-Coulomb)	Clayed Limestone (Hardening Soil)	Face shotcrete (Linear Elastic)
Parameter		Drained	Drained	Drained	Non-porous
γunsat	[kN/m³]	18	17.6	18.1	24
γsat	[kN/m³]	18	17.6	18.1	-
Eref	[kN/m²]	24670	56830	249000	210000000
ν	[-]	0.38	0.35	0.34	0.25
Gref	[kN/m²]	8,939,130	21050	-	84000000
Eoed	[kN/m²]	46190	91210	318000	252000000
cref	[kN/m²]	77	118	182	-
φ	[°]	21	29	31	-
Tstr.	[kN/m²]	0,150	0.16	0,18	-
Permeability		Neutral	Neutral	Neutral	

Table 4. Parameters of construction materials used for portal support class.

No.	Construction member	EA [kN/m]	EI [kNm²/m]	W [kN/m/m]	ν [-]	Mp [kNm/m]	Np [kN/m]
1	Shotcrete	7.39E+08	5.55E+06	48.2	0.1	1.00E+15	1.15E+16
2	Temporary Invert	1.68E+11	1.40E+05	2.8	0.25	1.00E+15	3.46E+16
3	Steel pipe	2.69E+07	8.95E+04	7.6	0,12	1.00E+15	1.73E+16
4	Anchorage	2.00E+05	-	-	-	1.00E+15	-
5	Fore-poles	1.40E+07	7.46E+05	8.4	0.15	1.00E+15	4.33E+15
6	Face Bolt	9.55E+04	-	-	-	-	1.00E+10

The method followed in finite elements analyses is explained below. In the analyses, construction stages are described as follows.

First Stage

First, fore-poles and anchorages of 0.8 m long were placed. Then, steel pipe arches of 9 m long (2 in, double row, wall thickness of 5.5 mm) with 4.5 m overlaps and face bolts of 12 m long with 6 m overlaps were defined. As the pre-support, a HEA 120 (19.9 kg/m) large steel flange at each 0.75 m , 0.30 m thick shotcrete and double-layer steel wire mesh were defined. By applying 0.05 m thick shotcrete stability of the face was increased. In addition, a temporary invert with 0.1 m thickness was also defined.

Second Stage

Following the solution of the first stage, an excavation of 0.75 m was implemented in the upper half of the tunnel. No pre-support was applied at this stage.

Third Stage

Pre-support and temporary invert were applied for the tunnel section excavated in the second stage.

Fourth Stage

The model appeared to be stable in the end of third stage. In the fourth stage, 0.75 m long excavation was implemented in the lower bench which was 3 m behind the upper bench of the tunnel. No pre-support installed in this stage.

Fifth Stage

Pre-support were applied for the tunnel section excavated in the fourth stage.

Sixth Stage

Invert excavation of 0.75 m was carried out 4 m behind the lower bench, which was excavated in the 4th stage.

Seventh Stage

Pre-support was designed to stabilize the invert excavation conducted in the 6th stage.

The data obtained from the solutions of the models in the end of seven stages of excavation and pre-support steps as explained above are given below in details. Values for vertical deformation from numerical solutions are shown in Figure 7.

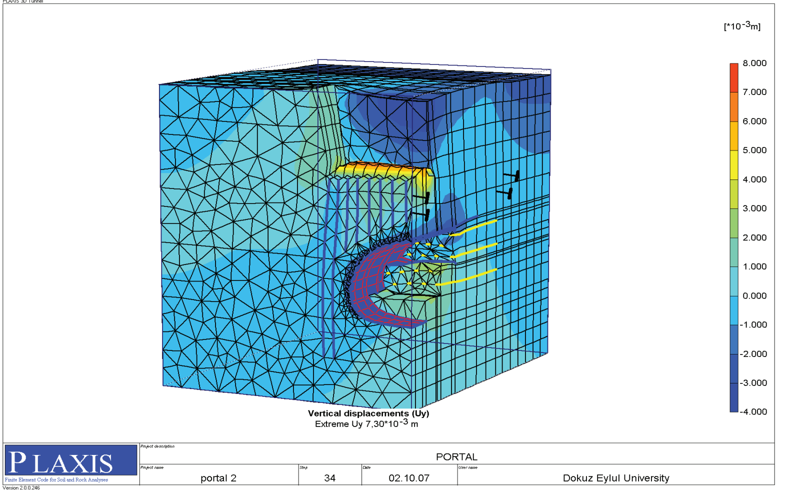


Figure 7. Colored displaying of vertical deformations occurred in case portal support class is applied.

The results of numerical modeling predicted vertical deformation of 3 mm in the crown in the location near the highway with heavy traffic. However, in the same location, numerical analyses indicated that tunnel failed in case pre-support members of umbrella arch and face bolts were not installed. Field measurements conducted in the same locations yielded a maximum surface settlement of 2.7 mm. The values obtained from surface settlement measurements are illustrated in Figure 8.

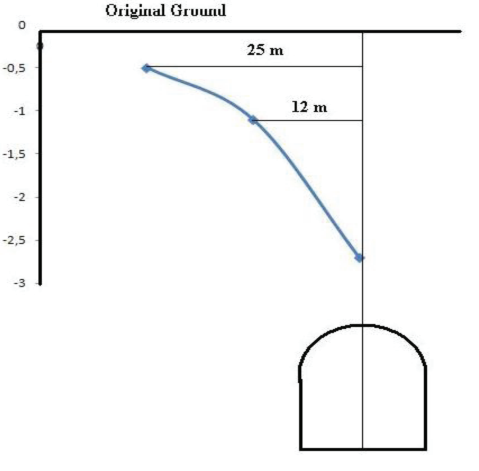


Figure 8. Ground settlement values obtained from field.

3 CONCLUSIONS

In densely populated cities or in particular parts of a city with high population construction of metro lines is becoming inevitable. Of course, tunnels as part of the metro systems must be excavated and their stability must be warranted. The most important problem inner-city metro tunnels is that tunnel excavations induce surface settlements and therefore damages on the building and infrastructure situated along the line of metro tunnels if the ground settlements exceed a certain amount. In order to prevent such damages, all possible measurements must be taken to prevent deformations around excavations. Application of umbrella arch method, which forms an artificial shell around the tunnel, and face bolts, which increase the internal parameters of excavation face and restrict the movement of face into the tunnel will significantly decrease the values of surface settlement. In this study, numerical simulation has shown that tunnel would collapse the applications above were not implemented for shallow tunnels in poor ground conditions. The numerical model with the simulation of umbrella arch and face bolts near a busy highway yielded a 3 mm surface settlement, which was actually measured in the field as 2.7 mm. The values of ground settlement measured in the field were found to be quite close to which obtained from numerical solutions. In result, the methods of umbrella arch and face bolts provide a stable tunnel excavation by limiting or preventing excessive surface settlements which may eventually create further damage on nearby buildings and infrastructure.

ACKNOWLEDGEMENTS

The authors performed this study based on the protocol signed with BOZOĞLU İnş. San. Ltd. Şti. Also, the authors thank to ESER Müh. Müş. A.Ş. and Izmir Greater Municipality for their support and convenience.

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İzmir Metrosu 2. Aşama İnşaatındaki Kayaların Özelliklerinin İncelenmesi ve Karşılaşılan Kısıtlamalar

Investigation of Rock Properties in 2nd Stage of Izmir Metro Construction and Restrictions

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ÖZET Bu çalışmada İzmir Metrosu 2. Aşama İnşaatı kapsamında açılan 64 m^2 kesitli Tip 1 hat tünellerinden alınan örnekler üzerinde deneyler yapılmıştır. İzmir Metro Projesinin birinci aşamasıyla entegre olan ikinci aşamaya ait hattın tünel inşası, İzmir Körfezinin güneybatısında, Üçyol ile Fahrettin Altay arasındaki güzergâhı takip eder. Tüneldeki kazı sekiz servis kuyusu üzerinden yürütülmektedir. Tünel hattındaki baskın kaya birimleri Yamanlar volkanitleri olarak bilinen, düşük ve orta derece arasında başkalaşım gösteren andezitler; Altındağ formasyonu olarak bilinen, aglomera-kumtaşı-silttaşı-kiltaşı dizisi; Bornova karmaşığı olarak bilinen sarımsı kahverengi filiş ve grimsi siyah filiştir. Bu birimlerden alınan örnekler üzerinde çeşitli kaya mekaniği deneyleri yapılmıştır. Ayrıca bu çalışmalardan başka yine aynı tünel güzergâhında yapılan jeoteknik çalışmalar incelenmiştir. Elde edilen veriler sonrasında özellikle Bornova Karmaşığında (laminalı–foliasyonlu–anizotrop) tek eksenli basınç dayanımı (T.E.B.D.) testi ile kayaç malzemesinin dayanımının standartlara göre (ISRM ve ASTM) tespitinin çok zor, zaman alıcı ve hatta bazı durumlarda imkânsız olduğu belirlenmiştir. Her ne kadar uygun numune üzerinde ve tünel aynalarında (çok az miktarda) T.E.B.D. testi, Nokta Yük Dayanım Testi (N.Y.D.T.) ve Schmidt Çekici (S.Ç.) Testleri yapıldıysa da sonuçlar tatmin edici bulunmamıştır. Bu nedenle kayaç malzemesinin dayanımının tespitinde Disk Makaslama Dayanım İndeksi (D.M.D.İ.) kullanılmıştır. Bu veriler kullanılarak kaya kütlesinin sınıflaması yapılmış, parametreleri hesaplanmış ve nümerik modellemede kullanılmıştır.

ABSTRACT This study includes experiments made on the samples which had been taken from type 1 line tunnels with 64 m^2 cross-section of Izmir Metro 2. Phase Construction. The tunnel construction of the line of second phase is integrated with the first phase of Izmir Metro Project and it follows the route between Üçyol and Fahrettin Altay in the southwest of İzmir Bay. Excavation of the tunnel is carried out by using eigth service shafts. The dominant rock units on the tunnel line are andesites indicate low and medium ranked metamorphoses which are known as Yamanlar ebonites, agglomerate-sandstone-siltstone-claystone serie which is known Altındağ Formation, yellowy brown flysch and grayish black flysch which are known as Bornova Complex. Various rock mechanic experiments are made on the samples which were taken from these units. In addition to these studies, previous geotechnical studies on the same tunnel route are analyzed. Judging by the acquired data, determining the strength of rock materials by Uniaxial Compressive Strength Test according to standarts (ISRM and ASTM) are defined as very difficult, time-consuming and even impossible in some cases, especially in Bornova Complex (laminar-foliative-unisotropic). Although Uniaxial Compressive Strength Test, Point Load Strength Test and Schmidt Hammer Tests were made on appropriate samples and tunnel faces (only a few) , results are not satisfying. For this reason, Block Punch Strength

Index was used for determining the strength of rock material. Rock mass is classified, parameters are calculated and used in numerical modelling by using these data.

1 GİRİŞ

Günümüzde metrolar kentsel ulaşım sistemleri olarak değerlendirilmektedir. Yenilenen ve gelişen şehircilik kavramları arasında ise metro, artık bir yaşam biçimidir. Amacı, kent insanını bir yere ulaştırmak değil, büyük bir kenti yaşanabilir kılmaktır. Temel hedef vatandaşların zaman ve yarar değerlerini en üst seviyeye çıkarmak, rahat ve huzurlu bir ortamda, düzenli ve güvenli koşullarda hizmet sunabilmektir. Bunun için ise güvenlik tedbirlerini elden bırakmadan inşayı ekonomik bir biçimde mühendislik projeleriyle birlikte sürdürmek gerekmektedir.

Mühendislik projelerine yapılan yatırımlar oldukça riskli ve yüksektir. Özellikle madencilik sektöründe bu risk ve maliyet unsurları diğer sektörlere nazaran daha fazladır (Aksoy, 2009). Madenciliğin en önemli özelliklerinden biri geri dönüşün olmayışı ve bilinmeyene doğru bir çalışma oluşudur. Bu nedenle yapılacak olan yatırımlar üzerindeki risk faktörünü minimuma indirmek için daha önceden küçük modeller üzerinde çalışma ve araştırmaların yapılması faydalı olmaktadır (Aksoy, 2009). Bu çalışmalar sonucunda proje için, teknik ve ekonomik açıdan en uygun sonuçlar elde edilmektedir.

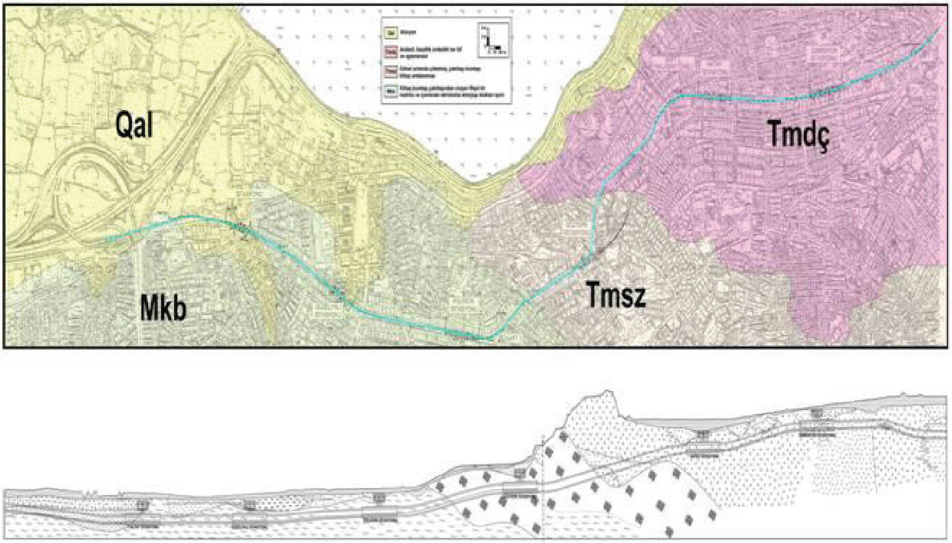
Kayaçları mühendislik işlerinde kullanırken, çeşitli özelliklerinin bilinmesi gerekmektedir. Bu özellikleri tayin edebilmek için laboratuar ve arazide birtakım deneyler yapılmaktadır. Bu deneylerde koşullar değiştirilebilmekte ve parametrelerin belirlenebilmesi için ayrıntıya girilebilmektedir. Ancak unutulmaması gereken bir konu da, bu deneyler laboratuar koşullarında standartlar doğrultusunda yapılmaktadır. Oysa doğanın yapısı çoğu zaman ideal olmamakta ve laboratuar koşullarıyla uyuşmamaktadır. Bundan dolayı yapılan çalışmalarda tam anlamıyla doğru sonuçlar elde edilememektedir.

2 GÜZERGÂHIN JEOLOJİSİ

İzmir Hafif Raylı Sistemi (İZRAY) güzergâhının Üçyol-F.Altay aralığını oluşturan jeolojik birimler egemen olarak Yamanlar volkanitleri ve Alüvyon, Altındağ Formasyonu ve Bornova karmaşığından kuruludur. Bu birimler yüzeyde yapay dolgu ile örtülüdür. Proje güzergâhının sonlarına yakın bölümlerde geniş alanlar kapsayan Kuvaterner yaşlı alüvyon, çakıllı killi kum, çakıllı kumlu kil ve kumlu killi çakıl seviyelerinden oluşur. Çakıllı kumlu kil; yeşilimsi kahverengi-koyu gri, orta katı-katı, orta-yüksek plastisiteli olup yer yer organik kökenli seviyeler içerir. Çakıllı killi kum; yeşilimsi kahverengi, ince-iri taneli, çok az çakıllı, orta sıkı-sıkı özelliktedir. Kumlu killi çakıl; kil ve kum birimleri ile grift olarak gözlenen çakıl birimi kahverengi, killi, kumlu, sıkı-çok sıkıdır (Aksoy vd. 2006). Güzergahın jeolojisi ve boyuna kesiti Şekil 1'de verilmektedir (Onargan ve Aksoy, 2006).

3 DENEYLER

Deneylerde, İzmir Metrosu 2. Aşama Çalışmaları'nın tünellerinden belirli kilometrelerde alınan Çört, Sarımsı Kahverengi Filiş, Silt Taşı, Gri Andezit, Kil Taşı, Pembe Andezit ve Grimsi Siyah Filiş örnekleri kullanılmıştır. Bu örneklerden, laboratuvarda karot alma makinesiyle karotlar alınmış ve bu karotlar standartlara uygun boyutlarda karot kesme makinesiyle kesilmiş ve parlatılmıştır. Hazırlanan bu örnekler üzerinde rutin kaya mekaniği deneyleri yapılmıştır. Bu deneylerde ISRM (2007) tarafından önerilen yöntemler kullanılmıştır. Çizelge 1'de deneyler sonucu elde edilen değerler verilmiştir.



Şekil 1. Metro güzergâhı jeolojisi ve boyuna kesiti (Onargan ve Aksoy, 2006)

Çizelge 1. Güzergah Boyunca Kayaç Malzemesi Üzerinde Yapılan Deneyler ve Sonuçları

Jeolojik Birim	**Kayaç Tipi**	**Mineral Tane Yoğunluğu (gr/cm^3)**	**Kaba Yoğunluk (gr/cm^3)**	**Ortalama Su Emme Oranı (%)**	**Tek Eksenli Basma Dayanımı (MPa)**	**Suda Dağılmaya Karşı Dayanımı (%)**	**Ortalama Schmidt Sertliği**
Yamanlar Volkaniti	Gri Andezit	2,61 ±0,005	2,64 ±0,087	2,51	47,15	97,35	33,12
Altındağ Formasyonu	Kil Taşı	2,51 ±0,068	2,62 ±0,062	-	14,05	-	34,25
	Çamurtaşı	2,70 ±0,017	2,47 ±0,052	0,54	48,80	95,95	44,83
	Silttaşı	2,68 ±0,003	2,47 ±0,019	4,17	51,10	96,92	48,40
Bornova Karmaşığı	Sarımsı-Kahverengi Filiş	2,11 ±0,010	2,26 ±0,031	5,48	11,57	71,10	22,60
	Grimsi Siyah Filiş	2,12 ±0,006	2,41 ±0,025	-	-	77,40	24,95

4 KAYA MALZEMESİNİN DAYANIMININ BELİRLENMESİNDE YAŞANILAN ZORLUKLAR

Mühendislik tasarımlarında kayaçların dayanım özelliklerinin belirlenmesinde en çok kullanılan parametre T.E.B.D.'dır. Bazı durumlarda T.E.B.D. indirek yollarla belirlenebilmektedir. Indirek yöntemlerin başında Nokta Yük Dayanım İndeksi (N.Y.D.İ) ve Schmidth Çekici (S.Ç.) testi gelmektedir. Ancak bazı durumlarda, standartlara ve önerilmiş yöntemlere uygun boyutta numune hazırlamak oldukça zor ve zaman alıcı olmakta ve bazende uygun numune hazırlama olanağı olsada sağlıklı sonuçlar elde edilememektedir. Özellikle çok zayıf, ayrışma derecesi yüksek, laminalı, foliasyonlu, ileri derece eklemli, şistoziteli, melanj tipi yapıya sahip, fliş tipi kayalarda uygun numune hazırlamakta çoğu zaman büyük problemlerle karşılaşılabilmektedir. Bu gibi durumlarda, kayacın dayanım parametresini belirlemek amacıyla son 10 yıl içerisinde D.M.D.İ. öne çıkmıştır. Literatüre bakıldığında, D.M.D.İ.'nin tarihsel gelişimi, test prosedürü, yenilme mekanizması vb hakkında geniş bilgi bulunabilir (Mazanti ve Sowers, 1965; Vutukuri Lama, 1974; Stacey, 1980; Lacharite, 1960; Van der Schrier, 1988; Gokceoglu, 1997; Ulusay ve Gokceoglu, 1997; Sulukcu ve Ulusay, 2001; ISRM, 2007; Ulusay vd., 2001; Sönmez ve Tunusluoglu, 2008). Bu konuda yapılan son çalışmada Hoek-Brown kaya sabiti (m_i) kullanılmıştır (Sönmez ve Tunusluoglu, 2008). Çalışmanın yapıldığı güzergâhta, yer altı suyunun da bulunuyor olması, zaten oldukça zayıf olan Sarımsı-Kahverengi filiş ve Grimsi-Siyah filişin dayanımını daha da düşürmüştür. Bu kayaçtan T.E.B.D. için çoğu zaman standartlara uygun numune hazırlanamamıştır. N.Y.D.İ. için yapılan testlerde genellikle numune kırılmadan ezilmiştir. S.Ç. ile yapılan denemelerde ise anizotropiden dolayı farklı sonuçlar elde edilmiştir. Güzergâhın belli bölümlerinde görece olarak daha sağlam kayaçlar geçildiğinde, buralardan alınan numunelerde (daha önce tablolardan verildiği üzere) standart deneyler yapılabilmiş ancak şartların uygun olmadığı lokasyonlarda D.M.D.İ. tercih edilmiştir. Bu çalışmada ISRM (2007) tarafından önerilen ve Ulusay vd. (2001) tarafından geliştirilen aşağıdaki formül kullanılmıştır.

$$D.M.D.\dot{I}._c = 3499 D^{-1,3926} t^{-1,1265} F_{t,D} \quad (6)$$

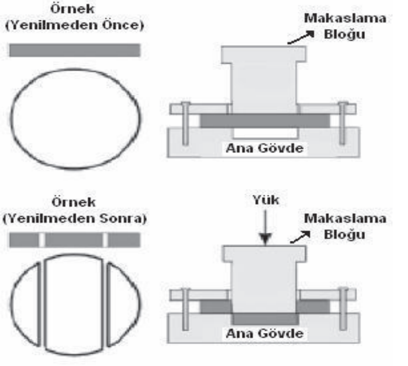
Burada D disk çapı (mm), t disk kalınlığı (mm), F yenilme kuvveti (kN) ve D.M.D.İ. MPa'dır.

5 D.M.D.İ. (DİSK MAKASLAMA DAYANIM İNDEKSİ) İLE KAYAÇ MALZEMESİ DAYANIMININ BELİRLENMESİ

Kaya dayanımı, özellikle tek eksenli basınç dayanımı, kaya kütlesi sınıflama sistemlerinde ve değişik türde kaya mühendisliği tasarımlarında önemli bir parametredir. Dayanımın tayini için standartların veya önerilmiş yöntemlerin belirttiği boyutlarda silindirik örneklerin hazırlanması gerekmektedir. Ancak kayaların sık aralıklı tabakalanma, lamina, şistozite yüzeyi vb. gibi süreksizliklerle bölünmüş olması halinde, dayanım deneyleri için uygun boyutlarda örnek hazırlanamamaktadır.

Yukarıda belirtilen örnek hazırlamayla ilgili güçlük ve sınırlamaları giderebilecek ve daha küçük örneklerin kullanılabileceği indeks deneyler her zaman ilgi çekici olmuştur. Bu amaçla, Hollanda'nın Delft Üniversitesi'nde yapılan bir düzenekle D.M.D.İ. deneyi ilk kez gündeme gelmiştir (Van der Schrier, 1988). Ancak ince disk şeklindeki örneklerin kullanıldığı bu araştırmada, az sayıda kaya türü üzerinde çalışılmış ve deneyde örnek boyut etkisi ile D.M.D.İ. 'nin kullanım alanları dikkate alınmamıştır.

Çalışmalar sonucunda, diğer indeks deneylerle karşılaştırıldığında, T.E.B.D.'nin D.M.D.İ. 'den daha az bir hata payıyla dolaylı olarak belirlenebileceği ortaya konmuştur. Ayrıca bu deneyde boyut düzeltme faktörleri ve kaya malzemesinin dayanıma göre sınıflandırılmasında D.M.D.İ.'nin alternatif bir parametre olabileceği de önerilmiştir. Bu deney; standartlara uygun şekilde örnek hazırlanamayan ve özellikle zayıf, kırıklı ve içerdiği sık aralıklı süreksizlikler nedeniyle dilimler halinde ayrılabilen kayaçlardan hazırlanmış disk şeklindeki örneklerin D.M.D.İ'nin tayin edilmesi ve D.M.D.İ değerinden tek eksenli basınç dayanımının belirlenmesi amacıyla yapılır. Deney aletinin şekli Şekil 2'de verilmektedir (Ulusay, 1997).



Şekil 2. Deney aletinin şekli.

Çizelge 2. BPI değeri.

Jeolojik Birim	Kayaç Tipi	BPI Değeri (MPa)	Tahmini T.E.B.D. (MPa)
Yamanlar Volkaniti	Gri Andezit	9,69±5,34	49,42
Altındağ Formasyonu	Kil Taşı	2,55 ±0,75	13,01
	Çamurtaşı	9,54 ±3,70	48,65
	Silttaşı	7,89 ±3,43	40,24
Bornova Karmaşığı	Sarımsı Kahverengi Filiş	2,15 ±1,15	10,96
	Grimsi Siyah Sert Filiş	3,21 ±1,48	16.37

6 SONUÇ

Kaya kütlelerinin davranışlarının belirlenmesinde kaya malzemesi önemli bir parametredir. Bu parametrenin belirlenmesinde en çok kullanılan yöntem tek eksenli basınç deneyidir. Zaman zaman T.E.B.D.'nın dolaylı olarak belirlenmesinde N.Y.D.İ. ve S.Ç. kullanılabilmektedir. D.M.D.İ. ise özellikle zayıf-çok zayıf kayaçlarda ortalama 5,1 dönüşüm katsayısı ile son yıllarda öne çıkan bir indeks deneyidir. Laminalı-foliasyonlu, anizotrop kayaçlarda kaya malzemesinin dayanımının belirlenmesinde D.M.D.İ. büyük kolaylık sağlamaktadır. Bu çalışmada, foliasyonlu, laminalı Bornova karmaşığında, kayaç malzemesinin dayanımının tespitinde D.M.D.İ. kullanılmış ve sonraki aşamada yapılan çalışmalarda bu veriler kullanılmıştır.

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The Prediction of Coal Bond Work Index Values by Using Point Load Index and Shore Hardness

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ABSTRACT Coal is a widely used energy source in producing electricity all over the world. Coal processing is a main step in coal utilizing process. In this process, grindability of coal has an important role in point of technical and economical aspects.

In this study, the correlation and relation between Bond work index (BWI), Shore hardness (SH) and point load index ($Is_{(50)}$) which are main engineering parameters of coal and other comminuted rocks was investigated. For this aim, Bond grindability tests, Shore hardness tests and point load tests were performed on six different coal (lignite) samples collected from different coal fields of Turkey. From the evaluation of test results, a logarithmic correlation with r=0.88 between BWI-SH and a linear correlation with r=0.69 between BWI- $Is_{(50)}$ were carried out. Additionally, a multi regression analysis was performed and a high correlation with r=0.94 was found between these parameters. From these results, it can be said that Shore Hardness and point load index values could be speedily, practically and economically used to predict the Bond work index rather than the standard Bond work index.

1 INTRODUCTION

Coal is the single most abundant source of energy available in most nations around the world. The depletion of high grade coal deposits compounded with the Clean Air Act, which puts limit on the amount of sulfur dioxide emissions, leads to the demand for coal preparation to meet the product quality suitable for world markets. The modernization of mining technologies, on the other hand, increases amount of frees in run-of-mine coal and puts the pressure on coal preparation. The use of coal as a run of mine causes various environmental problems and it is an inefficient way for energy production. For that reason, coal preparation must be the first step of coal usage.

Grindability is based on several techniques in order to measure of comminution characteristics which used to evaluate the crushing and grinding efficiency in mineral processing operations. Improving of grinding conditions and efficiency meaningfully affects the comminution cost and total energy producing economy. In this view, the selection of comminution equipment properly is another important aspect in mineral processing. In this selection, many methods are used to characterize the communed material. As known, one of them is Bond work index which was proposed by F.C. Bond 50 years ago and it is used in designing of ore preparation plants. Bond work index is determined by applying the Bond grindability test procedure (Bond 1960). However, this procedure requires time consuming and difficult laboratory studies.

In the design of grinding circuits in a mineral processing plant, the Bond method is widely used for a particular material in dimensioning mills, determining power needs and in the evaluation of performance. Its use as an industrial standard is very common, providing satisfactory results in all industrial applications. Despite having many advantages, this method has some drawbacks, such as being tedious and time consuming, and also requiring a special mill.

In order to found solution to rock mechanics problems, it is very important to know the rock formation properties. Parameters such as crystallizing degree of rock, water absorption, void water pressure, discontinuities, weathering degree mainly affect mechanical properties of rocks samples used in laboratory experiments do not exactly represent the field formation but given an information. The most important mechanical properties are UCS, tensile strength, shear strength, point load strength, and the rock mass classifications made by weighted sum of these properties (Köse and Kahraman, 1999). Hardness is one of the physical properties of rocks and the Shore scleroscope hardness (SH) is a suitable and economical method widely used for estimating rock hardness.

Several test methods are improved in the field of coal workability determination by mechanics of excavation. They consider point load strength index ($Is_{(50)}$) and Shore hardness (SH) can be mentioned of easy examples. In grindability tests, several difficulties are found in standard processes. Bond grindability tests are used to provide basic data for designing industrial comminution circuits. The Bond work index (BWI) of an ore is determined by Bond grindability Tests that indicate resistance tike to comminution.

Özer and Çabuk (2006), Bond grindability test, uniaxial compression, indirect uniaxial tensile, Los Angeles abrasion, point load index, sound velocity, Shore hardness and Schmidt hammer tests were performed on four different limestone and two different chromites samples. The relations between Bond Work index and the other test results were investigated.

The work index is very important but its determination is time consuming and requires skilled staff and 10 kg of specially prepared feed sample. However, tests are carried out on five coal samples of different properties to determine the existence of any relation between Bond grindability and $Is_{(50)}$, and SH. In this study, the different interrelationships between bond grindability, work index and the two test methods mentioned above are found a good correlation within the mean relative error values. From these relationships coal grindability and work index values can be determined with a good prediction.

2 EXPERIMENTAL STUDIES

In this study, six lignite coals were collected in different region in Turkey. Chemical properties of coal were given in Table 1. To predict of coal Bond work index values, the standard laboratory Bond test were carried out on six different coals samples. Then, as hardness and strength properties of coals were applied Shore scleroscope hardness and point load strength index.

Table 1. Chemical properties of coals (Original form).

Coal	Total Moisture (%)	Ash (%)	Volatile Matter (%)	Total Sulphur (%)	Net Calorific Value (Cal/g)
Dodurga	9.85	21.76	31.34	2.19	3978
Elbistan	24.89	25.77	29.74	2.09	2162
Aydın	12.95	7.64	33.98	0.40	4312
Tunçbilek	11.79	19.58	31.93	2.82	4657
Manisa Soma	13.23	14.65	34.28	0.82	4464
Kısrakdere	13.60	8.09	35.74	0.63	4873

2.1 Bond Grindability Test

Laboratory Bond grindability tests were conducted with -3.35 mm dry feed materials in a standard ball mill (30.5 x 30.5 cm) following a standard procedure outlined in the literature (Yap et al., 1982; Deister, 1987; Ipek, 2003). The BWI was determined at a test sieve size of 106 μm. Mill has no lifters and all the inside comers are rounded. It is operated at 70 rpm and is equipped with a revolution counter. The grinding charge consists of 285 iron balls weighing 20.125 grams with a calculated surface area of 842 inc^2.

The standard Bond grindability test is a closed-cycle dry grinding and screening process, which is carried out until steady state conditions are obtained. This test was proposed by Bond and Maxson (1943) and used by different researcher (Yap et al, 1982; Austin and Brame, 1983; Magdalinovic, 1989). The material is packed to 700 cm^3 volume using a vibrating table. This is the volumetric weight of the material to be used for grinding tests. For the first grinding cycle, the mill is started with an arbitrarily chosen number of mill revolutions. At the end of each grinding cycle, the entire product is discharged from the mill and is screened on a test sieve (P_i). Standard choice for P_i is 106 micron. The oversize fraction is returned to the mill for the second run together with fresh feed to make up the original weight corresponding to 700 cm^3. The weight of product per unit of mill revolution, called the ore grindability of the cycle, is then calculated and is used to estimate the number of revolutions required for the second run to be equivalent to a circulating load of 250%. The process is continued until a constant value of the grindability is achieved, which is the equilibrium condition. This equilibrium condition may be reached in 6 to 12 grinding cycles. After reaching equilibrium, the grindability for the last three cycles is averaged as a Bond grindability index (G_B).

The products of the total final three cycles are combined to form the equilibrium rest product. Sieve analysis is carried out on the material and the results are plotted, in order to find the 80% passing size of the product (P_i). The coal samples are crushed by a laboratory scale jaw crusher and the standard Bond grindability test were performed in the laboratory. BWI values are calculated from Equation 1. G_B and BWI values are presented in Table 2.

$$BWI = 1.1 \frac{44.5}{P_i^{0.23} G_B^{0.82} \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)} \qquad (1)$$

where W_i is Bond work index in kWh/t, P_i is sieve size at which the test is performed in µm, G_B is Bond's standard ball mill grindability in g/rev, P_{80} and F_{80} are sieve opening at which 80% of the product and feed passes, respectively in µm.

Table 2. Grindability and Bond work index values of coals.

Coal	G_B, g/rev	BWI, kWh/t
Aydın Linyit	0.79	22.54
Manisa Soma	0.74	19.10
Kısrakdere	0.50	30.30
Elbisan	1.37	14.10
Tunçbilek	0.95	25.44
Dodurga	1.05	17.46

2.2 Shore Hardness Test

Hardness is one of the physical properties of rocks and the Shore scleroscope hardness (SH) is a suitable and economical method widely used for estimating rock hardness. This test measures the rebound hardness of a rock specimen by the impact and rebound of a small diamond tipped impactor. The impactor free falls through a smooth tube to strike the rock sample, which is held by a steel anvil. The height of the impactor rebound is a function of the rock rebound hardness. 20 impacts were carried out at each sample and the mean Shore hardness values of coals are given in Table 3.

Table 3. Shore hardness values of coals.

Coal	Shore Hardness, SH		
	Min.	Max.	Mean
Aydın Linyit	15.4	31.3	25.7
Manisa Soma	53.7	64.4	58.4
Kısrakdere	48.6	58.1	55.4
Elbisan	7.4	14.3	10.9
Tunçbilek	21.9	33.5	26.9
Dodurga	21.6	33.7	29.6

2.3 Point Load Strength Index Test

The point load strength index test is intended as an index test for the strength classification of rock materials. It may also be used to predict other strength parameters with which it is correlated, for example the uniaxial compressive and the tensile strength. The point load test has often been reported as an indirect measure of the compressive or tensile strength of rock (Reichmuth, 1968, Broch and Franklin, 1972, Bieniawski, 1975). It has been used widely in practice due to its testing ease, simplicity of specimen preparation, and field applications.

The testing machine consists of a loading frame, which measures the force required to break the sample, and a system for measuring the distance between the two platen contact points. It was measured in accordance with the procedures recommended in ISRM (2007), with block samples.

In a point load test, a sample of coal was mounted between two pointed platens and pressure was applied until failure of the sample occurs. The peak applied load was recorded and used to calculate the Point Load Index. Rock specimens in the form of cut blocks were broken by application of concentrated load through a pair of spherically truncated, conical platens. The applied force at failure of the sample and distance between the platen tips were recorded in order to calculate the point load index as Equation 2. The mean point load strength values of coal are given in Table 4.

$$Is_{(50)} = F\frac{P}{D_e^2} \qquad (2)$$

where F is size correction factor $(D_e/50)^{0.45}$, P is applied load in MN, D_e is equivalent diameter in mm $(4A/\pi)^{0.5}$, A is minimum cross sectional area of the specimen in mm^2.

Table 4. Point load strength index values of coals.

Coal	$Is_{(50)}$, MPa		
	Min.	Max.	Mean
Aydın Linyit	0.20	0.41	0.30
Manisa Soma	1.08	1.09	1.09
Kısrakdere	0.86	1.02	0.94
Elbisan	0.26	0.27	0.27
Tunçbilek	0.15	0.61	0.35
Dodurga	0.49	0.67	0.58

3 STATICAL ANALYSIS

Both simple and multi regression analysis were made to estimate the values of Bond work index of coal.

3.1 Simple Regression Analysis

In order to determine the relationship between BWI-SH and BWI-$Is_{(50)}$ binary graphics were done. It is seen that with the increase of Shore hardness values of the coal the BWI values rise logarithmically (Fig. 1). Also, the BWI values increase linearly with the increase of point load strength index values in Figure 2.

Bond work index values of coals used in this study can be estimated by using Equation 3 with the help of shore hardness of coals and Equation 4 with the help of point load strength index of coals with the correlation coefficients 0.88 and 0.69 respectively.

$$BWI = 5.721\ SH^{0.379} \qquad (3)$$

$$BWI = 11.1\ Is_{(50)} + 14.87 \qquad (4)$$

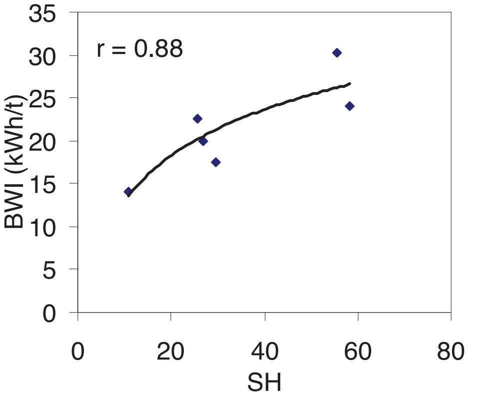

Figure 1. BWI versus Shore hardness.

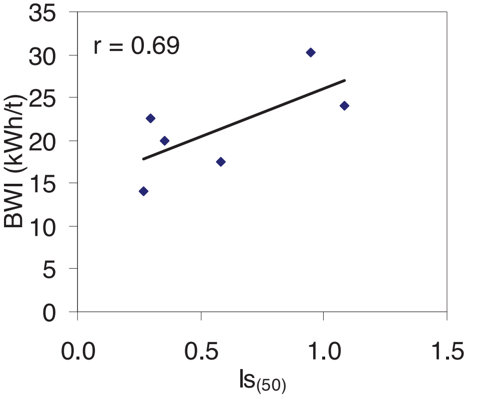

Figure 2. BWI versus point load strength index.

3.2 Multi Regression Analysis

To estimate the BWI values of coals by using multi regression analysis were done as independent variables shore hardness and point load strength index (Equation 5). Correlation coefficient of this model is very high (r=0.94). The cross-correlation graph of observed BWI values and BWI values obtained from Equation 5 by using SH and $Is_{(50)}$ data was given in Figure 3.

$$BWI = 11.743 + 0.676SH - 23.22Is_{(50)} \quad (5)$$

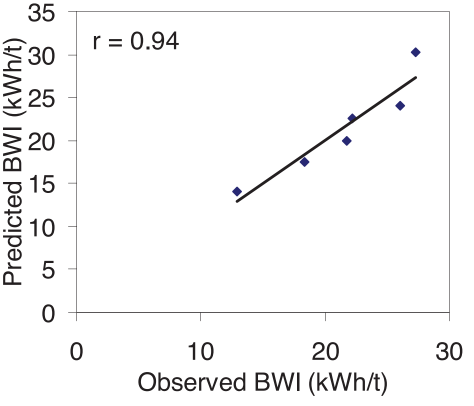

Figure 3. Cross-correlation of predicted and observed values of BWI.

4 CONCLUSION

BWI values that shows the grindability of coals has an important role in projects on the aspect of investment and planning. In order to estimate the BWI values standard Bond tests, Shore hardness tests and point load strength index tests were done on six different coals (lignite).

The increase in hardness and strength properties of coals causes the increase of energy consumption in grinding the coals.

The Bond work index values of coals can be estimated reliably by using only Shore hardness values and point load strength index values with Equation 3-4. Also, the BWI values may be predicted more reliably by using SH and $Is_{(50)}$ values together with Equation 5.

It can be said that Shore Hardness and point load strength index values could be speedily, practically and economically used to predict the Bond work index rather than the standard Bond work index.

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Zayıf Kaya Kütlelerinde Oluşturulan Dairesel Açıklık Aynası Etrafındaki Deformasyonların Tahmini

Prediction of the Deformations Around the Face of an Circular Opening in Weak Rock Mass

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ÖZET İlerleyen tünel aynası etrafındaki gerilme ve deformasyonlar açıklığın duraylılığı ile ilgili öneli bilgiler verdiğinden tünel aynası önü ve arkasındaki deformasyonların belirlenmesi çok önemlidir. Kapalı form çözümlemelerin olmayışı ve 2 boyutlu sayısal yöntemlerdeki sınırlamalar nedeni ile sözkonusu deformasyonların belirlenmesi için 3 boyutlu sayısal yöntemlerin kullanımı kaçınılmaz olmaktadır. Tünel aynası etrafındaki deformasyonların belirlenmesinden sonra uzunlamasına deformasyon profilinin oluşturulması mümkün olmaktadır.

Bu çalışmada, GSI kaya kütlesi karekterizasyon sistemi, FLAC3D adlı üç boyutlu sonlu farklar esaslı yazılım ve çoklu regresyon modelleme tekniği, uzunlamasına deformasyon profilinin oluşturulabilmesi için gerekli olan radyal yerdeğişimlerinin tespiti ve yerinde gerilmelerin, aynadan olan mesafenin, kaya kalitesinin ve radyal yerdeğişimleri arasındaki ilişkileri inceleyebilmek için kulanılmıştır.

Dört farklı derinlik için farklı kaya kütlelerinde oluşturulan dairesel açıklıkların ayna önündeki ve gerisindeki deformasyonların belirlenmesi için sekiz farklı çoklu regresyon modeli oluşturulmuştur. Oluşturulan regresyon eşitliklerinin kullanımı ile ayna ilerisi ve gerisindeki normalize edilmiş radyal yerdeğişimlerinin elde edilmesi ve anılan parametrelerin deformasyon profili üzerindeki etkileri anlaşılabilecektir. Önerilen eşitlikler burada varsayılan koşullarda oluşturulacak olan dairesel açıklık aynası etrafındaki deformasyonların tespitinde kullanılabilir. Saha mühendisleri bu çalışmada izlenen yolu kullanarak kendi özel durumları için benzer eşitlikleri üretebilirler.

ABSTRACT Since the stresses and the deformations around the advancing face of a tunnel give valuable information on the stability of an opening, the determination of the deformations occuring ahead and behind of the tunnel face is very important. Due to the unavailability of closed from solutions and limitations of 2D numerical modeling, for determination of these radial displacements the utilization of 3D numerical modeling becomes inevitable. Once, the radial displacements around the face of a tunnel are determined then longitudinal deformation profile of the tunnel can be constructed.

In this study, geological strength index, rock mass characterization system, a three dimensional finite difference code FLAC3D and multiple regression modeling technique were used to determine the radial displacements necessary for constructing the longitudinal deformation profile and also for assessing the relationship among in-situ stresses, the distance from the face of a tunnel, rock mass quality and radial displacement.

For four different depth, eight different multiple regression models were constructed for estimating the normalized radial displacements ahead and behind of the face of a circular

tunnel, excavated in different quality rock masses under hydrostatic stress conditions. Using the derived models the practical range of normalized radial displacements, occurred around the face of a tunnel for assumed ground conditions, can be assessed and the influence of the predefined parameters on the longitudinal deformation profile can be highlighted. The constructed models can be used for predicting the displacements around the face of an opening for assumed ground conditions here. Engineers can use the presented approach for their site-specific cases and for constructing the longitudinal deformation profile.

1 GİRİŞ

Kaya kütlesi ve malzemesinin dayanım ve deformabilite özellikleri, yerinde gerilmeler ve açıklık geometrisi açıklık duraylılığını etkileyen ana parametrelerdir. Başarılı bir tasarım ve yapı inşası için bu etkin parametrelerin hepside göz önünde bulundurulmalıdır. Özellikle zayıf kaya kütlelerinde açıklık duraylılığını korumak için uygun kazı ve tahkimat yöntemlerinin seçilmesi hayati önem taşımaktadır.

İlerleyen bir açıklık aynasının önü ve arkasındaki gerilme ve yerdeğişimleri açıklık duraylılığına ilişkin önemli bilgiler vermektedir. Empirik, analitik ve 2 boyutlu sayısal yöntemler gerekli tahkimat sisteminin belirlenmesinde ve açıklık etrafındaki gerilme ve deformasyonların belirlenmesinde yaygın olarak kullanılmalarına rağmen bazı belirgin eksikliklere sahiptirler.

Kaya kütlesi sınıflama sistemleri açıklık duraylılığını sağlamak için gerekli tahkimat sistemlerinin belirlenmesinde ve kaya kütle ve deformasyon parametrelerinin eldesinde sıklıkla kullanılmaktadırlar (Aksoy, 2008). Uygulamalarındaki basitlik ve kullanımlarındaki pratiklik nedenleriyle bu empirik yöntemler mühendisler tarafından tercih edilmektedirler. Ancak, bu sınıflama sistemleri ilerleyen ayna etrafındaki gerilmelerin ve deformasyonların kestiriminde sayısal bilgiler verememektedirler.

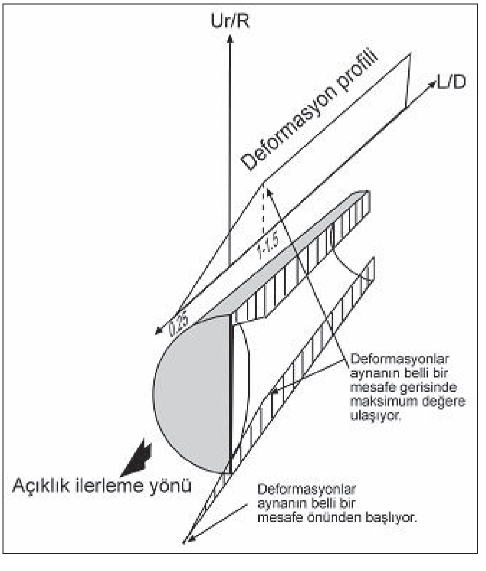
Literatürde farklı kaya kütlelerinde oluşturulan açıklıklar etrafındaki gerilme ve deformasyonların belirlenmesi için pek çok analitik yöntem önerilmiştir. Bu yöntemlerin bazılarında deformasyonların belirlenmesi probleminin basitleştirimi için bazı gerçekçi olmayan varsayımlarda bulunulmuştur (Brown ve ark., 1983).

Sayısal modelleme teknikleri tahkimat performansının kestirimi ve açıklık etrafındaki gerilme ve deformasyonların tespiti için yaygın olarak kullanılmaktadır (Detournay ve Fairhurst, 1987; Ozsan ve Karpuz, 1996; Asef and Reddish, 2000; Alanso ve ark., 2003; Basarir, 2006; Yavuz, 2006; Sari, 2007; Basarir, 2008).

Analitik yöntemlere benzer şekilde pek çok sayısal yöntem esaslı çözümlemedede asal gerilme bileşenlerinden birinin açıklığın uzun eksenine dik etkidiği düzlemsel birim deformasyon koşulları varsayılmaktadır. Bu iki boyutlu analizlerde açıklığın nihai deforme olmuş halinin ve mekanik değişikliklerinin belirlenmesi mümkündür (Ünlu ve Gercek, 2003). Ancak, düzlemsel birim deformasyon varsayımıyla açıklığın boylamasına kesitindeki kaya kütlesi davranışı ve mekanik özelliklerindeki değişimlerin saptanması mümkün olamamaktadır (Unlu ve Gercek, 2003). Dolayısıyla bu varsayım esaslı yaklaşımlarla uzunlamasına deformasyon profilinin oluşturulması mümkün değildir.

Uzunlamasına kesit hakkında fikir verebilen üç boyutlu analizler yardımıyla ilerleyen ayna etrafında açıklık oluşturumunun her aşamasında meydana gelebilecek olan deformasyonların gösterimi mümkündür. Açıklığın kullanım amacına bağlı olarak her projede izin verilebilir bir deformasyon oranı bulunmaktadır. Örneğin tünelcilik projelerinde bu oran %2 civarında olmakla birlikte madencilik amaçlı açıklıklarda daha yüksek olabilmektedir (Hoek, 2007). Izin verilebilir nispetteki deformasyonun gerçekleşmesini takiben yapılacak olan tahkimat proje ekonomisine ciddi katkıda bulunacaktır. Aynaya belli bir mesafeden itibaren tahkimat yapımı açıklık duraylılığını bozmadan tahkimat yapımını mümkün kılacak bu yollada proje maliyetine katkıda bulunacaktır. Ancak bunların gerçekleştirilebilmesi için deformasyon profilinin oluşturulması gereklidir. Bu sayede açıklık duraylılığını bozmadan tahkimatın yapımın hangi aşamasında kurulabileceği hakkında fikir edinilebilmesi mümkün olabilmektedir.

Deformasyonların aynanın belirli bir mesafe ilerisinden başlayarak ayna gerisinden belli bir mesafeden sonra azami değerine ulaştığı bilinmektedir (Şekil 1). İlerleyen açıklık aynası etrafındaki gerilme ve deformasyonların elde edilebilmesi için üç boyutlu sayısal modelleme kullanımı gereklidir. Diğer bir deyişle açıklık aynasının üç boyutlu etkisi hesaba katılabilir ve tahkimat yerleştiriminden önceki radyal yerdeğişimlerini belirlenebilir. Bu radyal yerdeğişmlerinin eldesinden sonra açıklık deformasyon profili elde edilebilir. Günümüze kadar farklı araştırmacılar deformasyon profilinin oluşturumu için farklı eşitlikler önermişlerdir.



Şekil 1. Dairesel açıklık için uzunlamasına deformasyon profili.

Panet ve Guenot (1982) ve Panet (1993) sonlu elemanlar yöntemi esaslı sayısal modelleme tekniği yardımıyla hidrostatik gerilme koşulları için takip eden eşitliği önermişlerdir.

$$\frac{u_r}{u_{r\infty}} = 0.28 + 0.72\left[1 - \left(\frac{0.84}{0.84 + x/R}\right)^2\right]$$

Corbette ve ark., (1991) aynı varsayımlarla deformasyon profilinin eldesi için takip eden eşitliği önermişlerdir,

$$\frac{u_r}{u_{r\infty}} = 0.29 + 0.71\{1 - \exp[-1.5(x/R)^{0.7}]\}$$

Panet (1995) tarafından önerilen diğer bir alternatif eşitlikse;

$$\frac{u_r}{u_{r\infty}} = 0.25 + 0.75\left[1 - \left(\frac{0.75}{0.75 + x/R}\right)^2\right]$$

Carranza-Torres ve Fairhurst (2000) Hoek tarafından önerilen ve Mingtan Güç santrali projesinden (Chern ve Shiao, 1998) elde edilen verilere dayalı bir eşitlik önermişlerdir.

$$\frac{u_r}{u_{r\infty}} = \left[1 + \exp\left(\frac{-x/R}{1.10}\right)\right]^{-1.7}$$

Bu eşitliklerde $x \geq 0$ aynadan itibaren olan mesafe, u_r radyal deformasyon ve $u_{r\infty}$ ise azami veya düzlemsel birim deformasyon varsayımıyla elde edilen radyal deformasyon değeridir.

Yakın zamanda Unlu ve Gercek (2003) varsayılan koşullar için deformasyon profilindeki Poisson's etkisini inceleyen bir çalışma gerçekleştirmişler dairesel açıklığın ayna gerisindeki deformasyon profilinin eldesi için takip eden eşitliği önermişlerdir;

$$\frac{u_r}{u_{r\infty}} = u_0 + A_b\left\{\left[B_b/(B_b + (x/R))\right]^2\right\}$$

Burada $u_0 = 0.22\upsilon + 0.19$, $A_b = -0.22\upsilon + 0.81$ and $B_b = 0.39\upsilon + 0.65$

İlk üç eşitlik elastik analiz için geçerlidirler. Son eşitlik hariç diğer eşitliklerin tamamında $u_r/u_{r\infty}$ oranının sadece aynaya olan normalize edilmiş mesafenin fonksiyonu olduğu varsayılmaktadır, kaya kütle ve malzeme özelliklerinin bu orana etkileri açık belirtilmemiştir.

Bu çalışmanın amacı anılan parametrelerden etkilenen ayna ilerisi ve gerisinde oluşan radyal deformasyonların kestirimi ve deformasyon profilini oluşturulmasıdır. Üç boyutlu sonlu farklar esaslı yazılım kullanılarak farklı kaya kütlesinde ve derinliklerde oluşturulan dairesel açıklığın aynasının etrafındaki deformasyonlar belirlenmiştir.

Çok hassas bir yöntem olmasına rağmen her bir durum için sayısal model oluşturulup çözüm alınması çok zaman alıcı olmaktadır. Çoklu regresyon modellemesinin kullanımı bu uzun ve zaman alıcı yöntemin alternatifi ve tamamlayıcısı olarak kullanılabilmektedir. Sayısal modellemeden elde edilen veriler ayna civarindaki radyal deformasyonların kestirimini sağlayabilecek olan regresyon modeli oluşturulabilmesi için kullanılmıştır.

2 KAYA KÜTLESİ KARAKTERİZASYONU

Kaya kütle özellikleri açıklık duraylılığı kestiriminde önemli rol oynar. Kaya malzemesi ve süreksizliklerin detayları mühendislik jeolojisi kaya kütle tanımlamalarında kapsanmalıdır. Bozunma, yapı, renk, tane boyutu, tek eksenli basma dayanımı, ve yönelim, devamlılık, aralık, ayrıklık, dolgu dalgalılık gibi süreksizlik özellikleri tam bir kaya kütle tanımlanmasında içerilmelidir. Sonuç olarak kaya kütlesi blok boyutu, şekli ve süreksizlik koşulları ile tanımlanabilmektedir. Anılan bu parametrelerin hepsi GSI sistemi tarafından kapsanmaktadır. Dolayısıyla, bu çalışmada her bir parametrenin ayrı ayrı değerlendirilmesindense tüm bu parametreleri beraberce içeren ve kaya kütlesinin tamamen tanımını yapabilen GSI sistemi kullanılmıştır.

GSI sistemi Hoek ve ark., (1995) tarafından önerilmiştir. Kaya kütlesinin genel görünüşünü (çok iyi, iyi gibi) ve kaya kütle yapısını (bloklu, ayrık) esas alan bir sistemdir. 2002 yılında Hoek ve ark., (2002) folyasyonlu/laminalı kaya kütle yapısını ekleyerek sistemi modifiye etmişlerdir. Yakınlarda Marinos ve ark., (2005) GSI sisteminin uygulanabileceği ve uygulanamayacağı alanları belirleyen bir çalışma yapmışlardır. Bu çalışmada GSI sisteminin 2002 versiyonu kullanılmıştır. Varsayılan kaya kütle tanımlama ve GSI değerleri Çizelge 1'de verilmiştir.

3 KAYA KÜTLE VE MALZEME ÖZELLİKLERİ

Mühendislik tasarımlarının ilk aşamalarında kaya kütlesi dayanım ve deformabilite parametrelerinin elde edilmesi gerekir (Basarir, 2006). Pek çok araştırmacı bu parametrelerin elde edilebilmesi için eşitlikler önermişlerdir. Bu empirik eşitlikler projenin erken aşamalarında tasarımcılara yardım amaçlı olarak önerilmişlerdir. Proje başladıktan sonra yapıdaki deformasyon değerleri kritik seviyeye ulaştı ise tasarımcı yeniden değerlendirme yapmak hatta yerinde ölçümleri başlatmak zorundadır (Hoek, 2007).

Çizelge 1. Varsayılan GSI, σ_{ci}, m_i değerleri ve hesaplanan kaya kütlesi Hoek Brown sabitleri.

Kaya kütle sınıfi	GSI	m_i	σ_{ci}, MPa	m_m	s_m	a
	10	7	1	0.281	0.000045	0.585
Çok	10	25	5	1.005	0.000045	0.585
	15	7	1	0.336	0.000100	0.561
Zayıf	15	25	5	1.201	0.000100	0.561
	20	7	1	0.402	0.000100	0.544
	20	25	5	1.436	0.000100	0.544
	25	7	5	0.481	0.000200	0.531
	25	25	25	1.717	0.000200	0.531
Zayıf	30	7	5	0.575	0.000400	0.522
	30	25	25	2.052	0.000400	0.522
	35	7	5	0.687	0.000700	0.516
	35	25	25	2.453	0.000700	0.516

Bu çalışmada Hoek-Brown (Hoek ve ark., 2002) yenilme kriteri kullanılmıştır.

$$\sigma_1 = \sigma_3 + \sigma_{ci}\left(\frac{m_b \sigma_3}{\sigma_{ci}} + s\right)^a$$

σ_1 ve σ_3 maksimum ve minimum asal gerilmeler, σ_{ci} kaya malzemesinin tek eksenli basma dayanımı, m_b, s ve a ise Hoek Brown sabitleridir.

3.1 Tek Eksenli Basma Dayanımı

Sayısal modellemedeki kaya kütlesi dayanım indirgenmesi GSI ın fonksiyonları olan m, s ve a sabitleri tarafından otomatik olarak yapıldığından tek eksenli basma dayanımınn belirlenmesi önemlidir. ISRM (2007) tarafından tek eksenli basma dayanımının çok zayıf ve zayıf kayaçlar için 1-5 ve 5-25 arasında değiştiği belirtilmektedir. Bu çalışma için varsayılan tek eksenli basma dayanım değerleri ve kayaç kaliteleri Çizelge 1'de verilmektedir.

3.2 Hoek Brown Sabitleri

Sağlam kayanın m değeri (m_i) farklı yanal basınçlarda yapılan üç eksenli basma dayanım deneyinden elde edilebilir. m_i değeri sağlam kayanın mineralojisine, kompozisyonuna ve tane boyutuna bağlıdır. mi değerleri en iyi ve en kötü koşulları karekterize edebilecek şekilde 7 ve 25 olarak varsayılmıştır. GSI ve mi değerlerine bağlı olarak kaya kütlesinin m değeri (m_m), s değeri ve a değeri aşağıdaki eşitliklerle hesaplanmaktadır (Hoek ve ark., 2002).

$$m_m = m_i \exp\left(\frac{GSI-100}{28} - 14D\right)$$

$$s = \exp\left(\frac{GSI-100}{9} - 3D\right)$$

$$a = \frac{1}{2} + \frac{1}{6}\left(e^{-GSI/15} - e^{-20/3}\right)$$

Eşitliklerde D örselenme faktörüdür kaya kütlesinin hangi ölçüde patlatma veya gerilme salınımı kaynaklı örselendiğinin göstergesidir. Bu çalışmada kaya kütlesinin örselenmediği varsayılarak bu faktör 0 olarak varsayılmıştır.

Varsayılan m_i değerleri ile hesaplanan kaya kütlesi m, s ve a katsayıları Çizelge 1'de verilmektedir.

3.3 Kaya Kütlesi Deformasyon Modülü

Kaya kütlesinin deformasyon modulunun (E_{mass}) yerinde deneylerle belirlenmesi zor ve pahalı bir işlemdir. Dolayısıyla, genelde E_{mass}'in empirik eşitliklerle hesaplanması tercih edilmektedir. Hoek ve Diederichs (2006) E_{mass} hesaplanması için takip eden eşitliği önermişleridir. Hesaplanan değerler Çizelge 2'de verilmektedir.

$$E_{mass} = 100000\left(\frac{1-D/2}{1+e^{((75+25D-GSI)/11)}}\right)$$

Çizelge 2. Kaya kütlesi dayanım ve deformabilite parametreleri.

Kaya kütle sınıfı	GSI	m_i	σ_{ci}, MPa	E_{mass}, MPa	c, MPa	ϕ, °	σ_{cm}, MPa
	10	7	1	100	0.026	12.36	0.043
Çok	10	25	5	224	0.081	30.88	0.451
	15	7	1	133	0.032	13.62	0.055
Zayıf	15	25	5	298	0.099	33.05	0.560
	20	7	1	178	0.038	14.82	0.067
	20	25	5	398	0.115	34.98	0.667
	25	7	5	530	0.082	25.79	0.395
	25	25	25	1186	0.223	48.69	3.866
Zayıf	30	7	5	707	0.093	27.48	0.456
	30	25	25	1581	0.251	50.44	4.403
	35	7	5	942	0.104	29.08	0.519
	35	25	25	2108	0.278	52.01	4.953

3.4 Kaya Kütlesi Dayanımı

Kaya kütlesi kohezyon ve içsel sürtünme açısı tahmini için minimum asal gerilmenin $\sigma_t<\sigma_3<\sigma_{3max}$ aralığında olduğu durumda yenilme kriteri eşitliğinin çözümü ile üretilen noktalardan lineer doğru geçirilmesi gereklidir.

$$c_m = \frac{\sigma_{ci}\left[(1+2a)s+(1-a)m_m\sigma_{3n}\right](s+m_m\sigma_{3n})^{a-1}}{(1+a)(2+a)\sqrt{1+\frac{6am_m(s+m_m\sigma_{3n})^{a-1}}{((1+a)(2+a))}}}$$

$$\phi_m = Sin^{-1}\left[\frac{6am_m(s+m_m\sigma_{3n})^{a-1}}{2(1+a)(2+a)+6am_m(s+m_m\sigma_{3n})^{a-1}}\right]$$

Eşitlikteki $\sigma_{3n}=\sigma_{3max}/\sigma_{ci}$.

Hoek ve ark., (2002) global kaya kütlesi dayanımının bulunabilmesi için takip eden ilişkiyi önermişlerdir.

$$\sigma_{cm} = \frac{2c_m Cos\phi_m}{1-Sin\phi_m}$$

Kaya kütlelerine ait hesaplanmış c_m, ϕ_m and σ_{cm} değerleri Çizelge 2'de verilmiştir.

4 SAYISAL MODELLEME

Sayısal modelleme yazılımı olarak üç boyutlu sonlu farklar yöntemi esaslı FLAC 3D (Itasca, 2005) yazılımı seçilmiştir. Bu yazılım direk olarak doğrusal olmayan yenilme zarfını kullanabilmektedir, dolayısıyla Mohr-Coulomb parametrelerini tahmin etmek yerine direk Hoek-Brown yenilme kriteri kullanılabilmektedir. Yazılımın uyarlanabilirliği sayesinde yerinde gerilmeler, açıklık geometrisi, derinlik gibi doğal ve mühendislik faktörlerinin etkenleri incelenebilmektedir. Yazılımın üç boyutlu olması sayesinde dairesel açıklığın uzunlamasına deformasyon profilide elde edilebilmektedir.

Girdi parametreleri sağlam kayanın tek eksenli basma dayanımı (σ_{ci}), kaya kütlesinin deformasyon modülü (E_{mass}), Poisson's oranı (v) ve Hoek brown sabitleridir (m_m, s, a). Kaya kütlesi dayanım veya yenilmesi Hoek-Brown yenilme kriterine gore değerlendirilmektedir.

Bu çalışmada açıklığı çevreleyen kaya kütlesinin kesişen süreksizlikler boyunca meydana gelen hareketler sonucu sıfır hacimsel değişimin gerçekleştiği elastik-mükemmel plastik malzeme davranışı sergilediği varsayılmıştır (Duncan-Fama, 1995). Modellenen kaya kütlesinin homojen ve isotropik ortam olduğu düşünülmüştür. Her iki varsayımda bu çalışmada örneklenen çok zayıf ve zayıf kaya kütleleri için geçerli olmaktadır (Hoek ve Brown, 1980).

Yerinde düşey gerilmenin hesabında aşağıdaki formülden faydalanılmıştır.

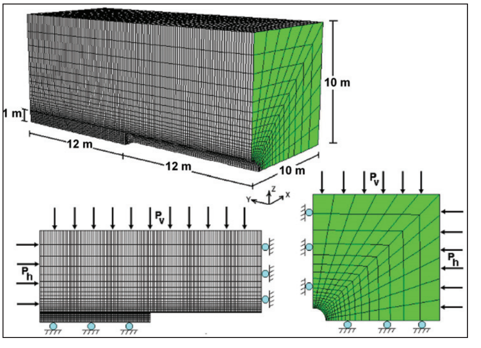
$$Pv = \gamma H$$

Bu çalışmada hidrostatik gerilme koşulları varsayıldığından düşey (Pv) ve yatay (Ph) yerinde gerilmeler birbirlerine eşit olarak alınmıştır. 100, 200, 300 ve 400 metre derinlikler için hesaplanan yatay ve düşey gerilmeler sırasıyla 2.7, 5.4, 8.1 ve 10.8 MPa dır.

Çok zayıf ve zayıf kalitede kaya kütleleri için GSI değerleri 10, 15, 20 ve 25, 30, 35 olarak alınmıştır. Her bir GSI değeri için en kötü ve en iyi durumları temsilen ikişer adet m_i ve σ_{ci} değeri kullanılmıştır. Bu değerler sırasıyla 1, 7 ve 5, 25 MPa dır.

Kullanılan kaya kütlesi sınıfları, GSI değerleri, Hoek Brown sabitleri, global kaya kütle dayanımları ve deformasyon modülleri Çizelge 1 ve 2'de verilmiştir.

Problemi tanımlayan model gövdesi Şekil 2 de verilmiştir. Gerilme durumu ve geometrideki simetriden dolayı problemin sadece ¼ ü modellenmiştir. Referans eksenleri ve orjin şekilde belirtilmiştir. Radyal simetrik grid yapısı Şekil 2 de gösterildiği gibi açıklıktan uzaklaştıkça aralık artacak şekilde tasarlanmıştır. Ayna yakınlarında inceleme alanında daha detaylı bilgiler alabilmek amacıyla daha küçük aralıklı grid ler kullanılmıştır. Sınırlar açıklık eksenine 10 yarıçap uzaklıkta seçilmiştir. Modelde 37440 zon ve 41817 grid noktası kullanılmıştır. Uzunlamasına mesafe ise açıklık yarıçapının 24 katı olacak şekilde belirlenmiştir.



Şekil 2. Model geometrisi ve sınır koşulları.

Model geometrisi ve sınır koşulları Şekil 2 de belirtilmiştir. Simetrik dış sınırlar yatay deformasyonu engelleyecek şekilde sabitlenmiştir.

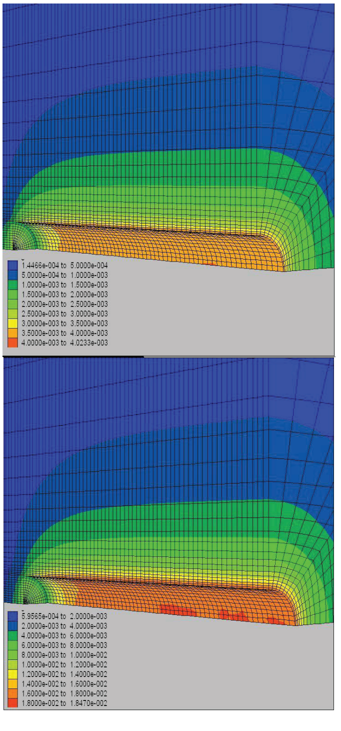
Analiz sonucunda elde edilen verlerden takip eden boyutsuz parametreler hesaplanmıştır. Ilk boyutsuz parametre açıklık duvarlarındaki radyal yerdeğişiminin (Ur) açıklık yarıçapına (R) bölümü ile elde edilen tünel birim deformasyonudur (Ur/R). Hoek (2001) tarafındanda belirtildiği gibi tünel birim deformasyonu muhtemel açıklık duraysızlıklarının belirlenmesinde önemli yere sahiptir.

İkinci boyutsuz parametre aynaya olan normalize mesafe (L/D) olarak isimlendirilir ve aynaya olan mesafenin açıklık çapına oranı ile elde edilmektedir.

Son boyutsuz parametre ise kaya kütle dayanımının (σ_{cm}) yerinde gerilmeye (P_v) oranı ile elde edilen dayanım faktörüdür (σ_{cm}/P_v).Bu oransa genelde kaya kütlesinin aşırı gerilmelerle yüklenmesi riskinin kestiriminde kullanılmaktadır.

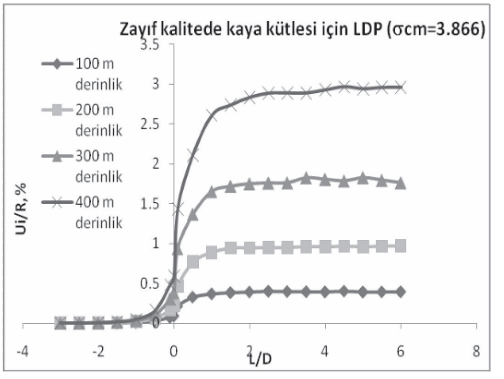
4.1 Sayısal Modelleme Sonuçları

Şekil 3'te 100 ve 300 m derinliklerde zayıf kaya kütlesinde oluşturulmuş 2 m çaplı dairesel açıklık etrafındaki radyal deformasyonları gösterilmektedir. Plastik zon uzanımı ve dairesel açıklık etrafındaki yerdeğişimleri dağılımları düzenli ve açıklık duvarına paralel şekilde gelişmiştir. Kaya kütle kalitesi düştükçe, derinlik ve gerilmeler artıkça plastik zon çapının ve yerdeğişim miktarlarının artığı gözlemlenmektedir.



Şekil 3. 100 m ve 300 m derinliklerde zayıf kaya kütlesinde açılan 2 m çaplı açıklık için yerdeğişimleri.

100, 200, 300 ve 400 m derinliklerde zayıf kaya kütlesinde oluşturulan 2 m çaplı açıklığa ait yerdeğişimi profilleri Şekil 4'te verilmiştir. Şekilden de görüldüğü gibi deformasyonlar aynanın 1-1,5 çap mesafesi önünden başlamakta ve aynanın 2,5 çap mesafesi gerisinde maksimum değerine ulaşmaktadır. Aynada ise maksimum deofrmasyonun yaklaşık %30'u meydana gelmektedir. Sonuçlara göre derinlik ve kaya kütle kalitesi artıkçadeformasyon profili veya plastik ozn kalınlığı göreceli olarak daha düşük mesafelerde duraylı hale gelmektedir.



Şekil 4. Farklı derinliklerde zayıf kalitede kaya kütlesi içinde oluşturulan dairesel açıklıklar için deformasyon profilleri.

5 ÇOKLU REGRESYON MODELLEMESİ

5.1 Model Oluşturulması

Ayna önü ve arkasındaki normalize radyal yerdeğişimlerinin tahmini ve tünel birim deformasyonu, dayanabilirlik faktörü ve aynaya olan normalize mesafe arasındaki ilişkilerin belirlenmesi için çoklu regresyon modellemesi kullanılmıştır. Modellerdeki bağımlı parametre tünel birim deformasyonu (Ur/R) olarak seçilirken bağımsız parametreler ise dayanabilirlik faktörü (σ_{cm}/P_v) ve aynaya olan normalize mesafe (L/D) olarak seçilmiştir.

Uzunlamasına deformasyon profilinin eğrisel yapısından dolayı modelleme aşamasında problemlerle karşılaşılmıştır. Bu problemleri aşabilmek için ayna önü ve arkasındaki deformasyonların tahmini için farklı modeller oluşturumu yoluna gidilmiştir. Farklı derinliklerde zayıf ve çok zayıf kayalarda açılacak olan dairesel açıklıkların aynaları önündeki normalize edilmiş radyal deformasyonların tahmini için 8 ayrı model oluşturulmuştur.

5.2 Model Geçerlilikleri

Oluşturulan modellere ait bilgiler Çizelge 3'te verilmiştir. Ayarlanmış çoklu belirleme sabiti değerleri (R^2) çizelgedende görülebileceği gibi %96.56 ile , %99.28 arasındadır. Bu yüksek değerlerdende anlaşılacağı üzere önerilen modellerin seçilen parametreler arasındaki ilişkiyi doğru bir şekilde kurduğunu belirtmektedir.

Çizelge 3. Regresyon modelleri ve ayarlanmış R^2 değerleri.

H, m	Pozisyon	Model	Ayarlanmış R^2
100	Ayna önü	$U_r/R=0.040x9.711^{(L/D)}x\sigma_{cm}/Pv^{(-1.802)}$	98.91
	Ayna arkası	$U_r/R=0.202x1.114^{(L/D)}x\sigma_{cm}/Pv^{(-1.695)}$	97.73
200	Ayna önü	$U_r/R=0.203x9.270^{(L/D)}x\sigma_{cm}/Pv^{(-1.369)}$	99.28
	Ayna arkası	$U_r/R=0.516x1.104^{(L/D)}x\sigma_{cm}/Pv^{(-1.417)}$	97.65
300	Ayna önü	$Ur/R=0.549x8.010^{(L/D)}x\sigma_{cm}/Pv^{(-1.144)}$	99.32
	Ayna arkası	$Ur/R=0.954x1.095^{(L/D)}x\sigma_{cm}/Pv^{(-1.273)}$	97.05
400	Ayna önü	$Ur/R=0.630x8.286^{(L/D)}x\sigma_{cm}/Pv^{(-1.110)}$	99.21
	Ayna arkası	$Ur/R=1.756x1.092^{(L/D)}x\sigma_{cm}/Pv^{(-1.148)}$	96.56

Modellerin genel geçerliliklerinin ise F-testi ile ıspatlanması gereklidir. ANOVA çizelgesi ve hesaplanan F değerleri çizelgede verilmiştir. Çizelge 4 ve 5'te görüldüğü gibi Fisher F testi çok düşük olasılık değeri (Prob(F)) vermektedir. Buda göstermektedirki önerilen modeller geçerlidir.

Çizelge 4. 100, 200 m derinlikler için ANOVA çizelgesi.

H, m	Pozisyon	Kaynak	DF	Kareler toplamı	Ortalama kare	F oranı	Prob(F)
100	Ayna önü	Regresyon	2	5706.49	2853.24	3590.07	0.00
		Hata	77	61.20	0.79		
		Toplam	79	5767.68			
	Ayna arkası	Regresyon	2	1791636.71	895818.35	3602.96	0.00
		Hata	165	41024.58	248.63		
		Toplam	167	1832661.29			
200	Ayna önü	Regresyon	2	28526.53	14263.27	5746.11	0.00
		Hata	81	201.06	2.48		
		Toplam	83	28727.60			
	Ayna arkası	Regresyon	2	8165298.63	4082649.31	3464.86	0.00
		Hata	165	194419.45	1178.30		
		Toplam	167	8359718.07			

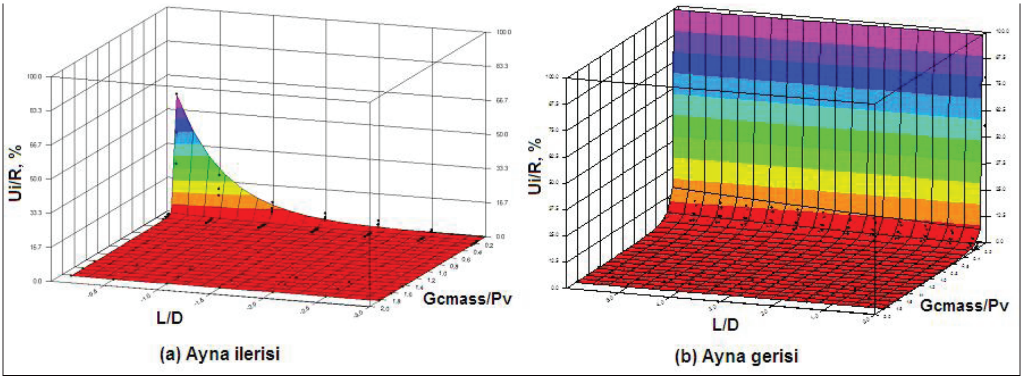
Çizelge 5. 300, 400 m derinlikler için ANOVA çizelgesi.

H, m	Pozisyon	Kaynak	DF	Kareler toplamı	Ortalama kare	F oranı	Prob(F)
300	Ayna önü	Regresyon	2	67836.45	33918.23	6047.35	0.00
		Hata	81	454.31	5.61		
		Toplam	83	68290.76			
	Ayna arkası	Regresyon	2	18368220.43	9184110.22	2743.73	0.00
		Hata	165	552306.61	3347.31		
		Toplam	167	18920527.04			
400	Ayna önü	Regresyon	2	117502.13	58751.07	5214.88	0.00
		Hata	81	912.55	11.27		
		Toplam	83	118414.68			
	Ayna arkası	Regresyon	2	32485153.03	16242575.52	2344.99	0.00
		Hata	165	1142871.40	6926.49		
		Toplam	167	33628024.44			

100 m derin için regresyon eşitliklerinden elde edilen ayna ilerisi ve gerisine ait cevap yüzeyleri Şekil 5'te verilmiştir. Şekilde gösterilen yüzeyler pratik bir tasarım aracı olarak projelerin ilk aşamalarında kullanılabilirler. L/D ve σ_{cm}/P_v bilindiği taktirde U_r/R yaklaşık tahmin edilebilir. Aynı zamanda L/D ve σ_{cm}/P_v nin U_r/R üzerindeki etkileride grafiklerden anlaşılabilmektedir.

6 SONUÇ

Bu çalışmada farklı kalitede kaya kütlelerinde farklı derinliklerde açılacak olan dairesel açıklık aynası etrafındaki deformasyonların tespiti amaçlanmıştır. Kaya kütlesi karekterizasyonu için GSI sistemi kullanılmıştır. İdealize edilmiş zemin koşulları için kaya kütle ve malzemesi için dayanım ve deformabilite parametreleri GSI sistemi esaslı eşitlikler kullanılarak belirlenmiştir. İki boyutlu sayısal analiz ve analitik yaklaşımların sınırlamalarını aşabilmek için üç boyutlu sayısal modelleme tekniği kullanılmıştır. Üç boyutlu doğrusal olmayan regresyon modellemesi tekniği ayna önü ve arkasındaki deformasyonların tahmininde kullanılmıştır. Önerilen regresyon eşitliklerinin geçerlilik ve performansları istatistiksel testlerle kontrol edilmiştir, bu regresyon eşitlikleri kullanılarak çok farklı kaya ve gerilme koşulları için cevap yüzeyleri oluşturulabilir.



Şekil 5. 100 m derinlik icin ayna ilerisi ve gerisine ait tünel birim deformasyonu cevap yüzeyleri.

Bu yüzey veya eşitlikler kullanılarak projelerin ilk aşamalarında faydalı olabilecek olan tünel birim deformasyonu farklı gerilme ve kaya koşulları içinde oluşturulabilecek olan açıklıklar için tahmin edilebilecektir.

Bu çalışmada bazı sınırlama ve kabuller mevcuttur. Kaya kütlesinin elastik mükemmel plastik malzeme özellikleri taşıdığı varsayılmaktadır. Tepe ve kalıcı dayanım parametrelerinin eşit olduğu ve genleşme açısının sıfır olduğu varsayılmıştır. Bu varsayım çok zayıf ve zayıf kaya kütleleri için geçerli olmakla beraber daha iyi kaya kütlelerinde yenilme karekteristikleri tekrar gözden geçirilmelidir. Örneğin kaya kütlesi belirli yönelimde hakim bir süreksizlik içeriyorsa, mekanik davranışın anizotropik olması beklenir ve bu gibi durumlarda Hoek Brown yenilme kriterinin kullanılmaması gerekir. Bu gibi durumlarda elasto plastik malzeme davranışı ve anisotropik modelleme kullanılmalı veya süreksizlik esaslı modelleme teknikleri kullanılmalıdır.

Kaya kütle özellikleri, derinlik ve gerilme koşulları inşaat ve maden mühendisliği amaçlı açıklıklarda sıklıkla değişen parametreler olduğundan bu projede önerilen eşitlik ve yüzeyler özellikle sınırlı verinin elde edilebildiği projenin erken safhalarında kullanılabilirler. Bu kullanım sırasında çalışmada kullanılan varsayım, sınırlamalara dikkat edilmelidir. Ayrıca önerilen eşitliklerin ilk tahmin için kullanılabilir olmakla birlikte varsayılan koşullar için geçerli olduğuna özellikle dikkat edilmesi gereklidir.

TEŞEKKÜR

Yazar çalışma boyunca desteklerinden dolayı Dr. Bahtiyar Ünver'e, Dr. Melih Geniş'e, Dr. Ahmet Özaslan'a ve FLAC3D yazılımının kullanımına imkan sağladığı için Hacettepe üniversitesi Maden Mühendisliği Bölümüne teşekkür eder.

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Üç Eksenli Sünme Test Aparatı Tasarımı

Design of the Triaxial Creep Test Apparatus

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ÖZET Bu çalışmada doğal gaz depolama amaçlı kullanılacak olan kaya tuzunun sünme deneyleri için yeni bir aparat tasarlanmış ve üretilmiştir. Tasarımı ve üretimi yapılan bu aparat kaya tuzunun üç eksenli basınç altında zamana bağlı gösterdiği deformasyonların belirlenmesi için kullanılmıştır. Aparat aynı anda iki adet örnek üzerinde hem düşey yük hem de yanal yük verecek şekilde tasarlanmıştır. Düşey yükleme kapasitesi 8 ton, yanal basınç kapasitesi ise 80 bar'dır. Yapılan ön deneyler aparatın üç eksenli sünme deneylerinde kullanılabileceğini göstermiştir. Sünme deney sonuçları gösterilmiştir

ABSTRACT This paper describes the apparatus designed and manufactured for creep test of rock salt which is going to be used for natural gas storage. The apparatus was used to investigate the long term deformation of rock salt at constant confining pressure. Apparat applied vertical and horizontal load to two specimen at the same time. The vertical load capacity of apparat was 8 tons and the horizontal pressure capasity was 80 Bars. Preliminary test indicated that the apparatus is capable of testing the triaxial creep experiments. The results of triaxial creep tests are reported.

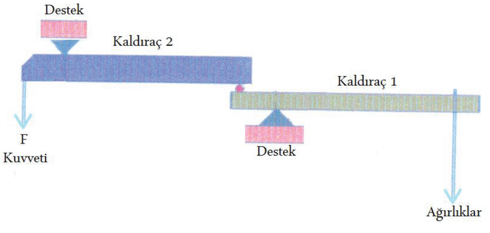
1 GİRİŞ

Sünme malzemelerin sabit yük veya gerilme altında zamana bağlı olarak gösterdikleri birim deformasyonlardır. Bu bağlamda; sünme, doğal gaz depolarının içinde açıldığı ortam kayacının davranışıdır. Doğal gaz enerji yönünden değerledirilirse depolamanın önemi anlaşılmaktadır. Depo tasarımı, depo kayacının mekanik davranışının incelenmesi ve açıklık gerilme hesaplamaları ile yapılır. Depo kayacı olarak alınan kaya tuzunun üç eksenli sünme davranışlarının incelenmesi temel etüd olmaktadır. Dolayısıyla uygun bir aparat tasarımı gerekmektedir. Tasarlanan sünme deney aparatlarında dikkat edilecek en önemli nokta uygulanan yükün veya gerilmenin deney süresi boyunca sabit tutulmasıdır. Bu bağlamda yaylı sistemler, ölü yüklemeli sistemler ve hidrolik sistemler kullanılarak çeşitli sünme deney aparatları tasarlanmıştır. Hulse ve Copley 1966 yılında yüksek sıcaklık değerlerinde ve basınç altında gerilme-birim deformasyon ve sünme verilerini kayıt edebilen üç aparat geliştirmiştir. Birçok tek eksenli sünme test donanımları literatürde tanımlanmıştır (Kreglo ve Smothers 1967, Clements ve Vyse 1966). Bunların bazılarının tasarımları karışık ve büyük idi. Smith ve Moore (1971) basit ve pahalı olmayan bir sünme aparatı geliştirmişlerdir. Deformasyon ölçer (LVDT) kullanılarak birim deformasyonlar ölçmüşlerdir. Carroll ve Weiderhorn (1988) yaptıkları sünme aparatında vidalı düzenek yardımı ile yükleme blokları ve ayaklar kullanmışlardır. Sünme deneyi süresince uygulanan yükün sabit tutulması önemli bir konudur.

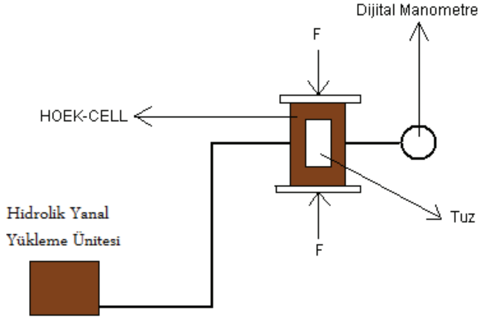
Bu çalışmada uygulanan düşey yükün sabit tutulması için yerçekimin kuvvetinden faydalanılmıştır. Yerçekimi kuvveti ile istenilen yük seviyelerinin elde edilebilmesi için de kaldıraç düzeneği kullanılmıştır. Yanal yüklemede ise hidrolik basıncından faydalanılmış olup, bu yükün sabit tutulması için ise hidrolik yağ basıncını sabit tutabilen bir hidrolik ünite kullanılmıştır.

2 TEST APARATININ TASARIMI

Deney aparatının amacı uygulanan düşey ve yanal yükün sabit tutulmasıdır. Aparat tasarımı iki aşamadan oluşmaktadır, bunlar: 1- Düşey yükleme çerçevesi tasarımı, 2- Yanal yükleme ünitesi tasarımıdır. Sabit düşey yük sistemi ve yanal yükleme sistemi Şekil 1'de şematik olarak gösterilmiştir.



(a) Düşey yükleme



(b) Yanal yükleme

Şekil 1. Test aparatının şematik görünüşü.

2.1 Düşey Yükleme Çerçevesi

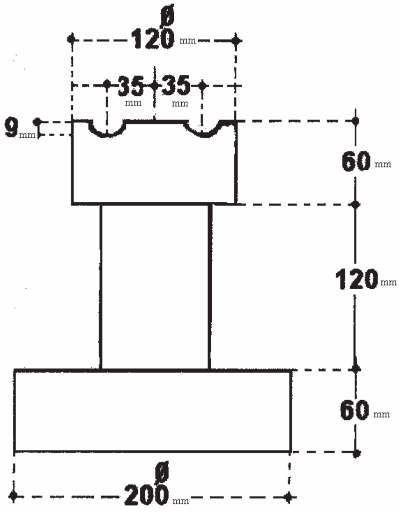
Düşey yük kaldıraçların kuvvet kollarına asılan ağırlıklar sayesinde yük kolundan sisteme sağlanmıştır. İstenen yükün elde edilebilmesi için kuvvet kolunun ve ağırlıkların optimizasyonu yapılmış olup iki kaldıraç sisteminin birbirine seri bağlanması yoluna gidilmiştir (Şek. 1a). Böylece uzun kuvvet koluna ve taşınması zor olan ağırlıklara ihtiyaç kalmamıştır. Aparat asılan ağırlıklar ile yerçekimi kuvveti sayesinde deney süresi boyunca sabit yük uygulayabilecektir. Her bir kaldıraç düzeneğinden 1'e 4 oranında bir kazanç sağlanarak toplamda yaklaşık 1'e 16 oranında bir kazanç sağlanmıştır. Kaldıraç kollarının taşıdıkları yüklere göre boyut hesaplamaları yapılmış olup kullanılan malzeme SD60 tür çelikten imal edilmiştir. Bu çeliğin kullanılmasının sebebi akma gerilmesinin 400GPa değerlerinde olması ve düzenekte kollarda zamanla meydana gelebilecek bükülmeleri ortadan kaldırmasıdır. Kullanılan mesnetler kaldıraç 1'de üçgen şekilli, kaldıraç 2'de ise daire şekilli seçilmiştir. Sisteme ağırlıklardan elde edilen yük düşey olarak uygulanacağı için kaldıraç kollarının da üstünde bulunacağı üst destek gövdesi gösterilmiştir. Üst destek gövdesi 10 ton yük taşıyacak şekilde yine SD60 çelikten imal edilmiştir. Bu gövdenin üst sağ ve sol kısmına toplamda 4 adet kaldıraç düzeneği ikili gruplar halinde monte edilmiştir. Her bir kaldıraç grubundan yaklaşık 5 ton toplamda ise 10 tonluk bir yük edilmiştir.

Elde edilen yükün örneklere aktarılması için tasarlanan yükleme pistonu Şekil 2'de gösterilmiştir.

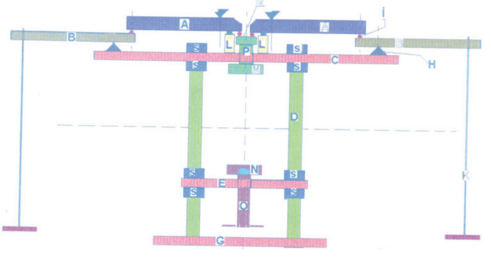
2.2 Yanal Yükleme Ünitesi

Şekil 1b'de gösterildiği gibi yanal yükleme ünitesi hidrolik ünite, hoek-hücresi ve dijital manometrelerden oluşmaktadır. Ünite istenen basıncı hidrolik pompa ile sağlayarak hoek-hücresi aracılığı ile örnek üzerine çevresel basınç olarak iletmektedir. Üniteye bağlı dijital manometreler ile sünme deneyi boyunca hoek hücresindeki basınç değerleri takip edilmiş ve kayıt altına alınmıştır.

Hidrolik ünite aynı anda 3 adet hoek hücre sini yükleyecek şekilde tasarlanmıştır.



Şekil 2. Düşey yükleme pistonu.

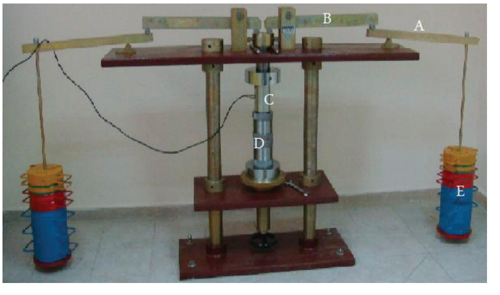


Şekil 3. Düşey yükleme çerçevesi.

Hidrolik basıncının deney süresi boyunca sabit tutulabilmesi için akümülatör kullanılmıştır. 0 – 80 Bar arası basınç uygulayabilen bu akümülatörlerden her bir hücre için birer adet kullanılmıştır. Bu durumda aynı sabit düşey yük altında farklı yanal basınçlarda deneyler yapılmıştır. Her bir akümülatörün istenilen yanal basınç değerlerine ayarlanabilmeleri için 3 adet 0,1 Bar hassasiyetinde valf kullanılmıştır. Hoek-hücresi içerisindeki basınç seviyesinin daha hassas takip edilebilmesi için Stauff SPG-DIGI marka dijital basınçölçer Şekil 1b'de gösterildiği gibi hoek-hücresi çıkışına bağlanmıştır. Maksimum basınç okuma kapasitesi 100bar (10 MPa)'dır. Kullanılmadan önce kalibrasyon ihtiyacı duymamaktadır. Fabrika kalibrasyonu belirtilen kullanım sıcaklıklarının üstünde veya altında bir ortamda çalıştırılmadığı sürece her yerde geçerlidir.

3 APARATIN KALİBRASYONU

Düşey yükleme çerçevesinin kalibrasyonunda aparata uygulanan yükün belirlenmesi için Şekil 4'te gösterildiği gibi bir yük hücresi bağlanmıştır. Kaldıraç kollarının kuvvet kollarına asılan ağırlıklar ile aparattan elde edilen toplam yük miktarı Çizelge 1'de gösterilmiştir. Ağırlık yük kalibrasyonu yapılırken aparat 2 kez tamamen boşaltılıp tekrar yüklenmiştir. 1. ve 2. Yüklemelerde kaldıraç kollarına 40 kg'lık ağırlıklar asılarak yük-hücresindeki toplam yük miktarları kayıt altına alınmıştır.



A: Kaldıraç 1 B: Kaldıraç 2 C: Yük Hücresi
D: Hoek-Hücreleri E: Ağırlıklar

Şekil 4. Düşey Yükleme Aparatı.

Çizelge 1. Ağırlık ve yük kalibrasyonu.

Ağırlıklar (kg)	1.Yükleme Load Cell (kg)	2.Yükleme Load Cell (kg)
0	140	140
40	820	780
80	1500	1350
120	2180	2080
160	2860	2770
200	3540	3460

Sünme deneylerinde yükün uzun zamanda sabit tutulması deneyler açısından önemli bir yere sahiptir. Aparat maksimum yük seviyesine yüklenerek 30 gün boyunca yük-hücresinden okumalar alınmıştır. Bu sayede aparatta uzun zamanda, elde edilen yükte bir kayıp olup olmadığı araştırılmış, sonuçlar Çizelge 2'de gösterilmiştir.

Sünme deneyinde düşey deformasyonlar 0,001 hassasiyette ve 5cm kurs boyuna sahip komparatör saatleri ile ölçülmüştür.

Hoek-hücresi üzerine sabitlenen ve küresel başlıkların hareketi aracılığı ile örnek de meydana gelen boyca kısalmalar 3 adet komparatör saati ile takip edilmiştir. Saatler hoek-hücresi çevresine 120 derecelik açılar ile yerleştirilmiştir. Son okuma olarak ise aynı zamanda alınan bu üç okumanın aritmetik ortalaması kullanılmıştır.

Yanal ünitenin kalibrasyon çalışmalarında ünitede kullanılan ayar valfleri ve emniyet valflerinin çalışma aralıklarında bulunan hidrolik basınç değerlerinde ön deneyler yapılmıştır. Bu deneylerde dijital manometreler yardımı ile ayarlanan basınç değerleri belirli zaman aralıklarında takip edilerek hidrolik basınç ta bir kaçak olmadığı belirlenmiştir. Yanal birim deformasyonların okunması için yeni bir yöntem geliştirilmiştir. Bu yöntem de hoek hücresi içerisine sıkıştırılan ve hapsedilen sabit hidrolik basıncındaki değişimler kullanılmıştır. Hoek-hücresi içerisindeki numune zamanla sabit düşey ve yanal yük altında deformasyona uğrayacaktır. Numunede meydana gelen bu deformasyonlar hacimsel genişlemeye sebep olacaktır. Hoek-hücresi içindeki sabit hidrolik basıncı bu sayede artarak dijital manometreden okunan hidrolik basınç değerini değiştirecektir. Bu değişimden yola çıkarak hidrolik hacmi hesaplanmıştır. Bu hacim numunede ki hacimsel değişime eşit alınarak dolaylı yoldan numuneye ait yanal birim deformasyon bulunmuştur.

Çizelge 2. Uzun süreli yük okumaları.

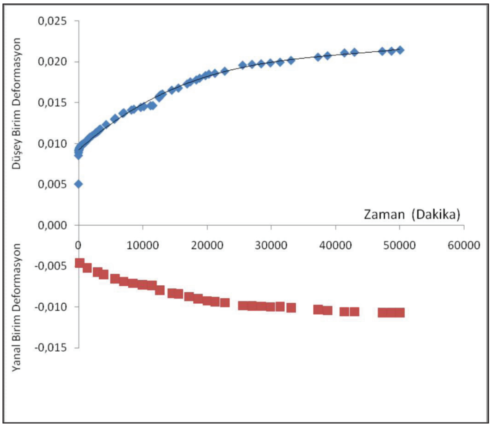
Tarih ve Saat	Yük-hücresi Okuma	Tarih ve Saat	Yük-hücresi Okuma	Tarih ve Saat	Yük-hücresi Okuma	Tarih ve Saat	Yük-hücresi Okuma
13.06.2008		15.06.2008		17.06.2008		19.06.2008	
11:45	3310 kg	12:35	3300 kg	14:25	3320 kg	16:10	3310 kg
11:50	3290 kg	12:41	3290 kg	14:27	3310 kg	16:14	3300 kg
11:53	3280 kg	12:44	3290 kg	14:29	3300 kg	16:21	3290 kg
12:00	3280 kg	13:00	3280 kg	14:32	3290 kg	16:27	3280 kg
12:15	3280 kg	13:20	3280 kg	14:35	3280 kg	16:45	3280 kg
12:30	3280 kg	13:40	3280 kg	15:25	3280 kg	17:10	3280 kg
Tarih ve Saat	Yük-hücresi Okuma	Tarih ve Saat	Yük-hücresi Okuma	Tarih ve Saat	Yük-hücresi Okuma	Tarih ve Saat	Yük-hücresi Okuma
23.06.2008		27.06.2008		02.07.2008		10.07.2008	
15:41	3310 kg	16:28	3290 kg	10:34	3260 kg	15:21	3290 kg
15:45	3300 kg	16:30	3280 kg	10:38	3260 kg	15:26	3280 kg
15:47	3290 kg	16:32	3270 kg	10:42	3250 kg	15:29	3270 kg
16:05	3280 kg	16:38	3260 kg	10:45	3240 kg	15:40	3260 kg
16:20	3280 kg	16:52	3260 kg	11:34	3240 kg	16:00	3260 kg
16:40	3280 kg	17:15	3260 kg	18:55	3240 kg	16:25	3260 kg

4 TEST METODU VE SONUÇLAR

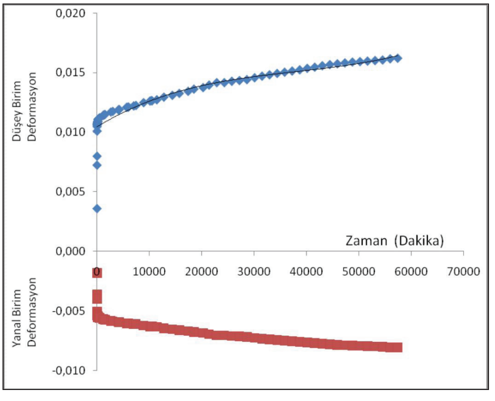
Sünme deneyleri zaman aldığından tanımlanan aparat ile sınırlı sayıda testler yapılmıştır. Deneyde kullanılan örnekler BX (42mm) çapındadır. Düşey deformasyonlar komparatör saatleri ile okunmuştur. Düşey yük kaldıraçların kuvvet kollarına asılan ağırlıklar ile sağlanmıştır. Bu şamada yanal yükleme ünitesi ile dijital manometrede ki basınç değeri takip edilerek hidrostatik yükleme koşulları sağlanmıştır.

Yapılan deneylerin sonuçları Şekil 6a ve Şekil 6b'de gösterilmiştir. Üçüncü sünme aşaması başlamadan deneyler bitirilmiştir

Tasarlanan bu aparat ile üç eksenli sünme davranışının temel prensiplerinin belirlenmesi sağlanmıştır.



Şekil 6a. Sünme deneyi sonuçları, σ_1 - σ_3= 20 Mpa.



Şekil 6b. Sünme deneyi sonuçları, σ_1 - σ_3= 15 Mpa.

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Technology of Benches Cutting at Final Slopes of High Waste Dumps at Cerovo 1

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ABSTRACT The technology of benches′planes formation at slopes of high waste dumps is carried out with bulldozers from the top of the slope to the bottom of the waste dump "Bugarski Potok", at Cerovo 1. Depending on the position of bulldozer′s plough in regard to the bench plane during cutting of benches the bulldozer is used an angle-dozer and tilt-dozer. The accepted height of bench slope is 20 m and the width of the bench plane is 4 m like the length of bulldozer′s plough.

1 INTRODUCTION

The Cooper mine Cerovo is located northwest of Bor, aproximately 13 km by the air line. The mine is situated at the ledge of Kraku Buguresku hill. The ore mine is composed of ore body Cementacija 1, orebody Cementacija 2 and Cerovo Primarno. The south part of ore body (Cerovo 1) has been exploated using open pit since 1993 till 2001. The increase of existing open pit Cerovo 1 and opening of a new one - Cerovo 2, new exploration reserves made possible (Figure 1.).

While Copper mine Cerovo 1 is being exploated, the hummus from ore body Cerovo 2 (distant 600 m) as been removed. Existing objects and infrastructure make continues production posible. At the sequent of existing tailing dump Kraku Bugaresku, tailings from existing enlargment segment of open pit Cerovo 1 and newly opened open pit Cerovo 2 is being dumped. The tailing dump is forming in three benches at levels K+420÷500 m; K+520 m and K+540 m, as shown at the Figure2. The height of the first tailing dump is 80 m, second and third benches have height of 20 m. Recultivation starts on the first bench slope with angle of 35^{o} with cutting terraces plane at each destination of 20 m (Figure 3.).

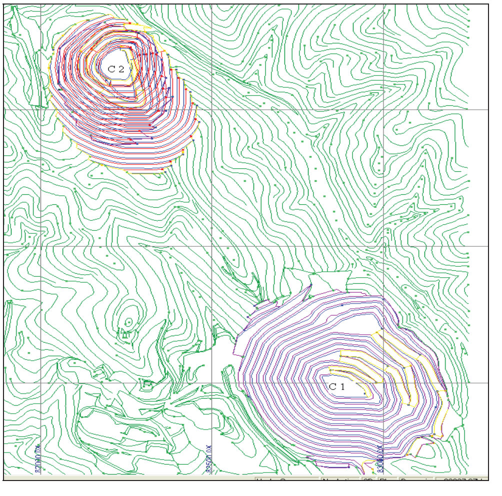

Figure 1. Open pits Cerovo 1 and Cerovo 2.

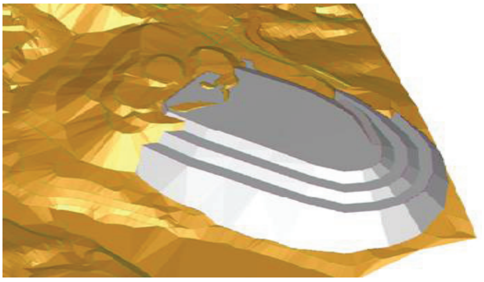

Figure2. Tailing dump at Cerovo.

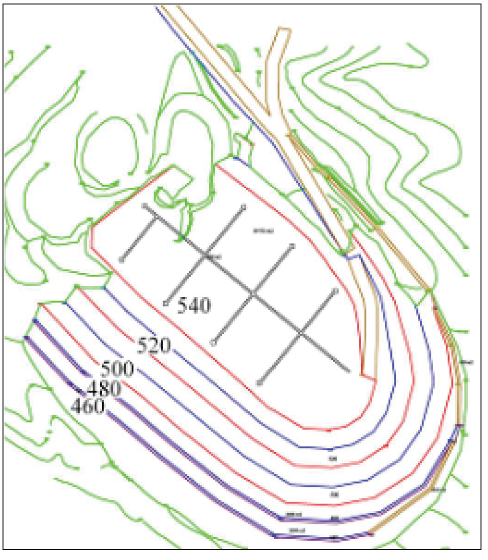

Figure 3. Cutting terraces (K+480; K+460) at final slope of the first bench Cerovo's tailing dump.

2 RECULTIVATION OF TAILING DUMP

Tailing dump quality at Cerovo belongs the class of the industrial soil. The proportion of nutritive substances of tailings is insufficient for normal plant development, considering that fact and in order to foresting degradation soil; there should be used the best type of recultivation which is optimal type of recultivation – eurecultivation; which considers using hummus during biological recultivation. Based on field expiriance, eurecultivation will perform by following schedule:

1) agro technical recultivation,
2) technical recultivation, and
3) biological recultivation.

Agro technical recultivation phase, represents the phase of preparing works, that means constructions of the entrance roads, surfaces leveling at bench and final plane and cutting landing plane which width is 4 m at the slope of the first tailing dump bench by levels K+480 and K+460. Landing plane grade to the inner slope terraces should be 3^0, as shown at the Figure 4.

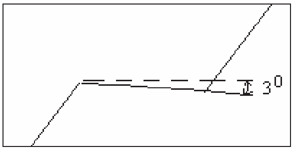

Figure 4. Final surface terraces plane at the tailing dump slope.

Technical recultivation phase includes hummus digging by bulldozer, loading and transportation hummus to tailing dump final and bench planes.

Biological recultivation phase includes a complex of bio technical and plant reclamation measures for Cerovo's ecosystem renewal.

3 CUTTING TERRACES AT FINAL SLOPE OF THE FIRST BENCH CEROVO'S TAILING DUMP

Cutting technology for terraces on the first bench slope of tailing dump will perform using bulldozer from top till bottom of tailing dump. Bulldozer can be used as angledozer or tiltdozer depending on the position of bulldozer plough in relation to the bench plane.

Cutting off terraces on slope of tailing dump begins when plough of angle dozer is putting down impacting the soil under angle in relation to the horizontal plane and stripping width of sidelong plough cut.

Angle dozer is working in filling material. In front of the bulldozer plough separated material is gathering forming pulling prism which has less width than bulldozer plough and considering that plough is under angle, material is being moved over the side and storage down the tailing dump slope. Terraces' cutting off is performing by segments, as shown at the Figure 5.

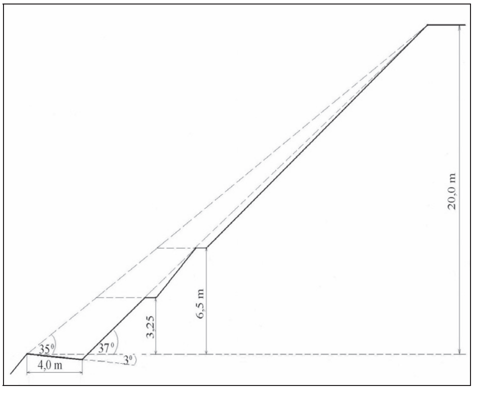

Figure 5. Cutting off terraces plane with angledozer by segments.

Cutting the first segment which has width of 4 m, starts at height of 6,5 m from the bottom of inner slope terraces plane in single pass. Volume of material which has been separated by angle dozer in one segment depends on length of cutting terraces plane.

Lenght of terrace at level K+480 is 900 m and for terrace at level K+460 is 663 m. Cutting off the second segment with perpose to form terraces plane starts at height of 3,25 m from calculate one. The third segment is cutting along level K+480, which is the first terraces plane.

Plough of bulldozer need to be angled in vertical plane, Figure 6, for getting terraces plane under angle of 5% towards inner slope. Tiltdozer capacity is half capacity of angle dozer.

The same technique is used for terraces plane at level K+460m. Slope stability of tailing dump is increasing by creating terraces, which causes leaking of surface waters slowing down, which has impact on surface erosion to regress, micro exposition is changing, which causes creation of better microclimate condition, especially moister and temperature that provides better success of biological recultivation phase of eurecultivation.

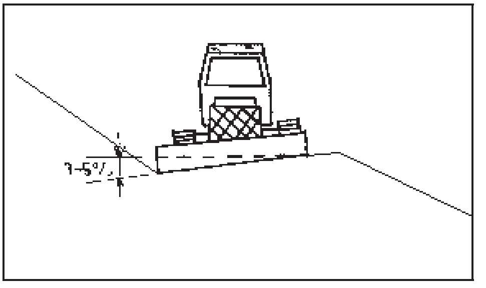

Figure 6. Tilt dozer is working on forming the final terraces plane.

Seedling's holes are digging mechanicly at flat surfaces (final planes) and they have rectangular shape (Figure 7.). Seedling's holes at slopes are digging manualy and they are roundy shape (Figure 8.).

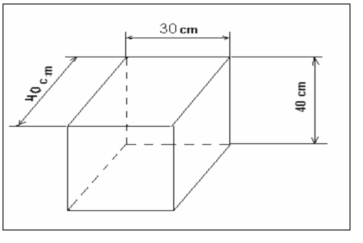

Figure 7. Rectangular shape of seedling' sholes at tilling dump final planes.

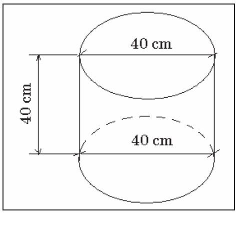

Figure 8. Round shape of seedling'sholes at tilling dump slopes.

Figure 9, schematic represents mechanical digging of seedling's holes at tilling dump final planes. Figure 10 represents mechanical digging using backhoe loader, when fixed at one position it can dig three holes at each position.

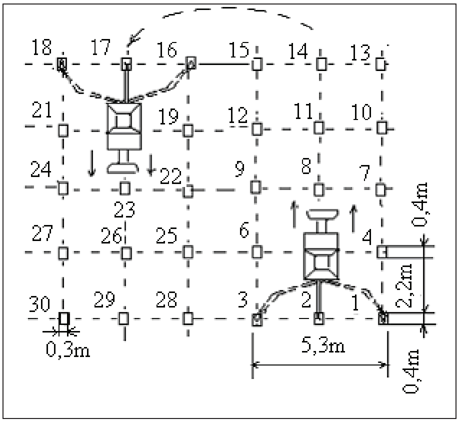

Figure 9. Schematic backhoe loader moving setup for seedling's holes digging.

Figure 10. Mechanical digging of seedling's holes at tilling dump.

Foresting at the teraces planes is performed in two rows acording to representing schematic at Figures 11 and 12.

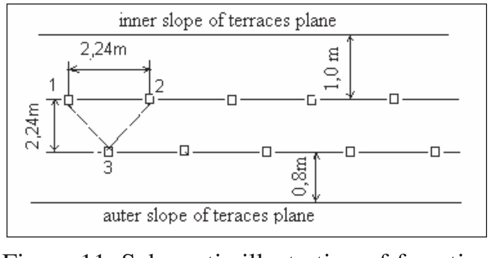

Figure 11. Schematic illustration of foresting teraces planes.

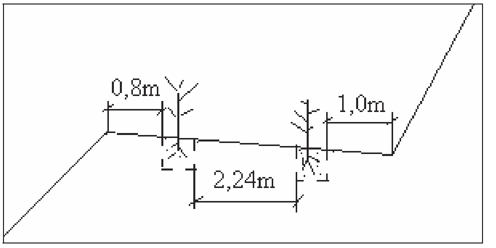

Figure 12. Schematic illustration in profile of foresting teraces planes

Seeding in the holes at the tilling dump slope is performing accordingly triangle schematic with seeding distance of 2.24 m (Figure 13), appropriate number of 2000 seedlings per hectare.

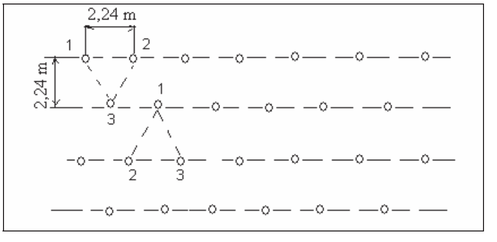

Figure 13. Schematic illustration of seeding at tilling dump slopes.

Before seeding 0.5 kg lime-filler has to be put into the holes to cover the round surface of the bottom. Organic materials has to be put into holes also. Along with hummus 5 l of peat should be added. Organic materials improves structural, nutritional and water-air characteristics of the substrate. For organic materials degradation is long period process fertilizers should be put also (200 gr/seedling NPK 15:15:15 . Seedlings should be earth up and fertilized in the first year of growing (KAN fertilizer 100 gr/seedling).

Taking into consideration local soil and climate conditions Acacia are good choice for recultivating tailing dump in Cerovo. The choice brings out the best results as far as adoption, coverage and maintenance is concerned.

4 CONCLUSION

Cerovo's tailing dump cutting off terraces plane increase slope stability of the tailing dump, causes that leaking of surface waters slowing down, which has impact on surface erosion to regress, better microclimate condition are forming, especially moister and temperature that provides better success of biological recultivation phase of eurecultivation.

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Cluster Analysis of Copper Ore Types by Neural Networks in Aid of Mine Operations Planning

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ABSTRACT Cluster analysis results for copper ore types can be used in the mine operations planning activities. The copper and sulfur grade are classified in fixed classes by the same analysis too. The information – raw exploration data, used for feeding neural networks is collected from the deposit exploration stage.

1 INTRODUCTION

The minerals industry of Bulgaria produces ferrous and nonferrous metals, and industrial minerals such as barite, clay, dimension stone, gypsum, limestone (lime), rock salt, and silica (Steblez, 2001). On a world scale, however, Bulgarian minerals industry is small and only of regional importance. The Bulgarian mines basic production is copper, iron, lead and zinc, and manganese. Additionally, byproduct gold, molybdenum, silver, and other metals are produced. Bulgaria's major copper deposits are in the Srednogorie-Panagjurishte region at the Asarel-Medet and the Elatsite surface mines and the Chelopech underground mine. The Assarel-Medet ore field has one of the biggest copper and copper-pyrite reserves in Bulgaria.

2 DESCRIPTION OF FIELD

In his work involved palaeozoic metamorfiti, granite, subvolcanic rocks as andesite, dacite, granodiorite, effusive rocks include lava cover and and andezite tuff. All rocks in the Panagyurishte ore region were undergoing of pre-ore, sin-ore and post-ore hydrothermal alteration. In the Assarel-Medet ore field are concentrate the largest reserves of Cu and Cu-Mo ores in Bulgaria. Assarel is copper – porphyry deposit. Three types of copper ore were formed – primary sulfide which consist chalcopyrite, pyrite and bornite; secondary sulfide which consist chalcosine and small quantity of coveline and oxide ores – malachite and azurite. The most significant ore minerals are chalcopyrite, pyrite, bornite, native gold, electrum and molybdenum. The main non ore mineral is the quartz. The chalchocite and covelyne from the secondary sulfide ores have industry interest. The oxide ores have high sulfur content. The mineralization age is upper Cretaceous. The primary zone is 1200 m, 120 m of the secondary and 12 m of the oxide part. The average of Cu grade is about 0.4 %.

Deposit exploration is realized in three stages – detail, pre-exploitation and exploitation. All raw data from the third exploration stage for three production levels are used in the investigation. There are individual data value, such as coordinates, copper grade and sulfur grade for the every sample.

3 METHOD OF INVESTIGATION

The term cluster analysis (used first by Tryon, 1939) comprises a different algorithms and methods for grouping objects of similar kind into respective categories. A general question for researchers in many areas of inquiry is how to organize observed data into meaningful structures, that is, to develop taxonomies. In other words cluster analysis is an exploratory data analysis tool which aims at different objects sorting into groups in a way that the degree of association between two objects is maximal if they belong to the same group and minimal otherwise. Given the above, cluster analysis can be used to discover structures in data without providing an explanation/ interpretation. In other words, cluster analysis simply discovers structures in data without explaining why they

Clustering techniques have been applied to a wide variety of research problems. Hartigan (1975) provides an excellent summary of the many published studies reporting the results of cluster analyses.

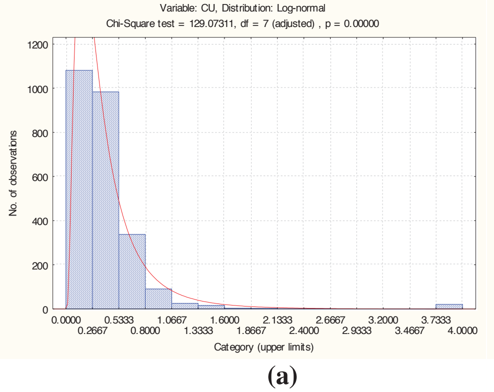
The geologists and mining engineers often come into need of classifying geological objects and processes in their practice. This classification is a subject to the following rules:

- in one classification applies the same basis;
- the volume of classified class is equal to the sum of the volumes of subclasses;
- classes and subclasses not intersect;
- subclass subdividding is done continuously.

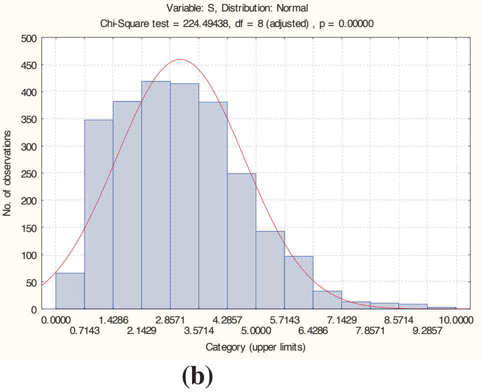
Most often the geologists solve one of the following two classification problems: natural division of the geological objects in clear marked groups or typifying in which objects are divided into small number of similar groups. The typifying has decision always.

The geological objects classification can be done by numeric, qualitative or classification signs, using formal mathematical methods. In the expert method, alternative to the formalized approach, grouping is accomplishing by specialists on the basis of heuristic knowledge.

The results from mine-geometrical analysis of the basic parameter - grade of copper - show that the empirical distribution is quite different from the normal and has positive asymmetry, i.e. it is lognormal (Fig.1a). The other parameter – sulfur grade is distributed nearly normally (Fig.1b). Investigating the character of grade changeability by the random functions theory gives reasons to affirm the presence of anisotropy in the variability of the basic parameter (Topalov and Hristov, 2007).



(a)



(b)

Figure 1. Copper and sulfur data distributions: a) lognormal for copper and b) normal for sulfur.

Maximum efficiency achievement of mineral processing, by mining works operational planning, needs to be ensured stabilized quality of mined ore. Knowledge of various copper and sulfur grade classes location will make easier the planning process and will reduced the risk.

Initially, for classifying of ore quality parameters were applied formalized approaches – cluster analysis. Stat Soft Inc. STATISTICA 7.0 software was used. Two statistical algorithms - Tree Clustering and K-mean clustering were executed. Unfortunately, the results of their implementation were inadequate. The reason of results quality is probably due to the parameter variation character. Groups with clear boundaries between them not been observed.

That is why the alternative of formal approach - an expert was used for number of groups determining and the parameter boundaries of each group. The group number was predetermined by mineral processing regime criteria. Three types of groupings were carried out: according to a copper grade, according to a sulfur grade and according to copper and sulfur grade simultaneously, respectively, for the three investigated production levels. Copper grade and sulfur grade values were divided into five intervals. The sulfur and copper grade, in common, were examined in 9 groups - 3 for each of the parameters. This was done for the three investigated production levels. When the copper grade was classified in five groups, the lognormal distribution was considered. That is the reason of different interval length. The intervals of sulfur grade dividing have same length and they are centered about the average sulfur grade.

Nine groups representing the two parameters together are defined by criteria low, medium and high content. On the basis of these grouping were defined relevant logical functions. Determining of each sample belonging to the cluster was done by them.

4 CLUSTER PREDICTION WITH NEURAL NETWORKS

Clustering neural networks belong to the self-organizing networks. They need to be trained so that when data sample is submitted to the input nodes of such a network, it can determine to which cluster these data belongs. The number of input nodes is equal to input data dimension. Output nodes number is equal to the number of clusters. The neural network has two layers and consists of one input and one output layer only, so that every input node is connected to all output. Every connection between units has weight.

When the network is well trained, putting an input data sample to the network it will activate only one output node (winner) - the unit responsible for the cluster of this input data sample. For clustering network training it is necessary to prepare a sample set from input data and an output vector with the same length as the number of input data, whose elements are code numbers or labels of the clusters formed from the input data.

STATISTICA 7.0 software has a module "NEURAL NETWORKS", including clustering networks. Using this software seven clustering networks were trained in this study – one for copper grade, one for sulfur grade for each of the three investigated production level, and one for copper grade and sulfur simultaneously. The results from the expert grouping were used in the training process. The six networks have as an input unit – for copper grade or sulfur grade and five output units for the five possible clusters (Fig.2a). The seventh network has two input units - for copper and sulfur grade together and nine outputs - for the nine possible clusters (Fig.2b).

The training process was carried out with the following parameters:

- Training algorithms:
  - K- mean with factors K=10 and L=10
  - Learned Vector Quantization with 5 000 epochs, Learning rate between 0.01 and 0.001, $\varepsilon = 0.35$ and $\beta = 0.25$.
- Classification without threshold

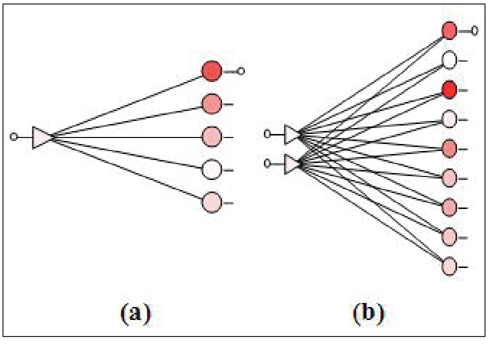


Figure 2 Architecture of clustering neural network – a) copper or sulfur grade separately; b) copper and sulfur grade simultaneously.

5 RESULTS

The analysis of the individual clusters location for each production level indicates some overlapping of different groups. Figure 3 shows the clustering results by expert grouping for the copper grade for all three production levels. Vertically can be seen the mutual cluster location in a one level, while horizontally – is displayed the cluster variation in depth (H – 870 → H – 855 → H – 840). On Figure 4 are shown the clusters of sulfur grade in the same way.

Figure 3 shows that in one level there is some spatial clustering of the data – copper grade in different intervals i.e. clusters are formed. On the other hand, these clusters overlap partly in the following order - each cluster with a higher grade value lies over the previous. These inferences are valid for all three extracting levels. Considering each cluster location in the depth (for the three investigated level) is prevailing the conclusion of preservation of cluster location relatively and more – the shape of each cluster too, which is quite important to know in the stage of operative mine activities planning.

Figure 4 is evident that sulfur is grouped spatial similarly by less overlapping of different clusters in a given level. This is true especially for the first two clusters. May be noticed a tendency to preserve the relative position and shape of clusters in depth.

The results of the seven clustering networks training are presented in Table 1. The last column gives the correlation between training clusters and network predicted clusters. The relationship between them is significant, according to the correlation coefficient value. The value of training error is acceptable. So, this demonstrates that clustering neural networks can be used for deposit quality parameters prediction.

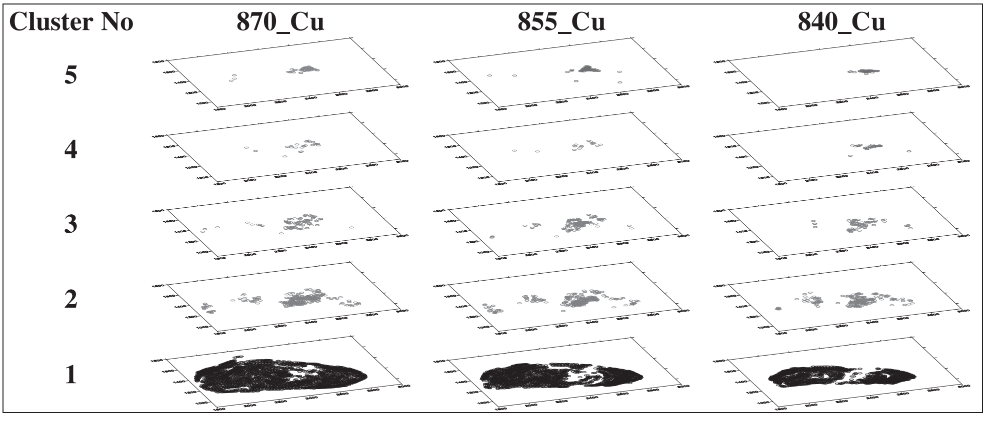


Figure 3. The clustering results by expert grouping for the copper grade for all three production levels.

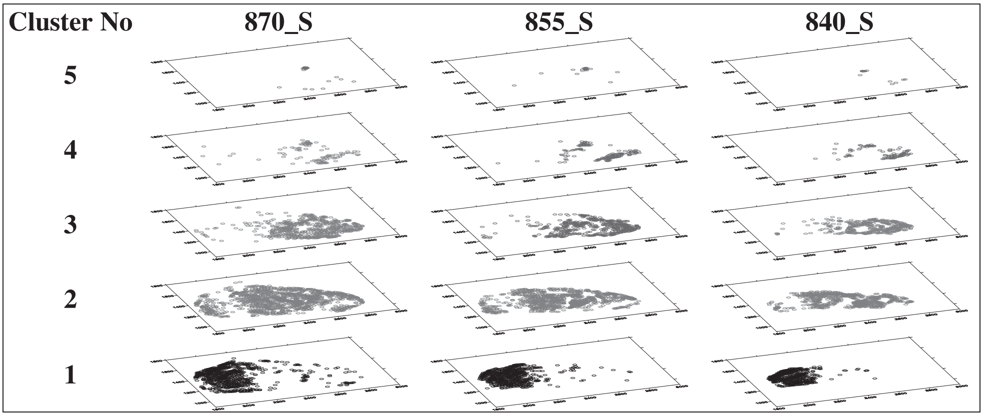


Figure 4. The clustering results by expert grouping for the sulfur grade for all three production levels.

Table 1. The results of the seven clustering neural networks training

Type	Profile	Train Perf.	Train Error	Training/Members	Correlation
CU, H 870	Cluster 1:1-5:1	0.889840	0.020478	KM,LV5000	0.88
S, H 870	Cluster 1:1-5:1	0.933826	0.040118	KM,LV5000	0.95
CU, S, H 870	Cluster 2:2-9:1	0.869988	0.055307	KM,LV5000	0.77
CU, H 855	Cluster 1:1-5:1	0.765137	0.028372	KM,LV5000	0.83
S, H 855	Cluster 1:1-5:1	0.920410	0.037801	KM,LV5000	0.95
CU, H 840	Cluster 1:1-5:1	0.851826	0.029989	KM,LV5000	0.90
S, H 840	Cluster 1:1-5:1	0.921433	0.036000	KM,LV5000	0.95

6 CONCLUSIONS

The results of formal cluster analysis algorithm direct implementation are not satisfied. That is why expert approach for quality parameter clustering was used. This cluster grouping process will support and make easier mine operation planning. The each cluster location and cluster shape within the framework of production level and in depth was analyzed. Seven neural networks were trained for each quality parameter for each production level and their cluster prediction reliability was estimated.

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Mathematical Model and Computer-Aided Optimization Strategy of the Methane Drainage Network

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ABSTRACT Restructuring of mining activities in Romania imposed by the transition to the market economy, concentration of extraction to fewer but higher output units in deeper seams, the high methane emission rates and coal rank, other current specific trends requires new methods to produce the optimum design and operation of a safe and economic methane drainage network. Under the circumstances, the paper describes a mathematical model together with a computer-aided optimization strategy to assist the analysis of the differences between predicted and actual methane drainage network performance. The application of the systematic auditing is highlighted by performing an analysis of the performance characteristics of a methane drainage network operating in Lupeni Colliery. The analysis carried out, with the support of the personnel in charge for degasification in Lupeni Colliery, allowed the identification of several feasible remedial maintenance and design measures able to improve the performance of the drainage system in operation.

1 INTRODUCTION

Coal mine methane, a by-product of mining operations, can be recovered to provide various types of benefits to a mining company. These benefits include, but are not limited to, reduced ventilation costs, downtime costs, production costs and the ability to use the recovered gas as an energy source, either at or near the mine site or by injecting it into a commercial gas pipeline system. There are many variables that play a part in the decision to implement and operate a coal mine methane drainage project.

In Valea Jiului coal basin the high firedamp content of the seams required, since 1969, resorting to local or central degasification techniques.

As it is well-known, the main objective of a methane drainage system is to remove the maximum amount of available gas from the relaxed surrounding strata so that the residual gas entering the mine workings can be effectively diluted by the design fresh air quantities. This forms an important part of an integrated ventilation planning strategy.

Meanwhile, the financial reasons related to the selling of coal can not be dissociated from those regarding the valorization of drained methane, valorization which in the present economical circumstances becomes more and more important requiring the integration in „global treatment of methane" at every colliery.

As workings in collieries of Valea Jiului coal basin have developed further from the surface openings there has been a correspondent increase in the number and length of installed methane drainage ranges. This has been accompanied by an attendant increase in the capacity of the surface extraction pumps and in the commissioning and operating costs of such systems. It has therefore become desirable to develop an analytical tool that is able to effectively simulate and analyze the optimal performance of installed or proposed drainage ranges.

The paper also describes how a ventilation engineer may employ the computer simulation model together with a decision support system to assist in analyzing the differences between predicted and actual methane drainage network performance. This comparative analysis will indicate the influence of such factors as air inleakage, the actual internal condition of individual pipelines, and identify the sections of the circuit that experience significant pressure drop and hence largest contribution to the operating power costs. The decision support system can assist in quantifying the effect of these factors and make suggestions as to the possible remedial maintenance or design measures that may be taken to improve the drainage range performance.

2 THE MATHEMATICAL MODEL

The methane drainage network is complex and consists of three elements with different characteristics:

- the drainage borehole;
- the drainage pipe;
- the extractor pump.

These characteristics can be expressed by corresponding formulae or presented graphically.

2.1 Characteristic of a Drainage Borehole

Research oriented towards the theoretical (Cias, 1976) and experimental (Szlazak, 1983) determination of the characteristics of drainage boreholes has proved that the mass flow of the methane - air mixture flowing from a drainage borehole, as well as from the whole group of drainage boreholes, is a parabolic function of the output depression.

Determinations achieved for the conditions specific to Valea Jiului coal basin have pointed out that the flow of the methane - air mixture (Q) and the methane flow (Q_m) through a drainage borehole or a group of drainage boreholes can be determined with sufficient accuracy by the following linear dependencies:

$$Q = A \cdot h + B \quad [m^3/s] \tag{1}$$

$$Q_m = C \cdot h + D \quad [m^3/s] \tag{2}$$

where: A, B, C, D - constant coefficients experimentally determined for each area; h-depression at the methane source level, Pa.

2.2 Characteristic of the Drainage Pipe

Using the formula proposed by Babut and Gontean (1993), pressure losses in horizontal pipes, for compressible gases, considering a continuous, isothermal and one-dimensional flow, in normal conditions of pressure and temperature and by replacing the concentration of the drained gas with the expression $c=Q_m/Q$, the following equation is obtained:

$$h^{(2)} = p_1^2 - p_2^2 = -R_1 \cdot Q^2 + R_2 \cdot Q \cdot Q_m \tag{3}$$

where:

$$R_1 = 212358 \cdot \frac{l \cdot \lambda \cdot z}{d^5} \cdot \frac{T}{T_0} \tag{4}$$

$$R_2 = 94605 \cdot \frac{l \cdot \lambda \cdot z}{d^5} \cdot \frac{T}{T_0} \tag{5}$$

where: p_1/p_2-upstream/downstream pressure, Pa; T-average absolute temperature of gas, $T=(T_1+2 \cdot T_2)/3$ (T_1/T_2, upstream/downstream absolute temperature of gas), K; λ-hydraulic friction coefficient; l-pipe length, m; d-inner pipe diameter, m; and z-factor of deviation from perfect gases law.

Relation (3) can be used in the case of inclined pipes as well, because the calculation error does not exceed 2 % for level differences of 800 m and 5 % for level differences of 1200 m, as initial data are usually determined with an accuracy of up to 10 %.

The level difference between the boundaries of the methane drainage network strongly influences the flow of the methane -

air mixture only if the operation of the extractor pumps is interrupted (breakdown situations).

The pressure losses due to fittings and valves are included in the model by employing an „equivalent length of pipe” for the pipe in which they are contained.

Supplementary air intakes from the atmosphere of the underground mine workings through due to the insufficient airtightness of the pipe network can be calculated by the following relation (Klassen, 1990):

$$Q_a = b \cdot l \cdot d \cdot (p_b - p)^{1,3} \quad [m^3/s] \qquad (6)$$

where: b-empirical coefficient defining the type of pipe joints; p-gas pressure in the drainage pipe, Pa; and p_b-barometric pressure in the mine working, Pa;

$$p_b = p_a \cdot e^{\frac{g \cdot h}{R_a \cdot T}} - 0,5 \cdot H_m \quad [Pa]$$

where: p_a-surface barometric pressure (pump station), Pa; h-depth of the mine working, m; g-gravity acceleration, g=9,81 m/s^2; R_a-air constant, R_a=287 J/kg·K; T-absolute air temperature in the mine working, K; and H_m-general depression of the mine, Pa.

As this relation is difficult to use in practice, the use of the following relation is proposed:

$$Q_a = k \cdot l \quad [m^3/s] \qquad (7)$$

where: k is an experimentally determined coefficient ($k=1,66 \cdot 10^{-5}$ (m^3/s)/m for the conditions in Valea Jiului).

In order to consider supplementary air intakes as well, in relation (3) the flow Q_j of branch j will be replaced by the total flow of branch j noted by Q_{tj} and given by the expression:

$$Q_{tj} = Q_j + Q_{aj} - 0,5 \cdot k \cdot l_j \quad [m^3/s] \qquad (8)$$

Relation (8) is not rigorously accurate, as supplementary air intakes represent an uniformly distributed flow, but for $Q_{tj}/Q_{aj} \geq 4$ (a generally satisfied condition), the error is negligible.

2.3 Characteristic of the Extractor Pump

The characteristic of the extractor pump can be described by the following equation:

$$p_v = A_0 \cdot Q_v + B_0 \quad [Pa] \qquad (9)$$

where: p_v-pressure of the methane - air mixture when flowing into the extractor pump, Pa; Q_v-flow of the methane - air mixture when flowing into the extractor pump, m^3/s; and A_0, B_0 - experimental coefficients of the extractor pump characteristic curve.

Such extractor pump characteristic curve is determined by measurements. Each characteristic curve should include the values of external pressure and temperature at which it is established.

3 THE ALGORITHM FOR THE COMPUTER SIMULATION OF METHANE DRAINAGE NETWORKS

As the drainage networks of Valea Jiului are of a branched type, with a single surface pump station, only this type will be subject to analysis (Babut, 1996).

3.1 Solving the Methane Drainage Network

In order to describe the algorithm for simulating methane drainage networks, the notion of flow route of the methane - air mixture will be used. This represents a system of methane drainage pipelines, serially connected, which constitute the branches of the network. The beginning of each route is the coupling point to the drainage boreholes and the end is the extractor pump.

For L coupling points of the drainage boreholes, there will be L independent flow routes for the methane - air mixture.

The use of relations (1), (2), (3), (8) and (9) leads to the following system of equations describing the methane drainage network:

$$p_v^2 = \sum_{j=1}^{R} a_{ij} \cdot [(p_{bi} - h_i)^2 - R_{1j} \cdot Q_{tj}^2 + \\ + R_{2j} \cdot Q_{tj} \cdot Q_{mj}] \quad \text{for } i = 1,...,L \tag{10}$$

where: a_{ij}-elements of the matrix defining the architecture of the network (a_{ij}=1, if branch j belongs to independent route i; a_{ij}=0, if branch j does not belong to independent route i).

Equation system (10) can also have the following form:

$$f_k(h_k) = p_v^2(h_k) - \sum_{j=1}^{R} a_{ij} \cdot [(p_{bk} - h_k)^2 - \\ - R_{1j} \cdot Q_{tj}^2(h_k) + R_{2j} \cdot Q_{tj}(h_k) \cdot Q_{mj}(h_k)] \quad (10') \\ \text{for } k = 1,...,L$$

In the equation system (10) the unknown values are represented by depressions in the coupling points of the drainage boreholes to the methane drainage network, values Q_{tj}, Q_{mj} and p_v (through Q_v) being functions of these unknown values.

Such an equation system can be solved by iterative methods, e.g. Newton's method.

3.2 Drainage Network's Adjustment

The regulation operations of a methane drainage network consist in determining the parameters of the regulators so that the pressures required for adequate methane drainage should be achieved in the coupling points of the drainage boreholes.

There are several criteria for establishing the required pressures, but the basic one consists in the achievement of a predetermined efficiency of drainage. This involves the drainage of a large amount of methane by keeping it at a high concentration. Knowing the characteristics of drainage boreholes, it is possible to determine the required methane - air mixture flows, as well as the necessary depressions to achieve these flows. The correlation of these depression values with the value of the pressure in mine workings at the coupling points of drainage boreholes allows the determination of the pressure in the drainage pipes at these points (Pawinski et al, 1992).

Based on the values thus determined, of the already - known network structure and of its geometrical parameters, it is possible to determine the parameters of the regulators and of the extractor pump. On the right side of equation system (10) there are given values; the pressure value at entering the extractor pump, for each route, is the unknown value. As all the routes are connected to the same extractor pump, these values should be equal and thus, regulators are required in the main branches. The right side of equation system (10) is completed by value $-\Delta p^2_{rk}$ i.e. the difference between quadratic pressures before and after entering the regulators.

Assuming that on the most difficult route, where $p^2_v(k) = p^2_v(k)_{min}$ we have $\Delta p^2_{rk} = 0$, by solving equation system (10), modified as mentioned before, it is possible to obtain pressure differences at the level of the regulator, as well as the required pressure at entering the extractor pump. If for the extractor pump type used $p_v < p_{vmin}$, this should be adjusted by using a regulator or a by-pass.

4 THE COMPUTER SOFTWARE

The use of the presented algorithm is difficult in practice, particularly for complex and large drainage networks. In order to facilitate its application, the SIMDEG software was elaborated (Babut, 1998). The development of SIMDEG software is based on the use of the most modern mathematical and computer tools, which allowed obtaining an original software product, which is, in the same time, of a high quality. This software was written in Borland C++ language, appealing to the object - oriented programming, while the methane drainage network's structure was described through the adjacent matrix, a specific field of graphs theory who finds his applicability in the study of binary trees.

The SIMDEG software offers the following computational facilities:

a. The methane drainage network's solving.

b. The methane drainage network's adjustment, when one of the following data sets is known:
   - the flows of the methane - air mixture (Q_j) in the joint points of the boreholes;
   - the methane flows (Q_{mj}) in the joint points of the boreholes;
   - the methane concentrations (c_j) in the joint points of the boreholes;
   - the depressions (h_j) in the joint point of the boreholes;
   - the flow of the methane - air mixture (q_v) at the extractor pump.

5 THE DEVELOPMENT OF THE OPTIMIZATION STRATEGY

The primary operational objective of the methane drainage network is to maintain the concentration, and hence the volume of drained gas, such that coal production targets are achieved, and that is potential to utilize the gas.

Reaching this goal imposes the need to accurately monitor and simulate the operation of the methane drainage network which will allow the identification and remediation of possible malfunctions within the system. Under these circumstances, the proposed computer model simulation will be applied together with an optimization strategy, to assess the drainage networks parameters, by comparing the predicted and measured data.

A detailed analysis of a methane drainage network can therefore be divided into four distinct stages:

1. the achievement of an initial simulation model, assuming new pipes and no air inleakage;
2. the development of a correlated simulation model employing measured data to predict air inleakage and the state of pipes and fittings;
3. the assessment of all the effects induced by remedial and maintenance measures applied;
4. the development of a simulation model to predict the effects induced by the reconfiguration of the existing methane drainage network.

Performing the above systematic analysis will enable the identification of:

- the magnitude and location of air inleakage;
- the extent of the deterioration of the pipes;
- the performance of the pipe fittings;
- the effect induced by the presence of water in pipes.

Based on the above - mentioned data, the ventilation engineer is thus able to establish a number of practical remedial maintenance measures that may improve the operational performance of the methane drainage network.

6 CASE STUDY - LUPENI COLLIERY

In view of illustrating and practical validation of the optimization strategy, the Lupeni Colliery methane drainage network was selected, while it can be considered as a representative one for the particular conditions of Valea Jiului coal basin.

The optimization strategy was applied by following the next steps:

I. The achievement of an initial simulation model, assuming new pipes and no air inleakage. From safety reasons, it was not possible to change the functional parameters of the methane drainage network, so that we couldn't introduce in the model the characteristics of methane sources and of vacuum pumps, to achieve the goal of a dynamic simulation of the analyzed network.

II. The development of a correlated simulation model employing measured data to predict air inleakage and the state of pipes and fittings. The drainage pipes were analyzed - in a first stage - from the point of view of air inleakage. The obtained data allowed to emphasize the following aspects:

   a. The assessment of air inleakage
      1. The air inleakage varies between the following limits: d=0,150 m, $10\text{-}35 \cdot 10^{-3}$ m^3/min. per 100 m of pipe; d=0,200 m, $11\text{-}30 \cdot 10^{-3}$ m^3/min. per 100 m of pipe; d=0,300 m, $14 \cdot 10^{-3}$ m^3/min. per 100

m of pipe. The presented values are correspondent - as magnitude - with those presented in literature. It must be noticed that differences relatively high of air inleakage for certain pipe sections of same diameter are registered. This is due to:

- the cumulative effect of errors generated by measuring devices;
- the persistence of several deficiencies in methane drainage network remedial maintenance activity;
- the gaps in procurement of necessary materials, leading - directly - to the use of some improvisations.

In some punctual situations, the high air inleakage are due to specific causes, such as:

- the existence of leakage from the coal massif, with a high potential of corrosion;
- the deterioration of pipes jointing systems induced by the displacements exerted on them;
- the malfunction of water traps.

2. The air inleakage quantity represents about 0,81-15,56 % from the entire gas quantity flowing in the branches of the methane drainage network. It can be - consequently - stated that for the specific conditions in which drainage networks are operating in Valea Jiului collieries the air inleakage have „normal" values situated at 7-11 % from the whole gas quantity transported in pipes. This corresponds to an air inleakage coefficient, ranging in the field of $15\text{-}25 \cdot 10^{-3}$ m^3/min. per 100 m of pipe. Any variations from these values are only accidental cases, which are not raising difficult problems related to the methane drainage systems sealing/jointing.

b. The assessment of pipes and fittings state. Introducing the air inleakage values in the initial model and carrying out a new simulation process, facilitated the assessment of pipes and fittings state, expressed through the ratio between the „effective" friction factor „λ_{ef}" (for operational pipes) and friction factor „λ" (for new installed pipes). The obtained data revealed the following conclusions:

1. The values of „effective" friction factor corresponding to those met in literature.
2. The high values of „effective" friction factor are registered in branches located in neighborhood of methane sources, the causes of this phenomena being:
   - the water can be ingested into the pipes from the surrounding strata, condensation and as the result of wet drilling operations, together with the presence of sediments and corrosion particles, will decrease the active cross sectional area of the pipes and hence rapidly increase the resistance to flow;
   - the high values of pressure losses were registered at regulator's levels, without being included in the initial model.

   These high values of friction factor are also related to some specific causes, such as:
   - the deterioration of fittings as a consequence of uncontrolled displacement of pipes;
   - the bad state of pipes, including an increased roughness level, presence of diaphragms and regulating devices, technically non-fitted to the achievement of their purposes.

III. The prediction of the effects induced by remedial and maintenance measures applied and by the reconfiguration of the existing methane drainage network.

The analysis carried out, together with the responsible personnel for degasification in Lupeni colliery, had facilitated the identification of certain feasible remedial maintenance and design measures able to improve the performance of the drainage system, such as:

- the systematic control required to identify the pipes sections with excessive air inleakages and pressure losses;
- the achievement of a regular maintenance of the drainage range, at pre-established terms and in certain specific conditions imposed by the Ventilation and Occupational Safety Department from Lupeni colliery;
- the replacement of pipes with a pronounced wear degree;
- the employing pipes having a lower frictional resistance;
- the use of new fittings, having an increased sealing degree.

The difficulty appeared from safety reasons, to perform structural changes in the network and to modify the actual operating regime, did not allowed the assessment of the effect of predicted remedial maintenance and design measures.

Anyway, a simulation was done for hypothetical conditions, this approach allowing us to put in evidence the facility of diminishing with about 25-40 % of pressure losses, and of air inleakage.

Through the appliance of optimization strategy of methane drainage network, it became possible to state the following conclusions:

- the analysis of methane drainage network of Lupeni colliery, with SIMDEG software, and following the presented stages of optimization strategy, represents the first type of such a kind of approach in a colliery from Valea Jiului coal basin, having a central degasification station;
- during the accomplishment of the above - mentioned study, a lot of intricacies had to be overcome, having technical basic reasons or issuing from the mining personnel's attitude and skills regarding the place and importance of safety equipments and systems;
- the change of view accorded to the importance of methane drainage, during the mining restructuring process in Valea Jiului, will allow the resumption of studies and „in practice" implementation of his results, with beneficial effects on safety levels and financial performances of mining activities;
- despite of all the difficulties encountered, the optimization strategy had proved his viability and can be clearly considered as a significant starting point in establishing a new and well - adapted methane drainage network auditing methodology.

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Data Collection, Visualization and Analysis in the Mining Factories

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ABSTRACT In an increasingly competitive mining marketplace, the ability to truly understand and control the factory operations is critical for success. The plant engineers and managers need to access to accurate, timely data to make informed decisions in real time. They need the power and security to precisely monitor and control every aspect of the process as well as the equipment and resources. With powerful MES (Manufacturing Execution Systems) solutions, they can visualize, control, analyze and optimize production data across the operations—resulting in enhanced decision making, faster time-to-market, improved productivity and reduced costs. The presentation will focus is on how to create unique approach to digitizing production operations by creating a unified, easy-to-configure "virtual plant" that provides unprecedented levels of analysis, reporting and insight throughout the operations.

1 INTRODUCTION

The realization of the automation project is made in one of the biggest mining plants with open pit in Bulgaria – Elatzite MED. We will shortly describe you how we automated the Milling process of copper ore in the plant with highly reliable standard based components from world class manufactures like General Electric, Entress+Hauser, End Armaturen and others. The main results from the new automation system are low energy consumption, less maintenance costs, highly efficient plant floor operations and streamlining the processes.

The main purpose of the project is to make sure that each of the 11 mills produces the most desirable fraction of the ore – particles with 0.080 mm size. The entire control of one mill aggregate is done by Programmable Logic Controller (PLC) VersaMax, from GE. The Human Machine Interface (HMI) is based on the award winning software platform - Proficy SCADA Cimplicity from GE.

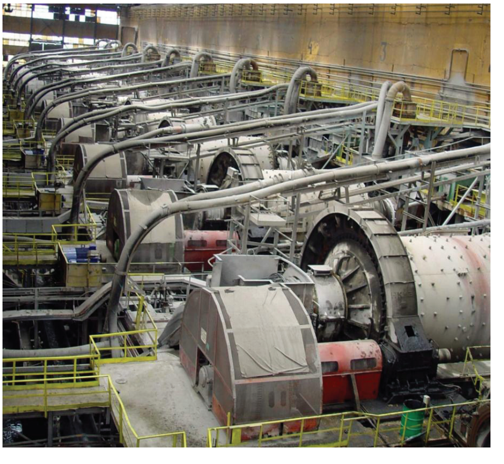

Figure 1. Milling department for copper ore in Elatzite ore-dressing factory.

2 ARCHITECTRE OF THE SYSTEM

As control systems designers we comply to the modern automation trends and architectures. We build the whole system from the sensor to the ERP system, connecting all layers in the production environment. At the bottom level we install the sensors, actuators, drives etc., and tie them in the control system (PLC). In the control and supervisory level we give to the operators, maintenance people and plant engineers the ability to interact with the process for further improvements. Here we capture the important plant floor data and put it into an industrial data collection and repository system - Proficy Historian from GE Fanuc.

Having the digitalization of data, we use standard software tools to put the data into a context and provide a meaning to the users. Using the real-time data, the plant engineers, quality, and maintenance engineers can see clearly the manufacturing picture with dashboards, charts, trend analysis, comparisons, benchmarks and KPI (Key Performance Indicators). With these advanced analytical tools the managers and operators can make better decisions to improve the process and achieve better understanding.

Figure 2. System's Architecture.

3 ENRGY SAVINGS WITH VARIABLE SPEED AC DRIVES

In this project, the energy saving benefits provided by the installed AC drives are also one reasonable criterion for the investment. For instance, in controlling pumps and conveyors, the AC drives improve process control by enabling collection of measurement and supervision information. Speed control also reduces the mechanical stress on the process equipment, increasing the life cycle of the equpment. The drive units are used with pumps, vibrating screens, crushers, conveyors, feeders and dosing. The main advantage from choosing variable speed AC drives for the automation project is to provide speed control that allows fully automated process control.

The installed AC drives are configured to operate via multiple fieldbuses. At the shop floor, they are connected to Profibus and Modbus. The use of Profibus reduces the amount of wiring and the number of connections. Long distances between the devices and the control rooms easily result in very high cabling costs, but thanks to Profibus, cabling is carried out in a less complex and more economical way. In addition, Profibus significantly reduces installation, commissioning and maintenance costs, and gives the system greater immunity to disturbance.

4 PROGRAMMABLE CONTROLLER

The controller (PLC) performs two major functions – it runs the control logic of the milling aggregates (the ball mills) and it regulates the technological parameters. The programmable logic controller tracks every possible situation, which may occur from the milling process and protects the unit from technological and electrical damages. The controlled parameters listed below: pressure of oil; temperature of oil; temperature and oil control of the slipping bearings; electrical protection of every electrical motions in the mill aggregate; pump production, conveyers and pumps speed, etc.

Figure. 3 The control cabinets.

The second major function of the PLC is connected with regulation of the technological processes. The goal is to achieve maximum performance of the milling units based on assimilation of the ore with desired grain metric structure.

4.1 Measurement of Technological Parameters

- Volumetric flow of process water.
- Volumetric flow of pulp.
- Ore mass flow.
- The measurement of the pulp density is before the hydro-cyclone and its outlet.
- Pulp level. The pulp level in the sump is measured by ultrasound level meter.
- Mill loading - the most efficient method is to measure the inside or outside noise, coming from the mill when it works.

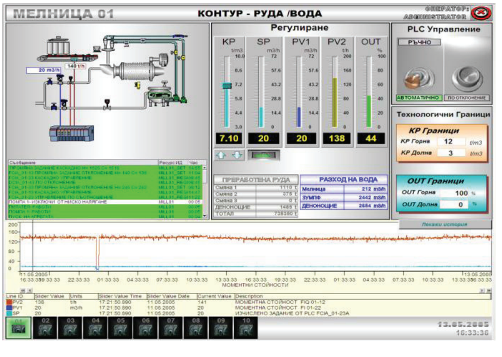

Figure. 4 Example of the HMI screen.

4.2 Advanced Control Loops of the Input Stream of Materials

- Mass flow of ore in the mill
- Volumetric flow of water in the mill.
- Volumetric flow of water in the sump before the hydro-cyclone classification.
- Pulp level in sump.

Figure. 5 Example of SCADA trends for process parameters.

4.3 Cascade Loops for Digital Regulating of the Regime Parameters

- Regulating the flow of water in the mill based on definite correlation with the mass flow of ore inside;
- Regulating the ore loading inside the ball mill;
- Regulating the density of the pulp which is ready for flotation;
- Calculating the circulating load and assessment of the effectiveness of the hydro-cyclone classification operation;
- Measuring the active power of the motor of the mill and controlling the ball-weight inside the mill.

The instruments for measuring the technological parameters are produced by Endress + Hauser (Germany) and Delta Instruments (Bulgaria).

5 OPERATOR STATION

The SCADA system monitors and visualize the process parameters like material and energy flow, temperature, density, pressures in real and historical time. It uses advanced graphics and animation objects to visualize the processes. The SCADA helps the engineers and operators to better control the production operations with excellent visualization capabilities. The system animates and provides alarms to any event with millisecond resolution, because of its fast communication with the other devices.

- ***Architecture*** – client/server. The central server visualizes and aggregates the data. There are multiple clients (thin clients) which are connected to the server. The clients are used to provide flexibility to the operators to deal with the production processes from multiple plant locations.
- ***Visualization.*** The SCADA system provides a clear picture to the production operations and equipment parameters with the help of advanced graphics and animation, process trends, alarms and statistical alarms based on the technological limits for the process. The operators can change set points, acknowledge alarms, and put comments on alarms for better analysis capabilities and fast reaction to problems.
- ***Data collection.*** The SCADA server passes the stream of data to the plant wide Historian. It also has its own SQL server to store alarms, events, text comments, downtime reasons, etc.
- ***Connectivity and Scalability.*** The system connects to various devices, PLCs and instruments via multiple specialized drivers and industrial standards like fieldbusses, OPC, XML, ODBC and others. The scalability of the SCADA allows the system to start with one server and grows with clients and redundant servers with the pace that the enterprise needs at the moment.
- ***Alarms.*** The alarm notification helps operators, maintenance, quality and production engineers to better utilize their equipment. They provide valuable information to the quality managers to inform them if the processes are in the technological specs and if the variance of the production processes is normal. The alarms are managed hierarchically, which means that only the important alarms will be seen first, to allow better root cause analysis of any given problem.
- ***Action calendar.*** This is a feature which helps the system to execute automatic operations based on calendar days and times. For example it can turn on automatically the lights at 20:00, or it can switch of the heating system during the weekends, etc.
- ***Video Graphical Replay.*** This advanced tool provides the ability to watch past periods of the process in real time and to observe what has happened on the shop floor to determine reasons for downtime or production out of specs.

6 PLANT-WIDE DATA HISTORIAN

The installed Proficy Historian is a plant-wide historian that digitizes the constant flow of data from the processes and equipment. The Data historian acts as a “black box” flight recorder capturing the real-time process data and storing these data points with a timestamp. This is an industrial system which is able to collect and store massive amounts of data from various shop floor sources and to pass them to the upper levels like SCADA, Information Portals (MES levels) for analysis and reports. Historian is very fast and processes up to 100 000 I/O operations per second. It compress and keeps the archives compact while performing high process speed of collecting data with 1 millisecond resolution. This product is highly scalable and easy to use and configure and it is used by many world class manufacturers.

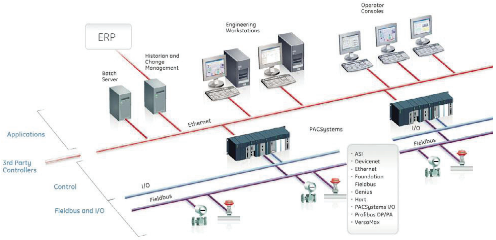

Figure 6. Architecture of the process control system.

The IT and control system in Elatzite is based on Proficy Historian and SQL. The process data with high resolution are processed by Historian, while the alarm and event data are stored in SQL server.

7 PRODUCTION INFORMATION SYSTEM

The implementation of Informational Based Control System in the milling department of Elatzite is the foundation of the plant automation structure. The plant managers are adopting the philosophy of viewing everything connected with the process everywhere. To digitize the plant operations we use structured approach to collect the process data and then transform it into real-time meaningful information for better and informed decision making.

7.1 Description of the Application

The application needs to capture data from a process data historian that is connected to the Mills' control systems. All efficiency events have to be captured automatically and indicate the start and end time of all events. If the system is not able to capture automatically the downtime reasons from PLCs and SCADA, the operator must choose from a drop down list for the event, three levels deep. Calculations are be done hourly with summaries provided at the end of the day and month.

The comprehensive analysis helps the managers to constantly improve the plant effectiveness and throughput. The reports are classified for different users and roles, so that everybody sees only the information relevant to his role. The system is client/server based and operates via internet or intranet with high level of data encryption and security.

The real-time and historical production reports including statistical alarms, downtime analysis, production counts, waste, energy consumption and others are easily exported into PDFs for any desired period of time. The system can notify users about some specific events that they are interested in via email or SMS.

A real advantage that the Information Portal provides is that everybody uses only standard web browser like Internet Explorer and can see his information from every location without installing additional software components.

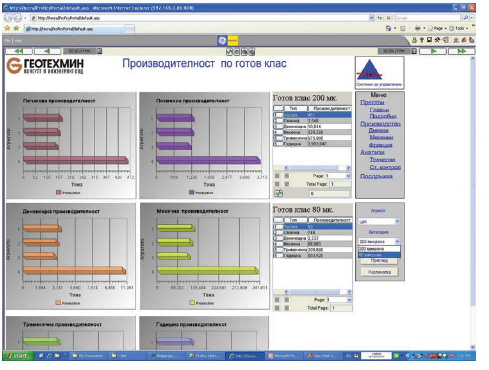

Figure 7. Example of production reports by product specs, mill aggregates, crew, selected time periods, etc.

They facilitate the job of managers and decision makers to constantly streamline, analyze and improve the plant operations. The communications between the lower automation levels, the supervisory control and the business levels are seamless. They allow continues process improvement and better understanding of the plant operations for greater profitability.

7.2 Statistical Process Control

With SPC (Statistical Process Control) charts the quality engineers can significantly improve the production quality by reducing the process variation and waste. For example the technology engineers now have the tools to compare the performance and process parameters of one mill versus other and to find out bottlenecks in the process. They can also impose different process variables to discover dependencies and process trends. In addition SPC alarms alert for out of spec conditions help to ensure consistency of the production.

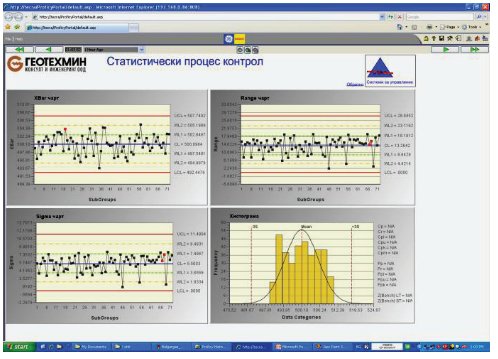

Figure 8. Statistical process control charts.

By drilling into a particular point the engineers can get more information of that specific item in production including run analysis, capability index Cp etc. Cp measures the capability of a process to meet its specification limits. It is the ratio between the required and actual variability.

7.3 Overall Equipment Effectiveness (OEE)

OEE is indicated by the *Industry Week* as a Key Metric for manufacturing. This means that the all mining plants have an opportunity to improve performance by reducing unscheduled downtime. Downtime reductions can be readily achieved by using OEE to gain visibility into machine status and to perform root-cause analysis of problems.

Fundamentally, OEE is a performance metric compiled from data on Machine Availability, Performance Efficiency and Rate of Quality that is collected either manually or automatically. These three data points are calculated as follows:

- *Availability* = (Operating time – Downtime) /Total Operating Time
- *Performance* = Total output/potential output
- *Quality* = Good output/total output

OEE is then calculated by multiplying those factors:

*OEE = Availability*Performance*Quality*

7.3.1 Availability

In this particular case the milling process runs 24x7x365 with 3 shifts. The planned maintenance time is subtracted from the total time, so the main component affecting the Availability is mainly the unplanned downtime. However to calculate the OEE the system calculates the scheduled and unscheduled maintenance time.

7.3.2 Performance

The performance is measured on each mill based on the speed of the process. The theoretical maximum production capacity of one ball mill is 140 t/h. Because the performance is varying constantly depending on the technological parameters and structure of the ore, the speed is measured and recorded in real-time. For example the speed of the mill aggregate may be slower because of a shortage of raw material or energy.

7.3.3 Quality

The quality of the produced pulp makes no sense to be measured as scrap or rework, although it affects strongly the flotation process and thus the quality and quantity of produced copper concentrate. We measure and record the output quality of the milling process with installed particle size meters PSM 400, which provide information about the fraction of the milled ore. The goal is to constantly produce particles with 0.080 mm size.

7.3.4 Key Performance Indicators Manager

The implemented OEE system on tree ball mills in Elatzite captures *reasons* for downtime (due to machine conditions, material status, production personnel or quality issues) and can encompass the individual machine level or a line or the entire plant. At the plant level, OEE metrics can be correlated with other plant metrics to provide Key Performance Indicators (KPIs). With enterprise level technologies, such as Executive Dashboard, managers are monitoring OEE plant metrics and drill down to find root causes of problems, getting minute-by-minute updates to enable real-time process improvement.

The implementation of the adequate OEE system brings immediate financial benefits to manufacturing operations.

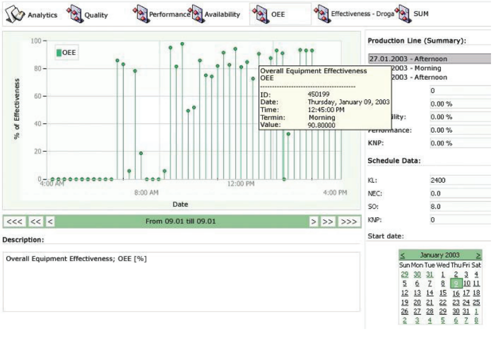

Figure 9. Real-time KPI manager for OEE.

7.4 Downtime Management

When a critical equipment of the process is inoperable, it brings downstream operations to a standstill. This can negatively affect delivery commitments to the customer, which in turn impacts cash flow and revenue. For example, in Elatzite, it is estimated that each hour of downtime for a critical unit of the milling process equipment can translate into \$50,000 of lost revenue. Conversely, reducing downtime by 1% on the 11 ball mills can provide revenue opportunities and cost savings \$2,500,000 nearing annually. The figure below illustrates report for analysis of the top reasons causing unplanned downtime. With drill-down capabilities the plant engineers can easily track the events and alarms that have preceded the breakdowns and focus their attention to specific areas for improvement. The information can be provided in Pareto charts to help easily distinguish the major reasons for stoppages versus the trivial ones.

Downtime Event Analysis is an important element in the overall OEE evaluation of a production environment. For further analysis the calculations of MTTR (Mean Time To Repair) and MTBF (Mean Time Between Failure) are automatically generated and displayed. The objective is to provide granular analysis of information that relates to specific KPI numbers.

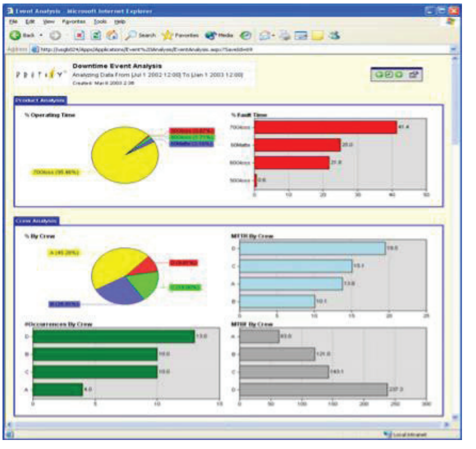

Figure 10. Example of downtime report.

7.5 Reduced Repair Costs

The OEE monitoring system in real time enables predictive maintenance that dramatically reduces repair costs. As the historical database of downtime reasons grows, the maintenance department can discern trends to predict an impending failure.

7.6 Increased Personnel Productivity

The OEE system enables the shop floor to go paperless. Typically, facility operators and supervisors spend an enormous amount of clerical time recording, analyzing and reporting downtime reasons and root causes on paper, then further explaining these reports to management. An OEE system captures and reports downtime and efficiency automatically. This saves time lost in non-value added reporting activities and allows personnel to focus on more valuable tasks. With OEE, everyone from the plant floor to the boardroom is more informed, more often, more easily.

7.7 Manual Efficiency Tracking System

Prior to this project, the Milling department had a manual efficiency tracking system. Operators were required to enter events and reasons for downtime manually. Log sheets were collected and consolidated in Excel spreadsheets and results were made available at the end of the week or month. The manual system provided an after-the-fact picture of the Mill status, and the results were completely dependent upon the consistent manual entry practices of the operators.

7.8 Limitations of Manual Systems

Manual systems "forget" events. They have a tendency to filter out high frequency (short duration) events because of the manual work involved to track these frequent occurrences. If operators are busy troubleshooting, they may not have time to log events consistently or accurately. If an event generates a lot of paperwork, it tends not to be tracked. The second very important limitation is that manual systems are not capable of tracking in real-time. Results are not known until the end of the week or month. There is therefore no opportunity to react in real time to a problematic situation on the shop floor. The third limitation of manual systems is that some events cannot be tracked manually because of the speed of the process and the quantity of information to be logged. The manual system will incorrectly indicate a higher efficiency than truly exists.

7.9 Organizational Critical Success Factors

Selecting an internal "champion" that has credibility with Mills management is the first critical success factor. The person needs to verify event numbers and have to be able to defend them with management and the different departments. Mills management initially rejected the numbers generated by the newly implemented system because it showed lower efficiencies than the previous manual system. This is normal because an automated, real-time system exposes lower efficiency and does not forget events. It also tracks events that cannot be tracked manually. In order to confirm the event numbers, the Elatzite appointed an internally credible individual to verify the data. This is one of the most important critical success factors in the project. After verification, this person explained to management the differences between automatic and manual systems and that the real-time numbers are correct. The system gained credibility after this.

The last critical success factor is in creating a team of people who can utilize the information. The system is only an enabler – it does not solve the problems. It will indicate where it is necessary to focus problem solving. In this case, Elatzite wants to be an industry leader. Its motivation was a requisite for success.

7.10 Payback

The first payback is time. There is no longer a loss of time in performing the following critical functions:

- Capturing data
- Making calculations
- Discussing the validity of the data and the calculations

Second, there is now one snapshot of plant efficiency in real-time so that plant personnel can be more accurately evaluated. The second payback is in the capability to act in virtual real-time and not after-the-fact. A situation that creates an efficiency loss is reported quickly, as is an indication of the specific downtime reason. Action is now possible almost immediately, and the status of the machines and the reasons for downtime are propagated in real-time throughout the Milling department.

The third payback comes from the fact that plant personnel know exactly where to act to increase efficiency. The system gives the primary reasons for downtime, loss of performance and material loss.

The documented increase of those 3 mills aggregates is 8% over six months (from 62 to 70% OEE). The Milling department is now extending the use of the system to the rest 8 ball-mills.

8 REAL-TIME ASSET MANAGEMENT STRATEGY

The automation and IT system will grow in the future with implementation of Proactive Maintenance Strategy. It will allow the Plant Supervisors/Personnel, Maintenance Engineers or other users to configure "important critical events" that need to be continuously monitored in the plant during regular operations.

8.1 Increasing the Plant Production Readiness by Improving Plant Reliability

One can say that production readiness is a cascade effect of improving asset reliability. When asset reliability improves, failure components of an asset reduce. This in term results in low downtimes and lower downtimes mean more plant availability and more plant availability leads to increased production readiness.

8.2 Current Practices

Today maintenance practice in many organizations are either based on run-to-failure, for what is known as Reactive maintenance, or calendar based Preventative maintenance.

8.3 Reactive Maintenance

Reactive maintenance of an asset means that the set asset is allowed to run until it fails and then the fault is fixed. This normally causes unplanned downtime of machines, because machines do go down unexpectedly. On an average more than 20% of the machine downtimes in the industry are due to unplanned shutdowns.

On other hand calendar based maintenance involves scheduling maintenance activities over periodic calendar based intervals. For example maintenance is scheduled every 1st Monday of every quarter or every 2th Tuesday of the month. This doesn't exactly take into consideration the actual usage of the assets on day-to-day bases, but relies mainly on elapsed time to indirectly predict usage that results on over-maintenance or under-maintenance. Over-maintenance is costly and under-maintenance results in unplanned breakdowns. So both are undesirable.

8.4 Proactive Maintenance

The right approach is to include Proactive maintenance as part of the overall maintenance strategy. This means that first we need to characterize and categorize the assets of the plant into two groups:

- Those that are critical to maintaining the normal expected uptime of the plant.
- Those that are non critical to maintaining the normal expected uptime of the plant.

Simply put a proactive strategy prevents failures. There are two kinds:

- Preventing failures by scheduling maintenance based on usage.
- Predicting failures before they occur based on symptoms form certain machine conditions.

8.4.1 Usage based preventative maintenance

There are three steps to implementing usage based preventative maintenance for an asset:

- Estimate how much an asset can be used before the risk of failure escalates and set an usage threshold based on the estimation.
- Collect data continuously from the machines to check if the threshold is crossed.
- If the threshold is crossed order preventative maintenance as soon as the threshold is crossed.

8.4.2 Predicting failures based on conditions

We need to monitor the conditions of the assets continuously. This means collection of data from the machines on some vital process variables that will provide a good understanding of the health of these machines. If one or more values of these process variables show undesirable values, than we need to order maintenance activity immediately even while the asset is still producing. This will cut unplanned downtimes significantly.

9 CONCLUSION

This project was successfully implemented in Elatzite ore dressing plant. The objective was to automate the Milling process of copper ore with modern automation equipment and to track in real-time and in automated manner the following key performance indicators of the milling aggregates:

- Availability (equipment failures and maintenance)
- Performance (speed of machines)
- Quality (losses of ore milled outside the specs)

The milling department had a manual tracking system and was migrated to a fully automated system. It is able to compare the theoretical capacity and the real production capacity of the aggregates every hour.

Today the reasons for the differences between the theoretical capacity and reality are indicated and logged. Continuous improvement teams can now work on the primary reasons for downtime, performance and quality problems. All efficiency data is archived for long-term data analysis. And wide-scale distribution of efficiency information in the Milling department is achieved by publishing reports on its intranet.

In addition the management is looking to implement proactive maintenance strategy to reduce unplanned breakdowns and reduce significantly the maintenance costs.

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Introducing the Fuzzy Logic Control at the Information Management System of the “Rudnik” Mine Flotation Plant

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ABSTRACT The flotation plant of the “Rudnik” Mine is a contemporary and active flotation system with reliable monitoring of the production flows. This was the decisive point in selecting this flotation system for the portion of our virtual experimental tests of fuzzy logic application in support to decision making and management in mineral processing. The paper gives a review of the real system and its fuzzy model, shows the tests results and an assessment together with conclusions, and a proposition for the introduction of fuzzy logic in all hierarchical levels.

1 INTRODUCTION

Mine "Rudnik" is a mining production system with stable production and a constant increase in production value. The main features of the production and technical "Rudnik" Mine systems are: standardization, contemporary technical and technological solution, comprehensiveness of the production chain from underground exploitation to final products - lead, zinc and copper concentrates, mechanized production, relatively low computerization and an uneven utilization of computer supported technologies in business and production functions of the company (Vujic et al, 2001).

The target production of lead, zinc and copper concentrate, is accomplished at the flotation plant of the "Rudnik" Mine. The flotation plant is a delimited productive technical and technological entity, with uniquely determined production structure - equipment, phase structured in an unique technological entity. The process is mechanized, continuous and divided by phases.

Apart from the technological and technical features, the mineral processing process is characterized by the following properties: ore quality, immediate costs of processing (energy sources, flotation agents, etc.), and the concentrate quality. The plant condition, immediate and overall production, variable ore quality and the demands for fixed concentrate quality, consumption of flotation reagents, electric power and other resources and the manpower data determine the process image of the ore processing immediately and provides elements for the analysis of the success of the flotation concentration plant and issuing the timely management decisions.

2 FLOTATION CONCENTRATION INFLUENTIAL PARAMETERS

Flotation concentration process is typical by a large number of influential parameters, i.e. the input variables influencing the process. Regarding the process control, variables may be classified as the depending (disturbance, and variables that can be influenced) and independent (intermediate and variables that describes the performance).

Regarding the process flow, i.e. the flotation concentration of the polymetallic ore of the "Rudnik" Mine, and the monitoring of collector (KAX) consumption and the activator (copper sulphate) as the foundation of the computer integrated system, the fuzzy logic model is related to the zinc mineral flotation circuit. Apart from certain simplification, this holds no or little influence on the validity of experimental tests, their significance and their value in drawing conclusions.

Table 1. Mine „Rudnik" mineral processing phases.

Process	Capacity (t/h)	Output size, (mm)
Primary crushing	146.69	145
Secondary crushing	184.03	30
Tertiary crushing	88.92	16
Vibro sieveo	216.66	-
Grinding	34.58	-
First stage classification	41.80	-
Second stage classification	-	65-70% (-0.074)

For the purpose of model "training", the data on the "Rudnik" Mine flotation plant operation, routinely collected by the responsible service at the mine, as well as the data collected by process monitoring at the plant were used. In order to extract the parameters participating in the model, a statistical analysis based on the general linear models ANOVA - Analysis of Variance was used.

Table 2. Agents regime of the flotation concentration process.

Agents' purpose	Type	Consumption (kg/t)
pH regulator	$FeSO_4$	0.4
	CaO	2.0
Depressing agent	$NaCN$	0.045
	$ZnSO_4$	0.10
	$Na_2Cr_2O_7$	0.001
Activator	$CuSO_4$	0.20
Collector	KEX	0.03
	KAX	0.05
Frothing agent	Dowfroth 200	0.06

A particular combination of factor levels was determined for each experimental unit for the purpose of tests planning, or parameters selection. Since the intention was to test the effects of the simultaneous influence of a large number of factors, the tests were designed to examine the influence of each individual factor, as well as mutual interaction. As the foundation for the determination of the order of significance, a particular form of the variance analysis - the factor plan was used. For seven parameters, extracted based on the experience, tests in virtual environment of the Fuzzy Logic Toolbox were performed. (Ćalasan, Petkovska, 1995; Etter et al., 2005).

Out of large number of parameters influencing the flotation concentration process, the following parameters were separated with particular influence on the Zn minerals flotation process: copper minerals flotation tailings zinc content (metal content – zinc feed – UZN), collector consumption in g/t (KAX), pulp density at the end of copper minerals flotation circuit (GPU), activator consumption in g/t (ACU), solution pH (PHS), grinding fineness (FML), and flotation time (TFL).

Table 3. Determination of the variables order of significance.

Conditions	Minimum	Maximum
Feed zinc content (%)	0.95	2.71
Quantity of collector added (kg/t)	0.009	0.040
Quantity of activator added (kg/t)	0.009	0.155
System pH	10	11.5
Quantity of frothing agent added(kg/t)	0.037	0.060
Quantity of collector added in the Pb and Cu flotation circuit (kg/t)	0.013	0.031
Quantity of Zn mineral depressing agent added in the Pb and Cu flotation circuit (kg/t)	0.054	0.130

The following were selected as the output variables, or parameters showing the success of the flotation concentration: quality of the concentrate expressed by the zinc concentrate content (KZN) and the metal (zinc) recovery in percentages (IZN). Such results of the analysis were expected, thus confirming the criteria of “knowing the process” as a suitability measure for the fuzzy technology application in mineral processing.

The values of the pulp density (GPU), grinding fineness (GPU) and flotation time (FTU) were considered as a constant for the purpose of the development of this model.

Pulp density is kept at approximately 40% (solids), grinding fineness is 62% (-0.074 mm), and zinc minerals flotation time. High pH is achieved and maintained by adding certain quantities of lime which cannot be separated from the overall consumption of lime. Based on the theoretical knowledge on the influence of pH on the zinc minerals flotation and by knowing the ore being processed in the plant (polymetallic ore of lead, copper and zinc with high pyrite content), it was adopted that pH vary between the extremes of 10 and 11.5 for the purpose of this research.

3 FLOTATION CONCENTRATION PROCESS FUZZY LOGIC MODEL

During the process of fuzzyfication and rules optimization, membership functions for every adopted input and output variable. For every membership function, an appropriate type and range was determined and assigned.

By combining the linguistic values from both fuzzy rule bases (of input and output values) with the aid of logic operators of conjunction (AND) or disjunction (OR), a fuzzy rule is created. In accordance with the procedure describe, rules such as the following one are created.

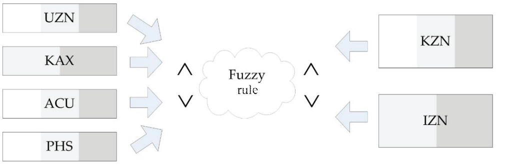

Figure 1. Creation of fuzzy rules.

IF the UZN is „high“ AND consumption of KAX is „high“ AND consumption of ACU is „medium“ AND PHS is „high“,

THEN the KZN is „high“ AND the IZN is „low“

Since the influence of four input variables with a total of 16 possible conditions on two output variables with five conditions was tested during the experiment, large number of fuzzy rules was created during the process

With the aim of verification of the fuzzy model developed, the comparison between the results obtained with the model and the experimental data from the flotation plant of the „Rudnik“ Mine for the year 2007.

For the comparison data (plant data for January – October, 2007), a Pearson correlation coefficient was calculated. For the zinc concentrate quality the value is 0.681 (significant relation between the data), and 0.781 for zinc recovery (strong relation between the data).

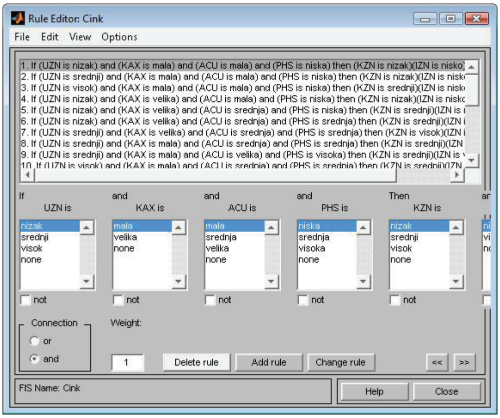

Figure 2. Fuzzy rule definition.

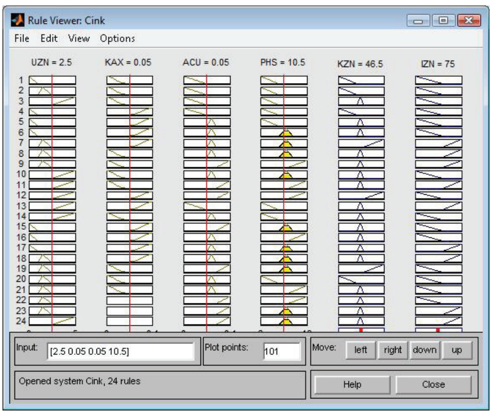

Figure 3. Resulting fuzzy model.

This result, according to which the model displays better results for zinc recovery is expected, having in mind high pyrite content in the ore, and the technology schematic with the open circuit as a consequence of this fact.

4 RESULTS, ASESSMENTS AND CONCLUSIONS OF THE EXPERIMENT

The tests in the virtual ambience of Fuzzy Logic Toolbox (Matlab) were accomplished on the flotation concentration process of the „Rudnik“ mine. To decide in favour of selection of this flotation plant for experiments, the following facts were of utmost significance:

- „Rudnik“ mine, with its flotation plant is an active production system;
- There is a longstanding continuity of selective and precise measurements and monitoring of all the key operational parameters of the mineral processing plant and the Mine as a whole;
- Availability of data;
- Possibility of data check;
- Longstanding experience in designing this plant and consultancy and excellent knowledge of the process system brings additional safety in assessments and estimations.

Due to monitoring of the collector consumption (KAX) and the activator – copper sulphide (ACU), the experimental fuzzy model was set for the zinc minerals flotation circuit. This had no influence on the validity of experimental results, it isn’t diminishing the tests accomplished and it had no influence on the generalization of conclusions.

The analysis of the essential connections between the zinc minerals flotation process parameters and its influence on the process outcome, was accomplished by using the statistical data collected during longstanding monitoring of operation at the mineral processing complex of the „Rudnik“ Mine.

The results shows that the fuzzy model developed describes the zinc mineral flotation process at the "Rudnik" Mine flotation plant. A positive correlation with a significant connection between the input values and the zinc concentrate grade (0.781) and a strong connection between the data for zinc concentrate recovery.

The results by which the model describes the process better for the zinc recovery is expected because of the high pyrrhite content in the ore and the open circuit technology, as a consequence.

The resulting fuzzy model may be considered theoretical, since the input values originate from the theoretical concentrate balances, and the product masses were not considered. In addition, the models disregards the change of three parameters important for the flotation concentration process.

In setting the fuzzy model the assumption was made that the allowed freedom in fuzzy connection selection between the parameters of flotation process with the aim of establishing the relation:

With this approach, the error, among other things depends on:

- Selection of mutually connected parameters;
- Utilization of simplified relations and insufficient number of parameters in model

Within the Mine "Rudnik" zinc minerals flotation process fuzzy model, the sufficient number of parameters for a complete depicting and comprehensive insight in the real process was analysed. This is more accurate than the utilization of the simplified model with a few parameters, from the methodological point of view.

Sensitivity of solutions obtained by the fuzzy modelling was accomplished by the type and width of the interval of fuzzy set membership function. In this case, sensitivity intervals were established upon the practical experience. The experimental knowledge shows that the development of flotation process fuzzy model is a demanding and complex task, as-suming quantity, precision and reliability of the statistical data for production active flotation plants, or the existence of high quality analogue data in cases of fuzzy model development for a new flotation plant.

The reliability of the fuzzy model is predominantly influenced by the number and the representation level of the flotation process integrated parameters. The functional topology of the model is correlated with the functional topology of the real system, while complexity depends on the number of parameters relevant for the process, and the physical and functional topology of the flotation system.

By comparing the model approach with (earlier) experiences of the flotation model development on the basis of traditional mathematical approaches, it is not sufficient to just state that the modelling procedure is far more simpler, but also that the depicting of the real process is more accurate, les demanding, adaptive and cheaper.

5 FUZZY LOGIC CONTROL OF THE PROCESS

Based on the fuzzy model of the zinc minerals flotation circuit, the generalizations made to the whole flotation concentration process and previous experience with this particular system, it is proposed that information-management system of the flotation plant should be based as firmly integrated and hierarchically distributed computer based system for surveillance and monitoring over production processes, vertically connected with business flows in the company, and higher decision making levels. Hierarchical structure is based on five levels of surveillance and management:

The first level consists of measuring and regulation equipment installed at the machines and equipment in the flotation concentration plant, on the systems for electric power and compressed air supplies.

The concept of the second level is based on process computers (PLC – programmable logic controllers), with integrated fuzzy logic controllers, for the primary acquisition of data from the first level and appropriate response ability, according to the results of fuzzy reasoning.

The third level consists of computers with SCADA functions. This level holds the executve surveillance and management functions, and it is vertically integrated with higher decision making levels, specialized services, operational management and central management of the Mine.

The fourth and the fifth level are equipped with computers and software for business and engineering, creative analysis, planing, expert assessments, data archiving, decision making, data interpretation, etc. By position within the system hierarchy, this level is enabled to access all other levels and groups of secure data.

Fuzzy reasoning and application of fuzzy logic control in this flotation system is proposed to be introduced at multiple levels. The way of utilization of fuzzy logic and fuzzy mathematics depends on the vertical position of the level, ranging from actual real time control at the ground level to development of fuzzy logic aid for decision making at the top management levels.

The model substantially differs in concept from the models developed so far (Flintoff) by integrating vertical fuzzyfied levels, which is fully justified by the complexity of the real system.

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NETPRO/Mine: An Integrated Resource Modelling, Mine Design and Automation Software

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ABSTRACT This paper describes a software for database management, geological solid modelling, surface or underground mine design, planning and automation. The software has a multi-module structure with friendly user interface. An actual case study is used to demonstrate the abilities of the software.

1 INTRODUCTION

The computerized modelling of geo-objects such as mineral deposits and their mine planning is a challenging task of the modern mining industry. Mining companies are aware of the power of computer technology and wish to use this power in 3D modelling and planning their resources.

Turkish Coal Enterprises (TKİ) is a leading public company in lignite sector of Turkey, achieving 43% of total lignite production of Turkey and having 21% of total lignite resources (TKİ, 2009). TKİ aims to produce its lignite resources efficiently and in accordance with modern mining science and technology and therefore to wish using an integrated software for geological solid modelling, mine planning and data management. For this purpose, commercial softwares such as Gemcom-Surpac, Minex have been used but a desirable success has not been achieved due to insufficient training support, language problem and inability to solve the local problems. Therefore a project has been launched by TKİ, ULUSAL (CAD and GIS Solutions Corporation) and MODEST (Ore Body Modelling and Mine Design Team of the Department of Mining Engineering at Hacettepe University), supported by TÜBİTAK (Scientific and Technological Research Council of Turkey). The project aims to develop an integrated software with abilities of database management, digital terrain and geological solid modelling, block modelling, resource estimation, surface and underground mine design and planning and effective monitoring and control of production stage.

This paper introduces the development of a part of the software which is used for database management, geological solid modelling, block modelling and resource estimation. The use of software is demonstrated through a real case study.

2 DESCRIPTION OF THE SOFTWARE

NETPRO/Mine has been developed using C# in .NET and Microsoft Visual Studio 2005 and 2008 as a developer tool. NETPRO/Mine is designed to be in Turkish for now. General characteristics of the software are given as follows:

- Provide 3D modelling and visualization tools for ore body characterization and production planning in surface and underground mines,
- Support local mining engineering practices and international techniques,
- Multi-module structure,
- Include general CAD and GIS properties,
- Support Google Earth and GPS,
- Able to run with standard equipment,
- Easy to use and perform well,
- Support standard data formats in mining industry,
- Read and Write the data of well known ore body modelling and mine planning softwares.

Some of the tools developed so far are described shortly as follow:

Database Tool is designed in such a way that application coding does not change depending on the type of database (MS Access, MS SQL, etc.). This tool provides a number of functions for data entry, control, import/export, management, editing, filters, statistics, compositing, reporting and 3D visualization of drill holes in various forms.

Surface Modelling Tool is basically developed for digital terrain model (DTM) and procedures derived from it. The tool supports Google Earth and provides functions for reading DTM data from database, construction of DTM, contours, and relief. The functions for construction of DTM and contours are provided by Netcad. Netcad is a CAD and GIS software providing solutions in a wide range of applications and is developed by ULUSAL. In addition NETPRO/Mine itself is able to construct DTM and contours, based on 2D Delaunay algorithm. NETPRO/Mine is able to share the same data with Netcad and show the data in the same 3D screen.

Geological Solid Modelling Tool is a tool for cross-sections, slicing, and construction of 3D solids from the sections. The cross-sections are able to take dynamically in any direction and visualize in 3D. Cross-section function has ability to clip and slice. In addition broken sections can be constructed. 3D geo-objects are obtained by combining various cross-sections and the corresponding volumes and areas can be reported. 3D grids are also possible.

Variogram Tool calculates experimental variograms in any direction and fits a model to them. For this purpose an interface to open source SGeMS-Geostatistics Software is written. In addition to functions involved in SGeMS, NETPRO/Mine is able to make automatic model fitting and show experimental variograms in a single chart.

Block Modelling and Interpolation Tool is basically designed for dividing 3D solid model into small mining blocks and estimating the mean values of these blocks for any attribute using one of three estimation algorithm: nearest neighborhood, inverse distance and kriging. For estimation by kriging, an interface to SGeMS-Geostatistics Software is written.

Modules for fault modelling, surface mine design and underground mine design together with ventilation network analysis and blasting design will be added to NETPRO/Mine in near future.

3 CASE STUDY

Data come from a lignite mine located in Turkey and are provided into four excel tables: collar, survey, sample and geology. Tables include the following information:

- Collar table; identification number, northing, easting, elevation and depth of the drill holes,
- Survey table; azimuth and dip angles of the drill holes,
- Sample table; drill hole id., depth-from, depth-to, calorific value, ash content and moisture content of the core samples collected from drill holes.
- Geology table; drill hole id., depth-from, depth-to and lithological identification of the core samples.

The number of drill holes is 102. The number of the core samples analyzed is 526. The data is entered into database using data reading interface (Fig.1). In this study ash content is considered only. Figure 2 shows the drill holes in 3D. The drill holes are colored depending on the lithology of the units intersected. Lignite is shown in red.

The average length of the raw samples is 1.63 m. This is calculated by using statistics function of database tool (Fig.3). The raw data is compozited for 2m with 50% acceptance limit (Fig.4).

The experimental variograms of ash content are calculated in various directions (down hole direction and horizontal direction) and a model is fitted (Fig.5). The model consists of a pure nugget effect plus spherical scheme.

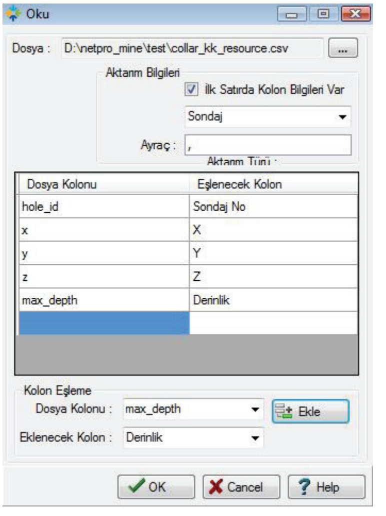

Figure 1. Data reading interface.

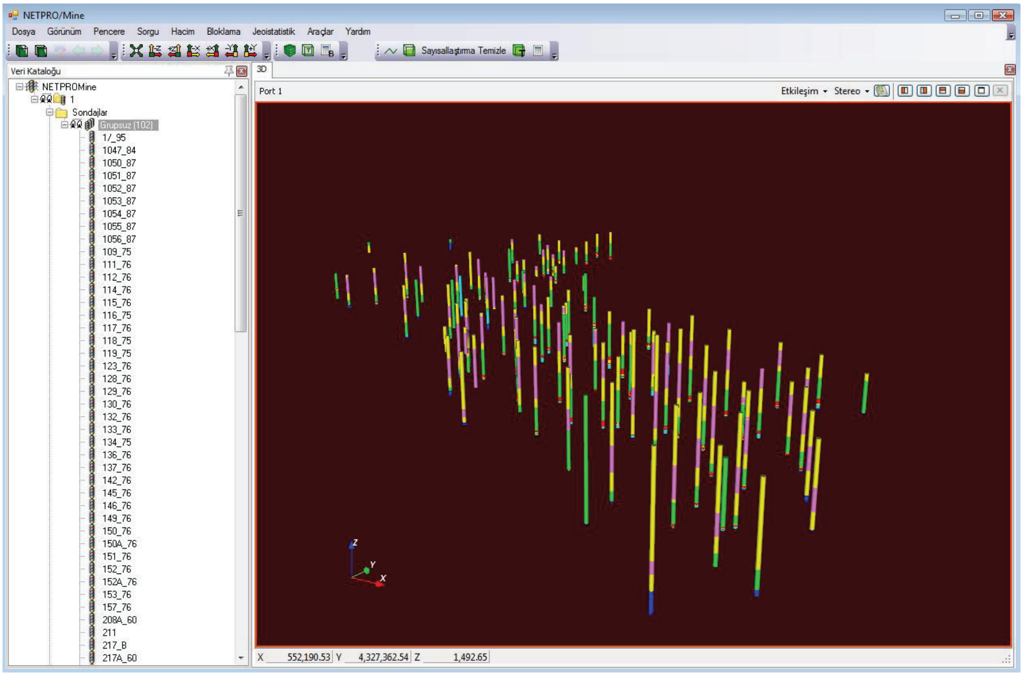

Figure 2. 3D view of drill holes.

Figure 3. Summary statistics of raw ash values.

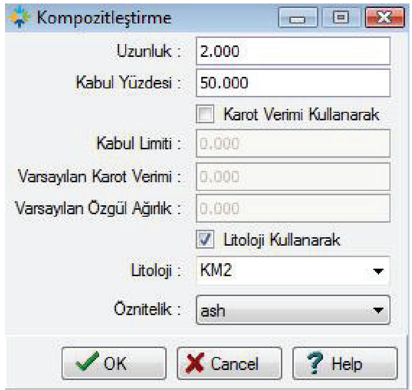

Figure 4. Compositing interface.

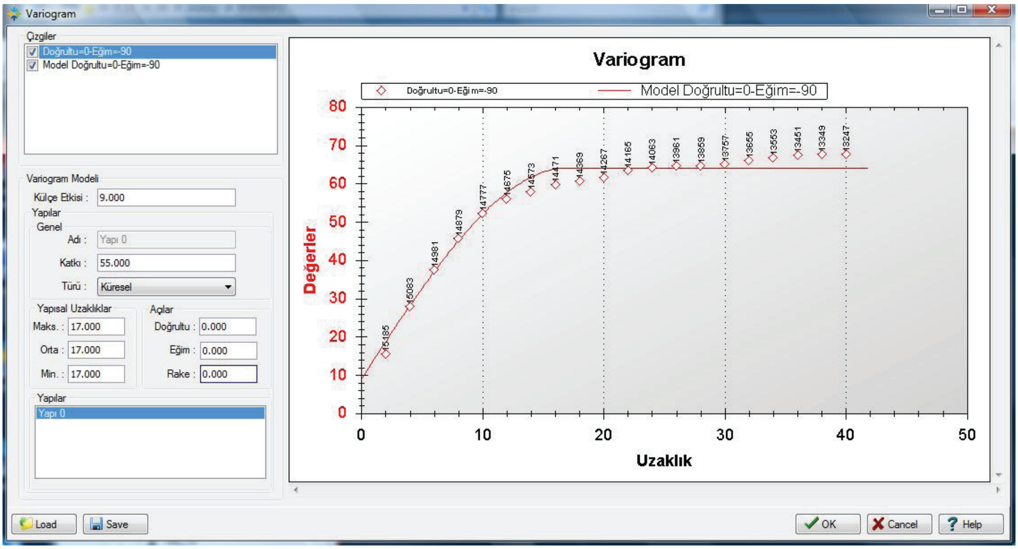

Figure 5. Downhole experimental and model variograms for ash.

The deposit is highly faulted and the faults divide the deposit into three coal blocks. Only one block is considered and various cross-sections are drawn manually in order to define lignite boundary for this block (Fig.6). Then these sections are combined to construct a solid model (Fig.7). This is the geological solid model for the coal block considered. The volume of the solid is 17 Mm^3.

The block model is constructed into the solid model using block sizes with 20mx18mx2m and the mean ash content of these blocks are estimated by ordinary kriging (Fig.8).

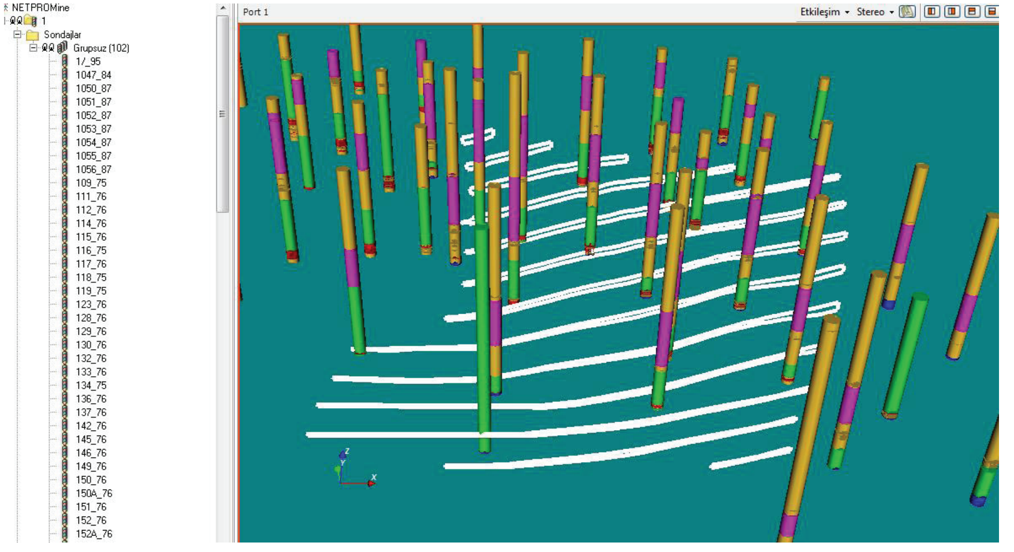

Figure 6. Cross-sections taken in EW direction.

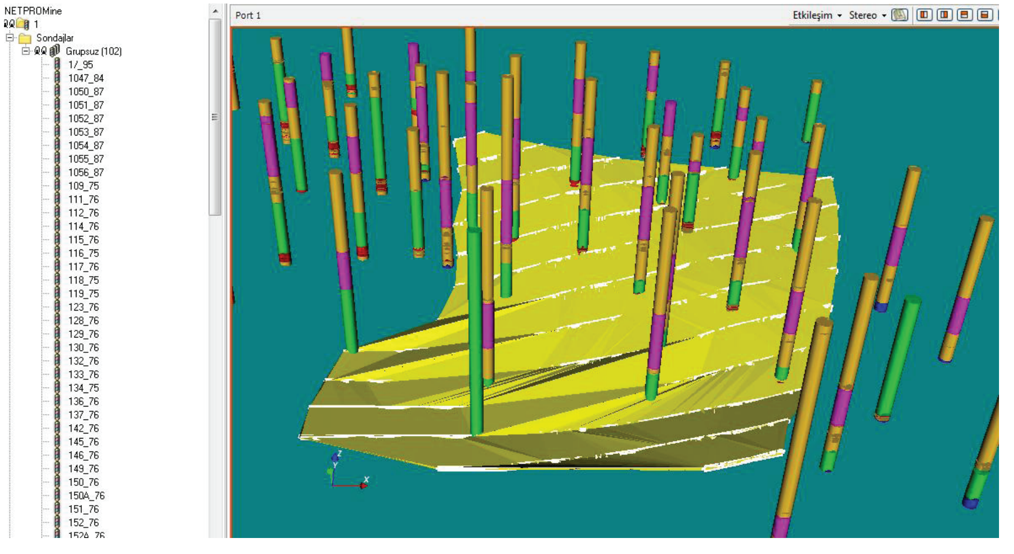

Figure 7. Solid model created from the cross-sections.

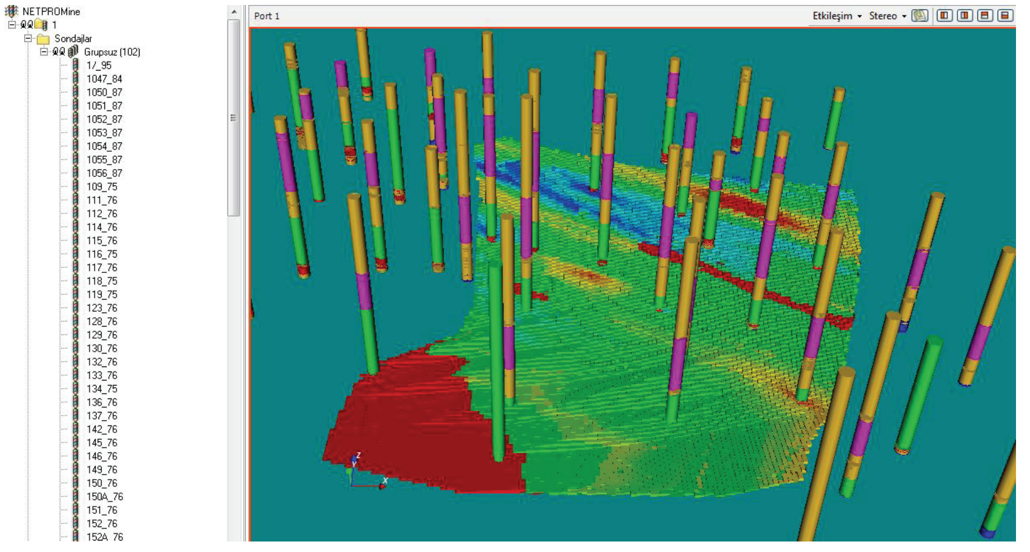

Figure 8. Block model inside solid.

4 CONCLUSION

NETPRO/Mine is an integrated software with abilities of database management, digital terrain and geological solid modelling, block modelling, resource estimation, surface or underground mine design and planning and effective monitoring and control of production stage. The tools for mine design and automation will be added to the software in the near future. As demonstrated in this paper, NETPRO/Mine will be a very powerful tool not only for TKİ but also all mining companies and consulting services.

ACKNOWLEDGEMENT

The work presented here was supported by Scientific and Technological Research Council of Turkey (TÜBİTAK) under the grant 108G036.

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Determination of Particle Size and Shape Characteristics of Quartz Sands

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ABSTRACT The ever-progressing development of industrial processes and products regularly requires significant progress in the development of associated measurement techniques. With the advance of technology, there have been many developments in mining sector. Mineral particle size is a critical parameter in any process involving the liberation and separation of minerals. In most mineral processing plants, product grade and mineral recovery require sufficient mineral liberation and optimum size distribution. There are many methods of measuring mineral particle size. Sieving, sedimentation, microscopy, digital image processing, laser diffraction are the most common particle size analyzing methods. The shape of the particles plays an important role in the assessment of particle size distribution. Most sizing techniques, however, assume that the sample being measured is spherical, as a sphere is the only shape that can be described by a single number. Therefore, different techniques can give different results for the same sample depending on this aspect. Within the scope of this study, the particle size distribution of two different sand samples (Sarikum and Senkoy) was assessed by sieving, digital image processing and laser diffraction techniques. Additionally the convexity and circularity parameters of the samples were measured by Morphology G device. The particle size distribution results obtained from different techniques for each sample were discussed depending on the values of convexity, circularity and elongation of the sample particles.

1 INTRODUCTION

The determination of particle size distribution for several raw materials used in glass, cement, paper, plastic and ceramic industries and mineral processing operations is important for the success of whole operation. Size distribution, specific weight, specific surface area and particle shape factor are important parameters in identification of these materials (Rona, 2006). During many stages of mining, which include successive operations, a proper identification of particle size distribution is important in terms of productivity and economy from mine to mill. Each department of mine uses different methods of particle size measurement depending on their own conditions and priorities (Kursun, 2009).

Assessing the particle size distribution of the material, which is processed to the different stages such as crushing, grinding in mineral processing, is critical for the productivity control of total mining operations. Minerals are separated from each other depending not only on mineral properties, but also particle shape (Ulusoy et al. 2007, 2008). In mineral processing, flotation, jigging, sieving, classification, shaking table, dense media separation, cyclone separation and the other many unit operations are sensitive to particle shape. Most sizing techniques assume that the material being measured is spherical and report the particle size as the diameter of the "equivalent sphere", which would give the same response as the particle being measured.

The way the equivalent sphere approximation works is shown above for an irregularly-shaped particle. An example of the application of the equivalent sphere approximation is shown in Figure 1. (Kippax, 2005).

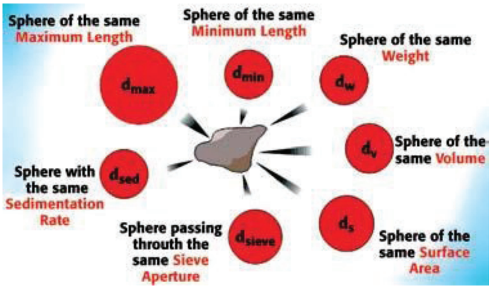

Figure 1. Equivalent sphere representations for an irregularly shaped particle.

Sieving is carried out with wet or dry materials and the sieves are usually agitated to expose all the particles to the openings. When applied to irregularly shaped particles, sieving is complicated by the fact that a particle with a size, which is close to that of the nominal aperture of the test sieve may pass only when presented in a favorable position. As there is inevitably a variation in the size of sieve apertures, prolonged sieving will cause the larger apertures to exert an unduly large effect on the sieve analysis owing to the irregularity of weaving (Wills 1985).

Sieve analyses presents three major difficulties: With woven wire sieves the weaving process produces three dimensional apertures with considerable tolerances, particularly for fine-woven mesh. The mesh is easily damaged in use. The particles must be efficiently presented to the sieve apertures (Allen 1997).

Within the scope of ore dressing, the size determination is generally made on the basis of size range through which particles can or cannot pass. In the simplest sense, the size distribution can be determined through the use of sieves. When the size or size range of the material is determined through sieves, the third dimension depending on particle shape factor is ignored.

2 DIGITAL IMAGE PROCESSING METHOD

The developments in image capturing technology in recent years, notably in the computer technology, have increased the usage of the image processing methods, which are used for measuring and observing in various research areas. For this reason, image processing and analyzing techniques have been investigated intensively and thus many new techniques have been applied in an industrial scale (Ozdemir 2003, 2007). Digital image processing method in mineral processing has become one of the most important improvements in mining sector by providing reliable data and time saving features. This method has played a significant role in ore processing area by performing the same capacity compared to the conventional mining methods and displaying the same results in a shorter time (Karakus 2007). The study of image analysis techniques in geology, engineering geology and rock mechanics started in the last two decades. Especially in different areas, such as physical properties of rocks and their engineering properties, which are difficult to measure, researchers have considered image analysis methods as an alternative. Research has been conducted under the titles identification of minerals under microscope, determination of void ratio of rocks (porosity), analysis of transients, determination of the limits of minerals, determination of the limits of particles, particle size distribution and formal analysis. Image processing techniques have been also used for determining the sieve analysis of conglomeration of minerals with different sizes. These studies conducted research at a micro-level and made size determination on the images obtained from optical and electron microscope. Jenkins et al (1991) studied the industrial application of the image processing method with the same principle and the effect of particle sizes on refractory magnesite (Goodchild, J. and S., Fueten, F. (1998).

3 LASER DIFFRACTION METHOD

Laser diffraction has become one of the widely used techniques for particle size analyses in many industries. The first laser diffraction system was first introduced in the 1970s. With the developments in technology, image processing with laser is now used in size specification. The technique of laser diffraction is based upon the principle that particles passing through a laser beam will scatter light at an angle that is directly related to their size. As the particle size decreases, the observed scattering angle increases logarithmically. The observed scattering intensity is also dependent on particle sizes and diminishes to a good approximation, in relation to the particle's cross-sectional area. Large particles therefore scatter light at narrow angles with high intensity, whereas small particles scatter at wider angles but with low intensity (Kippax 2005). The size analysis is based on intensity distribution measurement of coherent laser light scattered by the particles. The form of the scattering pattern is described by the Mia theory and the width of the pattern is dependent on the size. When laser light meets a population of particles, volumetric size distribution can be calculated back from the scattered light distribution (Kongas 2004). Laser diffraction-based size distribution can be assessed in seconds, and a complete analyses can be run in less than 1 minute (Hudson 2008).

4 IMPORTANCE OF PARTICLE SHAPE

Particle shape is of prime importance, in the same way as the size, in order to understand the behavior of powders. Shape parameters are generally used to express some particle attributes, such as elongation, convexity, circularity and roundness. The practical importance of shape analysis is that the shape of particles can assist in identification and analysis, influence the physicochemical properties of a powder, elucidate the process of formation and affect the behavior of the powder in its application.

While huge amounts of comminuted material are produced and used yearly, few works on determining the effect of particle shape on processes exist . The reason of neglection of this area of powder technology is partly owing to the difficulties in measuring the shape of particles (Kursun and Ulusoy 2006). The size of a spherical homogeneous particle is uniquely defined by its diameter. For a cube, the length along one edge is characteristic, and for other regular shapes there are equally appropriate dimensions. With some regular particles, it may be necessary to specify more than one dimension. For example cone, diameter and height, cuboid, length, width and height (Allen 1990). Analyzing the dimension and shape of the particle with laser is a new expansion in particle characterization. Analyzing particles with laser, presents flexible solutions for sample distributions through data analyzing which also gives prospective information about the particle and its behavior during the process.

5 MATERIALS AND METHODOLOGY

Within the scope of this study, two different sand samples from Istanbul-Kemerburgaz Mining Plant, were studied. These sands are used in the production of construction chemicals after they have been exposed to a series of washing and classification operations. It is important to accurately determine both the particle-size distribution of silica sand used in construction chemicals and their shape factor. First of all, the samples were sieved and their size distributions were determined.

The size distribution of the samples was also determined by the Mastersizer 2000 device, which carries out measurement in accordance with laser diffraction method. As the third and the last method, digital image processing is used in order to assess the distribution. Split-Desktop software is used as a digital image processing tool. In addition to particle size distribution measurements by different techniques, the shape characteristics of both sand samples were measured by Morphology G3 device. The composition of sands was firstly identified by XRF analysis, given in Table 1.

Table 1. XRF Analyses results of the samples

Item	Mass (%)	
	Senkoy Sand	Sarikum Sand
SiO_2	88.30	89.4
Al_2O_3	5.50	4.20
TiO_2	0.13	0.12
Fe_2O_3	3.02	2.01
CaO	1.40	1.31
MgO	1.34	1.33

Evaluating the mineralogical composition of two sand samples, it is seen that both sand samples have similar compositions, however especially Senkoy sand contain more clay than Sarikum sand sample. The Mastersizer has a fully optimized optical design that allows particles in the size range 0.1-2000 microns to be characterized effectively (Malvern 2008).

The Hydro MU is designed to create a suspension of particles in water or other liquid media, which can be optimized through the use of ultrasonics and surfactants, where necessary. In this study, particles are fully characterized by using a number of morphological parameters such as circle equivalent diameter, circularity and convexity.

Samples were sieved to remove the very coarse (up to 5mm) particles and agglomerations prior to sprinkling the sands on to a large glass slide for analysis. Analysis was performed by using a Nikon CFI 60 2.5x objective lens and 1/1.8 'Global shutter progressive scan CCD. Every particle in the samples is analyzed avoiding any sub-sampling and a statistically significant number of particles are captured in seconds or minutes.

6 RESULTS AND DISCUSSIONS

6.1 Sieve Analyses

Wet sieve analysis is conducted from 500 μm to 38 μm in laboratory for one hour on Senkoy and Sarikum sand samples. Cumulative % Passing curves of Sarikum and Senkoy samples are given in the Figure 2. According to the results of screen analysis, the average sizes of Senkoy and Sarikum samples are determined as P50=0.278 and P50=0.300, respectively.

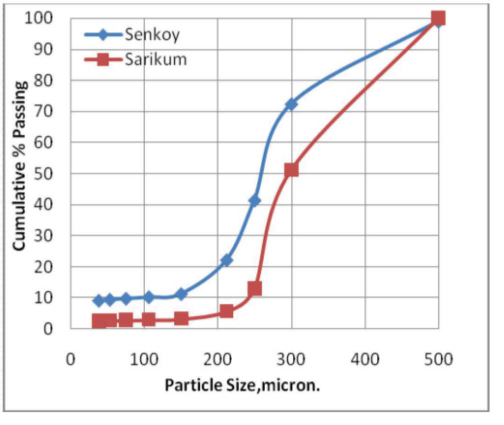

Figure 2. Comparison of sieve analyses results of Sarikum and Senkoy samples.

6.2 Digital Image Processing Analyses

In this research, Split-Desktop digital image analysis software was used as a digital image processing tool for particle size distribution assessment. When using Split software, a special procedure has to be followed to achieve a successful assessment. Briefly the procedure can be described as follows; taking images from material samples, loading the images to program, scaling the images by using the scale reference, delineating the fragments automatically or manually, editing the delineated images, measuring the delineated particles in the images and getting the cumulative size distribution graphics and results. (Ozdemir, 2004)

The images of Senkoy and Sarikum samples were taken through microscope with a digital camera. The images taken by 90^0 were processed in Split-Desktop digital image analysis software and the analysis of particle size distribution was done. Before taking images from the samples a reference scale should be used. Therefore, a ruler was placed next to the sample under the microscope. The particles in the image were delineated manually in order to eliminate possible errors, which can likely to occur by automatic delineation of the image. Although the delineation of the particles manually takes pretty too much time than a normal automatic procedure, it is the most effective method in terms of the results. After the delineation of the particles in the image, some editing is applied. The blanks between the particles are filled with gray color in order not to take these areas into account while program measures the delineated areas. In accordance with the results of the application of Sarikum sample on Split Desktop software, the original image of the sample under microscope is given in Figure 3; the gray-scale image converted by the software is given in Figure 4; manually delineated image is given in Figure 5; and the particle size distribution result of image analysis is given in Figure 6. By the application of digital image processing of Sarikum sand sample, the cumulative size distribution is determined. According to the results, the critical percent passing of the sample particles are found as follows, P20: 0.213 mm, P50:0.295 mm and P80:0.398 mm. In accordance with the results of the application of Senkoy sample on Split Desktop software, the original image of the sample under microscope is given in Figure 7; the gray-scale image converted by the software is given in Figure 8; manually delineated image is given in Figure 9; and the particle size distribution result of image analysis is given in Figure 10. By the application of digital image processing on Senkoy sand sample, the cumulative size distribution is determined. According to the results, the critical percent passing of the sample particles are found as follows, P20: 0.196 mm, P50: 0.273 mm and P80: 0.381 mm.

Figure 3. The original image of Sarikum sample.

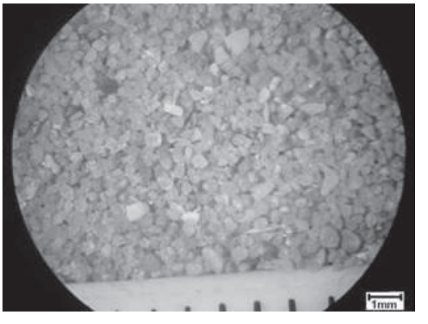

Figure 4.The gray-scale image of Sarikum sample.

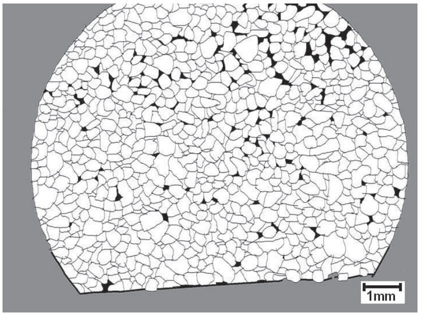

Figure 5. The delineated image of Sarikum sample.

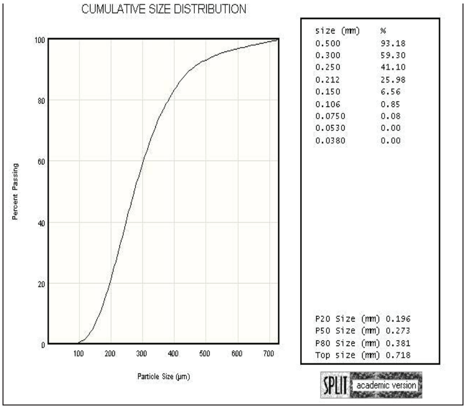

Figure 6. Cumulative particle size distribution of Sarikum.

Figure 7. The original image of Senkoy sample.

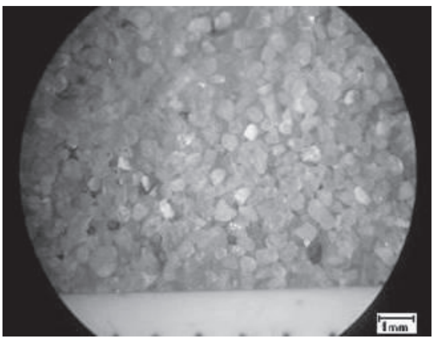

Figure 8.The gray-scale image of Senkoy sample.

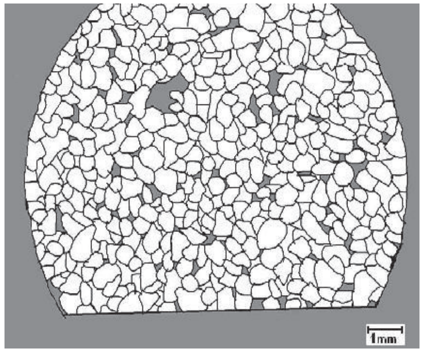

Figure 9. Manually delineated image of the Senkoy sample.

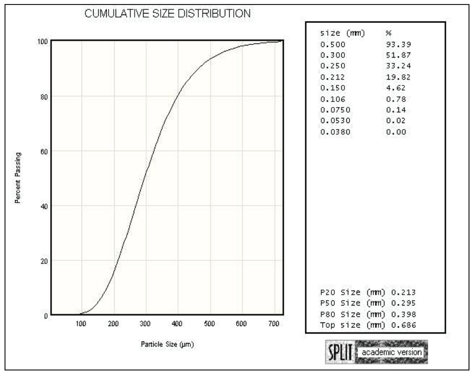

Figure 10. Cumulative particle size distribution of Senkoy sample.

6.3 Laser Diffraction Analyses

Laser Diffraction experiments were conducted by using Mastersizer 2000 and Hydro MU dispersion unit. Using the technique of laser diffraction, it was seen that generally both samples present a well classified coarse particle size distribution, though the presence of fines in each of the sample in reality means that the overall particle size distribution is broad. This dispersion unit with a capacity of 18 ml makes an effective positive ultrasonic impact on dispersion. Experiment results are presented in Figure 11.

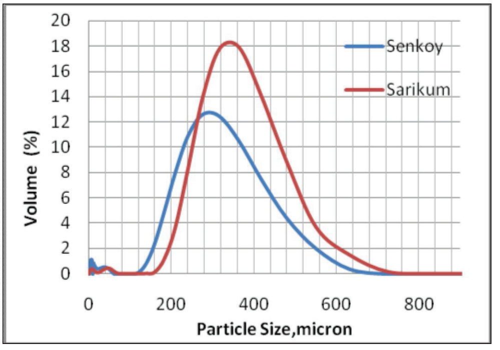

Figure 11. Particle size distributions of two samples obtained by Mastersizer 2000.

It can be seen that the Senkoy sample shows a significant proportion of fines and a smaller mode than does the Sarikum sample. It was necessary to introduce a short period of ultrasonication to the Senkoy sample in order to achieve full dispersion of the fines in this sample. According to the results of laser analysis, the average size of Senkoy and Sarikum samples are P50: 0.275mm and P50: 0.353mm, respectively.

6.4 Particle Shape Characterization of Samples

In this part, the shape analysis of both samples is conducted using the device Morphology G3. The shape differences of Sarikum and Senkoy samples and their effects on measuring particle size are observed. In terms of particle morphology it can be seen that significant differences exist between the two sands. . The comparator tool within the Morphology software readily shows that the differences between the two samples are within the convexity shape parameters and less so in terms of particle size parameters (Fig.12).

An over plot of the volume CE diameter distributions are shown in Figure 10 below and shows that both sands show a similar particle size. However, it needs to be borne in mind that the results below, due to the manual dispersion process, are likely to have fines adhering to the surface of the sand particles – this is especially true for the Senkoy result. It can also be seen that significant differences were seen in terms of the Circularity and HS (high sensitivity) Circularity distributions. This can be seen from the over plot of the HS Circularity distributions in Figure 13.

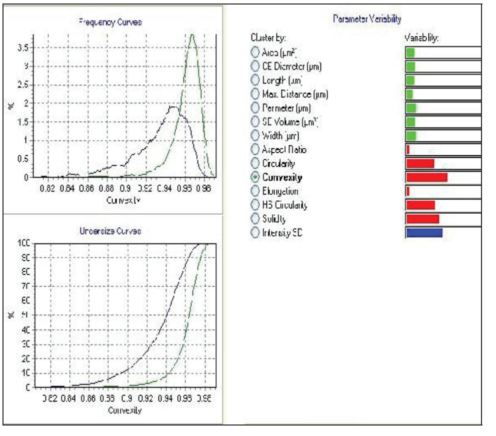

Figure 12. Comparasion of the samples particle shape analysis results.

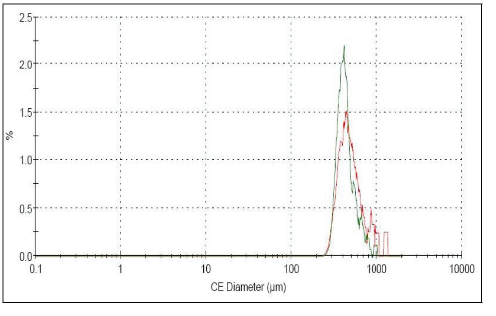

Figure 13. Volume CE diameter distributions of samples.

As seen from both the HS circularity and convexity distributions it is clear that the Senkoy sample has a significantly more irregular perimeters than does the Sarikum sand. This sample also shows the greatest level of fines and hence some of the perimeter irregularity could be attributable to fines adhering to the surface of the large particles, though it is thought that this is not totally responsible for more complex particle perimeter.

The Senkoy sample may also be “softer” and more prone to breakage – owing to the increased level of fines and more complex perimeter than the Sarikum sand.

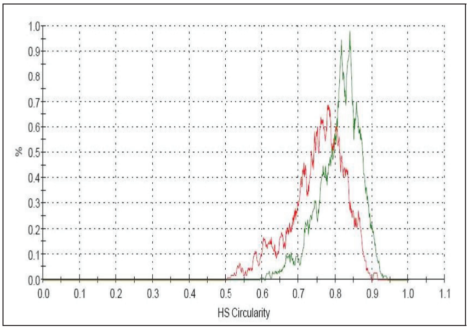

Figure 14. HS Circularity of the samples.

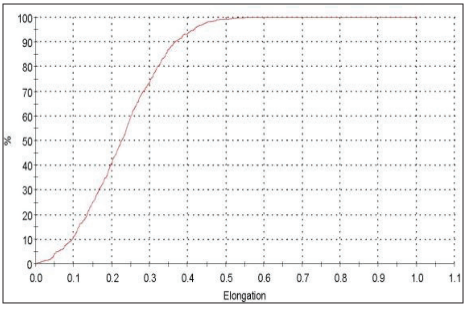

Figure 15. Elongation of Senköy samples.

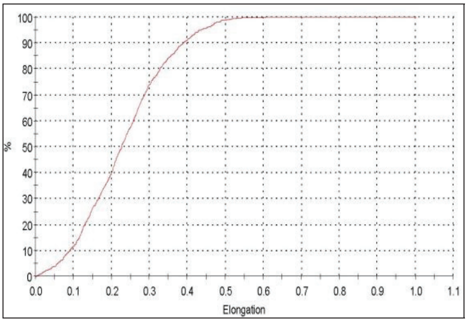

Figure 16. Elongation of Sarıkum samples.

CONCLUSIONS

The following major conclusions can be drawn from this study: It is expectable that size measurement analyses conducted with different techniques on the same sample yields similar values. Comparison of sieve analysis, digital image analysis and laser diffraction analysis P50 values are given in Table 2.

Table 2. Comparison of sieve analysis, digital image analysis and laser diffraction analysis of d50 values.

Method	Senkoy Sample P50 mm	Sarikum Sample P50 mm
Laser Diffraction	0.275	0.353
Digital Image Processing	0.273	0.295
Sieving	0.278	0.300

According to the results obtained through sieve analysis, it was observed that the sieve range through which 80 percent of the sample Sarikum passed was P80: 0.401mm; that its average particle-size was P50: 0.300 mm and that the sieve range through which 20 percent of it passed was P20: 0.214 mm; that the sieve aperture through which 80 percent of the sample Senkoy passed was P80: 0.375 mm; that its average particle-size was P50: 0.278 mm and that the sieve aperture through which 20 percent of it passed was P20: 0.201mm. According to the results of laser diffraction measurements, on the other hand, the sizes of the samples are P50: 0.275 mm and P50: 0.353 mm, respectively. In accordance with the results of digital image processing measurement, the size of the sample Senkoy is found to be P50: 0.273 mm and P50: 0.295 mm for Sarikum sample. According to the particle shape analysis results, as seen from both the HS circularity and Convexity distributions, it is clear that the Senkoy sample has a significantly more irregular perimeter than does the Sarikum sand. This sample also shows that the greatest level of fines and hence some of the perimeter irregularity could be attributable to fines adhering to the surface of the large particles, though it is thought that this is not wholly responsible for the more complex particle perimeter. The Senkoy sample may also be "softer" and more prone to breakage – hence the increased level of fines and more complex perimeter than the Sarikum sand. The convexity value of Sarikum sample is approximately 0.97, the circularity value is 0.85. The convexity value of Senkoy sample is approximately 0.94, the circularity value is 0.78. As circularity and convexity values are close to value 1, particle size measurements with different techniques have given similar results. According to the results of this study that the particle size measurements were compared, with using different techniques, no significant difference was found among measurement results. However laser diffraction and digital image processing make on-line measuring of particle size possible. Besides, it completes the size distribution in a much shorter time than screen analysis. Ignoring the third dimension and shape of the particle are the other disadvantages. The fact that the programs used have yielded such accurate data in a very short time indicates the extent to, which they are advantageous in terms of time and practicability. Also, with the advance of technology and development of this kind of programs, software and devices, one saves money in terms of the costs spent on operations such as crushing, grinding and sieving as well as manpower.

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Robust Systems for Surveillance and Management at Open Pit Mines

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ABSTRACT Aiming at increase in market competitiveness and the reliability of production, the intentions and efforts of the Mining and Energetic Enterprise Bitola are directed toward introducing the contemporary technological solutions, increase of temporal and capacity recovery, improvement of operation readiness of machinery, decrease of production costs, etc. One of the preconditions for achieving such goals is the establishment of the efficient surveillance and management over mining production complex. The paper gives a review of the production system, the goals, basic demands, a concept and the configuration of the surveillance-management system. At the end, the effects expected after the introduction of the surveillance-management system at the lignite Open pit Mine "Suvodol" are given.

1 INTRODUCTION

Open pit Mine (OPM) "Suvodol", is operating within the Mining and Energetic Enterprise (MEE) Bitola is situated in the vicinity of the city of Bitola, Republic of Macedonia. The Mine is supplying the Thermal Power Plant (TPP) Bitola with 6 millions of tons of lignite per year. With the beginning of exploitation of the deeper - lower coal seam, at the "Suvodol" Open pit Mine, plans were made to establish a computer surveillance-management system that should provide real time support to the general production, attached and logistic processes.

Production complex of the Open pit Mine is characterized by:

- Dynamic spatial development of mining operations in the synclinal depression in which the coal deposit is formed;
- Very complex conditions of the working environment for exploitation works;
- Water-bearing of the working environment and significant dewatering dependency of mining operations;
- Complete mechanization of production;
- Connection of the production chain from the lignite exploitation to the TPP;
- Autonomous Open pit Mine "Suvodol" within the MEE;
- Lesser degree of the production system;
- Experienced and creative engineering staff.

In these circumstances of increased market competitiveness and production reliability the intents and efforts of the MEE Bitola are directed to the introduction of contemporary technology, increase in temporal and capacity recovery, increase of machinery readiness, decrease in production costs, more rational utilization of the resources available, etc.

One of the preconditions for achieving such aims is the establishment of the efficient surveillance and management over mining production complex.

2 PRODUCTION SYSTEM

Production system of the "Suvodol" Open pit Mine is made of three major groups of the production process functions.

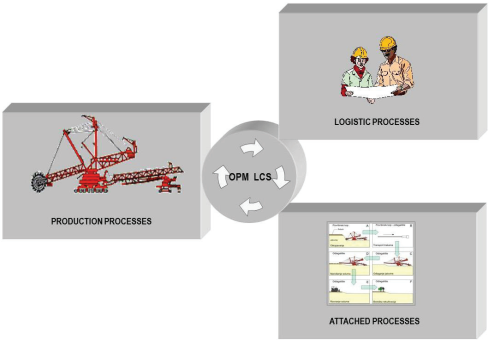

Figure 1. Ground structure of the OPM "Suvodol" production system

- **Production processes,** the functional entity is encompassing immediate production processes: excavation, transport and waste disposal; exploitation, transport and deposition of coal; preparation and auxiliary works.
- **Logistic processes,** the functional entity encompassing the activities and processes supporting the production. This group is made of: protection of the Open pit Mine from the water; meteorological monitoring; open pit mine slopes monitoring; geological support; geodesic support; equipping the equipment and machinery, electric power supply, operational surveillance and planning.
- **Attached processes,** the functional entity encompassing activities and processes: land reclamation and revitalization of coal exploitation demoted areas; work safety; legislation, staff data, norms and regulations; project and technical and technological documentation.

3 ORGANIZATION STRUCTURE OF THE MINE

The "Suvodol" has an organizational structure ensuring and supporting business and production functions within the wider business system such as MEE Bitola and Stock Company Power Plants of Macedonia (PPOM). The architecture of the organizational structure of the "Suvodol" Mine is of a pyramidal type.

The organizational structure of the "Suvodol" Mine consists of six functional hierarchical levels, shown in Table 1.

Table 1. Hierarchical levels of the organizational structure of the "Suvodol" Mine.

Level	Level function
I	Executive production level encompasses: preparation works, excavation, transport and overburden disposal (ECS systems – excavator-conveyor-spreader); exploitation and transport of coal (ECC system – excavator-conveyor-crusher); dewatering and machine maintenance.
II	Executive surveillance and management over production and logistic system functions.
III	Specialization services for production support: mining, geological, geodesic, mechanization, and electric power services, mining and civil operations, warehouse.
IV	Operational management over productive and business functions, Manager, Chief Engineer with the team of experts.
V	Central management of the "Suvodol" Mine – Director with the team.
VI	Supervisory management of the MEE Bitola and PPOM.

The equipment and machinery of the real system at the Open pit mines are belonging to of the stationary of non-stationary group, from the aspect of spatial distribution.

Stationary objects are entities whose location is not to be changed during the course of Mine life and the entities with slowly changing location as the Open pit Mine operations advances. This group is made of: buildings and the Mine complex, dispatch centre (DC), main phone switchboard, coal disposal site, electric power supply network, rim channels for pumped water, observing piezometers, etc. Drainage dewatering wells can also be placed in this group, since these are the objects with longstanding temporal function at single site.

The group of non-stationary entities is made of machinery and equipment: for exploitation, transport and deposition of coal (ECC system); excavation, transport and deposition of overburden (ECS systems), equipment and machinery for preparatory, auxiliary and other logistic activities. The location of these machines is changing as the operations at the Open pit mine are advancing.

The spatial position of machinery and equipment at the open pit mine and the terrain configuration works in favour of the wireless communication between the non-stationary and a portion of the remote stationary measurement-regulation and computer resources. This fact does not diminish the advantages of cable or combined wireless-cable connection at certain locations.

In accordance with the previously stated, the Surveillance-Management System (SMS) must satisfy three basic conditions: firm integration, spatial and functional distribution and modularity.

In the sense of surveillance and management, the entities of the open pit mine are belonging to one of the following two groups:

Remotely surveyed machinery: bucket-wheel excavators, self-propelled transporters, spreaders and machines for intermittent work are surveyed remotely from the DC. The immediate management of the operations of these machines is completed by the operators. The interactive connection between the DC and the operator is established by the feedback and voice communication.

Permanent surveillance over machinery is completed from the DC in real time. This consists of the surveillance over technological functions (operational task, spatial position, output, energy consumption, …), vital electrical and mechanical functions, registering, diagnostics and archiving of stoppages and failures on machines.

Remotely surveyed and managed machinery, Operation of transporters, dewatering system pumps and the electric power supply system are monitored immediately from the DC. The PLC's (programmable logic controllers) on these machines are acquiring, processing and sending the process data and signals (digital and analogue) to the DC. Furthermore, they have data logging function, enable the transfer of feedback signals from the DC to the actuators (regulators, limiter, protection, …) on machines.

Conceptually, the functioning of the SMS should be based on securing the reliable real time information surveillance in order to collect process data necessary for issuing the correct management decisions and recurring actions to the technological, logistic and attached Open pit Mine processes.

The ultimate goal is to maximize financial production effects, high work safety and production reliability.

4 SMS CONFIGURATION

The dominant part in surveillance and management at mines is stilled held by humans. This is effectively bringing down the function of computer resources in information-management systems to efficient and effective determination and provision of relevant data and information in an acceptable form regarding the main production and technological and logistic processes and the events in plants and Mine objects, with the aim of timely and rational, desirably optimal, management decisions by the dispatcher.

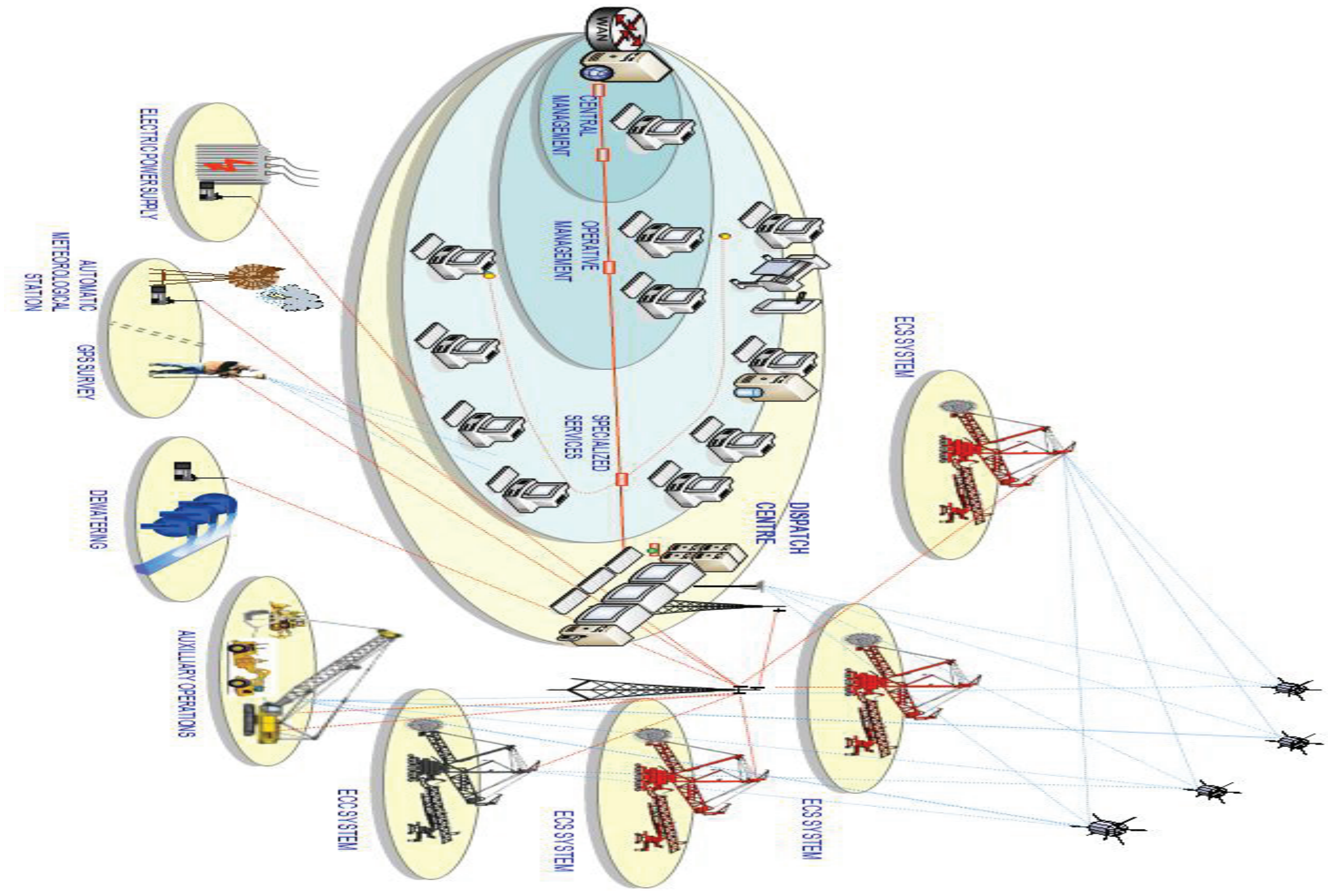

Figure 2. Open pit Mine “Suvodol” SMS Configuration.

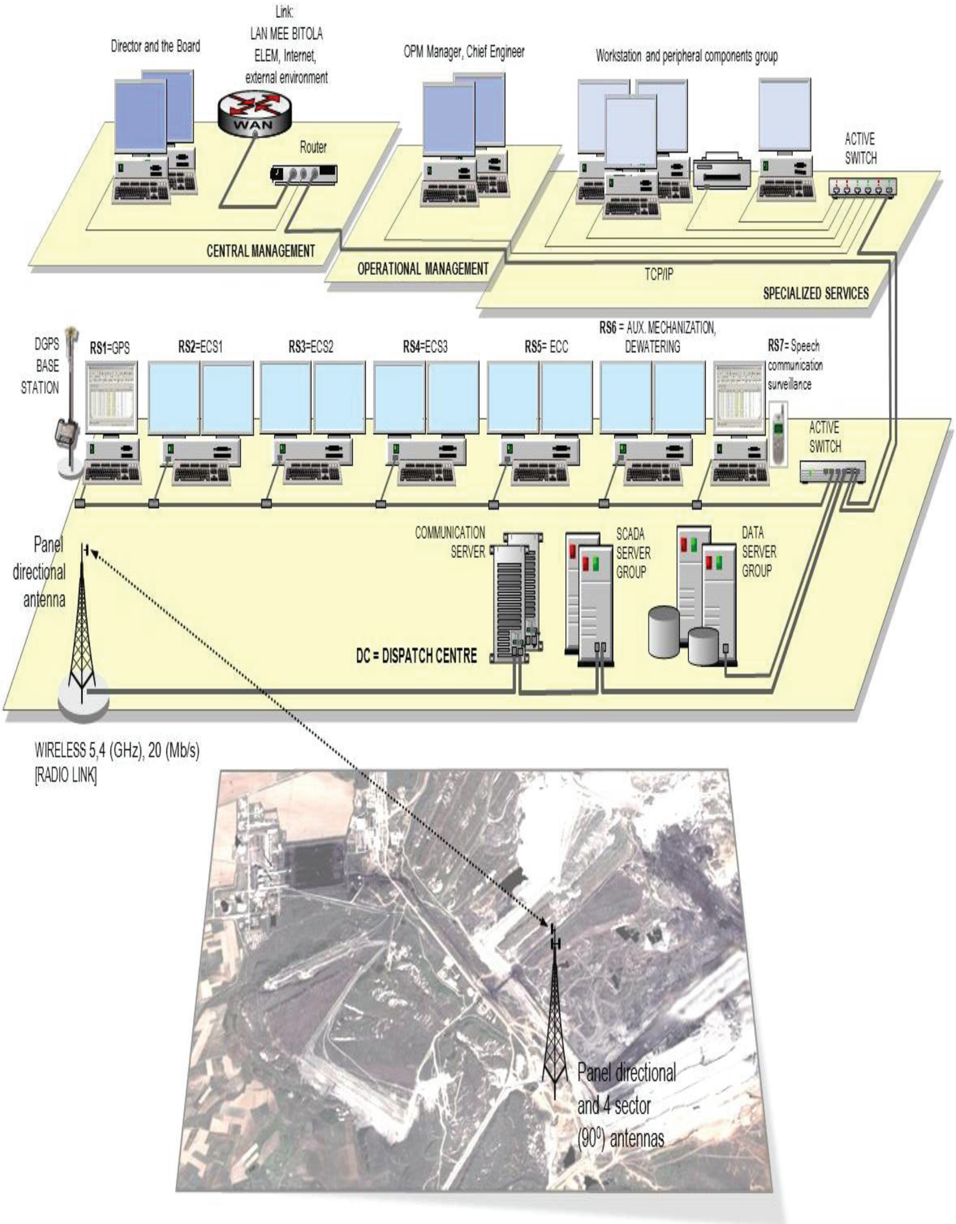

Figure 3. DC configuration and the integration with higher surveillance and management levels.

Furthermore, the function of computer resources is securing the transmission of dispatcher commands to the actuator level and their execution. This general image differs in practice, with an aspiration for complete replacement of human even at command posts.

The configuration of the surveillance - management system of the "Suvodol" Open pit Mine was defined on the basis of the architecture of the mining system and functional tasks of the SMS.

Figure 2 shows the configuration of the surveillance-management system of the Open pit Mine, and Figure 3 the configuration of the Dispatch Centre (DC) and its connection with higher surveillance and management levels.

The suggested solution of SMS configuration is fully taking into accounts the demands and conditions of rationality, functional effectiveness, and reliability, adjustability to the real system, extensibility and standardization. The system is highly redundant, or having little sensitivity to the failures and disturbances of certain segments or system elements. Therefore, the fall out of one of the DC workstations or some of the servers will not disturb the operation of the SMS, since their functions are automatically taken over by other workstation, i.e. server.

The number of workstations and servers in the suggested configuration of the DC is a result of the needs, scope and the functions of the surveillance and management tasks. Workstations are planned for the ECS systems, for the ECC system, dewatering, logistics processes, video surveillance and the voice communication control. The ratio of the workstation price and benefits weigh significantly in favour of benefits and the work comfort, thus effectively removing the dilemma concerning the smaller or larger number of workstations.

5 SMS EFFECTS

The basic purpose of the computer supported SMS, i.e. the robust system of surveillance and management over the mining complex of the "Suvodol" open pit Mine, and the effects achieved by operating such a system are related to:

- Machinery and equipment surveillance (prevention of stoppages, failures, damages and maintenance regime. Borderline and accidents are prevented, thus eliminating or diminishing the failures, breakdowns and damages on equipment, prolong its lifetime, with diminished expenditures for maintenance and repairs);
- Process control (Indication of crisis and accidental operational regimes);
- Process management (Regulating the machinery and equipment operation, and managing the process parameters);
- Equipment functionality and reliability (Optimization of exploitation parameters of working machinery and equipment);
- Efficiency of the technological process (Technological process lead at the required level and operational regime);
- Efficiency in managing decisions (Timely and professionally grounded);
- Economical utilization of the resources available (Managing response to the changes of operational conditions);
- Minimization of the subjective role of the human factor, particularly in the conditions of sudden and critical disturbances of equipment and plant operation;
- Positive intention and motivation of the staff (Training, development, education);
- Work safety (Diminishing the disturbing influence of atmospheric conditions, decreased visibility, snow, rainfall, …), particularly significant in very complex operating conditions.

6 CONCLUSIONS

The resulting key effects are: increase of efficiency, productivity, profitability and safety of the production process, which will influence paying off the investment in the SMS through immediate savings.

Particularly important are the indirect effects (increase in working discipline, responsibility, relation to work and working tasks, …) of the introduction of surveillance and management computer system. The introduction of surveillance and management system is resulting in increased motivation of the employees in work and development, due to advantages provided by the system in completion of operational tasks, handling the contemporary technology and enabling the self-sufficient development of the applications, etc.

System of surveillance and management is having an immediate and an indirect influence on creation of the objective technical conditions and more efficient accumulation of knowledge and experience for further development and widening of the system utilization areas, and thus increasing the positive investment effects.

The modularity of the SMS, computer and other equipment is a ground and a convenience for the phased establishment of the computer supported surveillance-management system at the OPM CLS.

In the environment of financial resources shortage, the establishment of SMS OPM CLS is technically feasible in sections (closed subsystem entities), i.e. in stages, with separate phases and sub phases, not endangering the functionality of system parts already established.

Establishment and introduction of the SMS OPM LCS can be divided in the entities which can be both technically and financially done in several stages, thus relaxing the dynamics of the investments and alleviating the financing.

The establishment and introduction of the SMS OPM LCS has other significant socially useful effects. Aside from the obvious technical and technological positive effects already listed, measurable by financial savings and other indices, there are other, harder to measure, but perceivable immediate and indirect benefits with wider social implications, such as: mastering the highly sophisticated technology, positive environmental effects, development, education of the staff, keeping up with the trends in technology, possible knowledge transfer, etc.

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Economic Trends in Bulgarian Mining Industry Development

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ABSTRACT The mineral resources are the basic means of the stable functioning of the modern societies. The access to mineral resources and their acceptable price is a very important for each country.

In the first half of 2007 year the world economy holding a high rate of growth, which in big stage is result of the high economy growth in the new arising markets.

The world recession has not affected seriously the Bulgarian economy until September 2008 year. The Bulgarian growth rate in the first three quarters on 2008 year is high level, especially compared with the minimum growth in EU, USA and Japan. In the last three months on 2008 year because of spoiling of the international situation and the decreasing of resource prices in the international markets the Bulgarian mining industry reported considerable reduction. Bulgarian mining industry holds out against the unfavorable economic factors, having in mind that 2009 is going to be quite difficult for mining as a whole.

1 INTRODUCTION

In a period of more than ten years the global political leaders had defended the necessity of world economy globalization. However in the year of 2007 appeared the symptoms of upcoming "economic tornado" with its financial and economic parameters.

The initial expectations were that the crisis will bypass Bulgaria, foreseeing the specific national special features and the considerable slowdown in relation to the average European Union social-economical indexes. The reality however appeared to be rather different and the specific data for the negative trends came along.

The Gross Domestic Product (GDP) decreased 3.5%, the Gross Added Value is down with 2.8% as well as the foreign commodity circulation with 21%, according to data of the National Statistical Institute of Bulgaria released in June this year.

These facts confirm the evaluation included in the Report of the World Bank. According to experts the global economical crisis is becoming more sensible in the ten countries of European Union, featuring Bulgaria as the recession is a fact with expectations of 3% drop of the economical activity. This medium impacts each branch of the national economy and companies differently and the results they achieve depend on a series of external and internal factors.

In the mining companies the negative processes and results appeared earlier than in other economic activities because of the sudden price drop of the basic mineral resources.

The reaction was that in a series of mining companies were undertaken measures to decrease the production costs through labour work load optimization. Some companies ceased their mining activities hoping of better times to come.

The largest non-government organization in the field – Bulgarian Chamber of Mining and Geology responded adequate to the critical situation. By different initiatives it proposed to the Bulgarian government a set of possible anti-crisis measures.

The problems that the mining companies were about to face were also analyzed by specialists non-related directly with the mining business (Velev, Radev, 2008). Currently may be prognosticated that the trends of the mining companies economic development will depend on the individual adaptability of each owner and each manager in an unfavorable but not catastrophic macro medium.

2 STEPS OF BULGARIAN MINING INDUSTRY DEVELOPMENT

Mining in Bulgaria has millennium history and the first mining act is from the second half of XIX century. In the Bulgarian mining industry brand new history are clearly outlined three periods.

The first starts immediately after the political regime change in Bulgaria and is related mainly with the administrative decisions related to the mining works undertaken by the Bulgarian government.

The task of the actions is harshly limiting of the mining activities and the reasons are the economic inexpedience of operation mostly of the ore mining enterprises and serious environmental problems. As a result the uranium and lead – zinc and copper underground mining ceased. The extraction in Chelopech mine exploiting the biggest gold deposit in Europe stopped. In the process of liquidation is suspended the activity of big organization and production structures, classic example of which is the biggest lead and zinc underground mining enterprise – the state company "Gorubso".

The second period starts in the second half of 90`s, with the privatization of almost all mining enterprises with the exception of some coal mines. In general the privatization is realized in accordance with the "labour - manager" approach and the formation of small companies with modest financial and resource potential. The results are speaking for themselves for the lead-zinc ore, where from one mining-processing enterprise, extracting and processing more than 4 million tones of ore are crowded in various, including an ore pit as a separate economic object, that extracts ore which is processed by processing plant owned by other mining company.

The other mining enterprises, in this case the bigger ones extracting copper ore in open pits, the privatization is relatively integral and includes the existing production complexes. It may be assumed that this period in which the newest owners face reasonable financial and management difficulties ends in the beginning of 21st century.

The third period is defined as golden for the survived Bulgarian mining enterprises which extract their production in conditions of constantly raising metal and energy carrier prices on the world markets.

These conditions created prerequisites for all mining companies to record record-breaking financial results and to accumulate capital resource high for the Bulgarian standard. For this also contributed the relatively favourable tax regime brought in towards the financial subject – 10% profit tax and low concession payments. The future will show how these auspicious conditions and financial results are used from each mining company but in the moment almost all of them are in an emotional shock created by the specific economical difficulties that the world financial crisis have unlocked.

3 LEGAL FORMS OF FUNCTIONING

The organization of the mining business in Bulgaria is based on two general formulations in the main act of Bulgaria – the Constitution. First this is the approved inviolability of the private property (Art. 17) and the state property of the underground resources (Art. 18).

The connection between these formulations is provided by special acts – Underground resources act and Concessions act. In compliance with them the utilization of the mineral resources in Bulgaria is carried by application of concession regime, as the procedures are realized through regulated schemes and conditions (Velev, 2007). As a result of the privatization carried in the second half of 90`s of XX century the mining activity is performed by private shareholding companies. For a series of reasons their legal form does not have the expected representativeness of the owners.

Prove for this is the official information of the Bulgarian stock exchange according to which only two Bulgarian mining enterprises – Kaolin AD and S & B Industrial Minerals are listed and have a real public statue. In this sense the impression is that the “specific” Bulgarian mentality is displayed also in this business aspect and that exist concrete reserves to raise not only the financial investment culture, but also the financing options.

At the same time the last information of BSE shows existing potential for new emitees and participants in the capital market to appear from the list of the Bulgarian mining companies.

4 ORGANIZATION FORMS OF THE BUSINESS

They are defined by the scale of the so called “players” in the mining business as a financial resource and level of management potential.

The variety of organization forms for the mining business in Bulgaria is pretty reach. The new mining companies were legally isolated as a result of the performed privatization. In this process in some of the cases were broken existing organization links, built on logical sequence of the technological conversions.

In this manner were formed enterprises which were introduced by different production units – mine next to the classical Bulgarian conditions – mine/s and processing plant. In the moment in the public space are posted desires for realization of new privatization strategies with the creation of large scale corporate structures.

They may include production of end product – metallurgy metal for instance or industries utilizing various mineral resources. In the spirit of these tendencies the opinion is that there are signs for attractive investment possibilities.

5 SOCIAL EVALUATION OF MINING ACTIVITIES

It is not a secret that the mining activities are not one side accepted in the public space.

From one side often the mining companies have leading importance for the social – economical condition of the regions and from the other are subject of discussions in relation to factual or eventual negative environmental impact.

Currently the environmental component is very impacted. The problems are related mainly to the mining of the gold bearing ores and their cyanide treatment. Concrete subjects of strong social discussions are the deposits “Chelopech”– owned by Chelopech Mining EAD, “Chala”– “Gorubso Kurdzhali” AD, “Ada tepe” – near the town of Krumovgrad, etc.

At this stage the environmentalists keep the balance and cyanide gold production is not performed in Bulgaria. However the position of the Bulgarian government is hard to be forecasted. The social opinion is very sensitive in relation to the direct participation of the state in the mining activities. More often appear questions about the necessity the government to have its concrete juridical presence by share of property in the specific companies in the form of public – private partnership.

6 THE RISK OF INVESTMENT IN THE BULGARIAN MINING INDUSTRY

On this most important for the potential investors question may be said that the investment in Bulgarian mining companies is accompanied large levels of risk with its different components.

In relation to the political component, our appraisal is that it is extremely low. Bulgaria is a law country, part of the European Union and practically does not exist any possibilities for investment crashes because of the inner political cataclysms.

Bulgarian mining production enterprises were deeply influenced by the world financial crisis and the metals price decrease since 2008 and at the beginning of 2009. Companies develope various projects of up-dating their production programs for risk management. They all intend investment program decrease at least 50 %.

As a general conclusion we might say that Bulgarian mining industry holds out against the unfavorable economic factors, having in mind that 2009 is going to be quite difficult for mining as a whole.

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Information System for Quality Monitoring and Management During Coal Mining

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ABSTRACT The contemporary market relations require the introduction of modern information technologies in the management of mining business.

The control of quality influences considerably the management of coal mining. The specificity of the processes in the mining and manufacturing phases requires the designing and development of such an information system which renders an account of the natural value of the coal as well as the indicators of the market.

The present article focuses on the idea of implementing an information system which controls the basic quality indicators during the coal mining and initial dressing process.

The goal of such an information system is to provide trustworthy information about the specific object of management and through monitoring of quality to locate the deviations from the target goals of the predefined criteria. The monitoring and management quality during coal mining uses predominantly technical methods for collecting the necessary information and statistical methods for processing the results.

The factual data for quality of the managed object can be analyzed through the monitoring carried out with the help of such an information system and on that basis; the factors that cause the disparities with the pre-defined criteria can be defined.

The purpose of the information system is to facilitate the making of quick and effective management decisions which are relevant to the dynamics of the market.

1 INTRODUCTION

The contemporary understanding of quality includes all activities in one organization which ensure its existence, growth and prosperity.

Quality is a complex concept which characterizes the effectiveness of all sides of the activities of a given organization, namely developing strategies, organization and management of the production, marketing strategies, etc.

The quality management is a basic philosophy for the development of every one activity.

The basic factors which define quality as a multidimensional category are the triad – system, process, result (effect) (Figure 1).

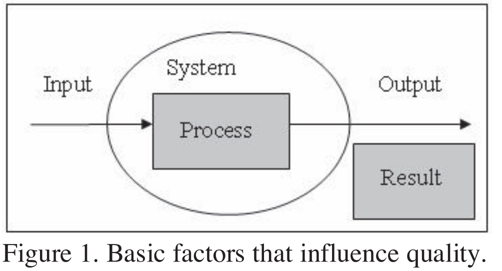

Figure 1. Basic factors that influence quality.

According to research, the process inherits the properties of the system, and the result -

the properties of the process. The better the indicators of the system and the process, the better the result as regards quality. The cycle of quality by rule is a result of the efforts of all managers and employees of a given company during the realization of every phase of the live cycle of every product or service.

The quality management includes the following basic subcategories:

- Quality planning;
- Quality control;
- Quality assurance;
- Quality improvement.

Quality assurance is the basic factor in quality management where the so called control indicators (inspection) are set. This is a procedure for monitoring and control which includes the respective measurements, trials, calibration and conclusions.

Quality management as strategy consists of:

- Concentration upon customers;
- Concentration upon processes;
- Continuous improvements;
- Decisions based on facts.

Quality Management System is actually the organization structure which includes all means, functions and processes, oriented to quality management.

The information system for quality management is a complex of software and technical means, methods for analysis and organizational components, which ensure the creation, processing and analytical usage of the information, which is necessary for the Quality Management System to function.

2 INFORMATION SYSTEM FOR QUALITY MONITORING AND MANAGEMENT DURING COAL MINING

The concentration upon the management of processes is the basic phase in quality management during coal mining because the quality of the final product depends on the quality of the separate processes taking place during its creation and the relationships among them.

The coal is deposited to layers with different thickness and quality of the raw mineral. The coal mining – open pit or underground – is carried out with the help of specialized mechanization and the raw minerals mixed with rock mass, clays and etc.

The main goal in the management of each process is ensuring authentic information about them. In this respect, control plays an important role in quality management. The deviations from the predefined criteria and restrictions for every single process are detected through the control being exercised.

The main indicators for quality are:

- Calorific value (measured in Kcal/kg);
- Ashes content of the coal (measured in %);
- Moisture content in the coal (measured in %);
- Sulphur content in the coal (measured in %);
- Efficiency (measured in t/h);
- Gathered quantity (measured in t).

The control of these indicators collates the factual data from coal mining with the criteria and restrictions, thus the discrepancies with the predefined criteria are defined.

Controlling this data through technical methods and chemical analysis is the primary data for input into the information system. For that purpose, a data base has been created, which is permanently filled in. Filling in this data base is done by different departments of the company.

Figures 2 and 3 show part of the daily filling in of primary information with data about coal mining.

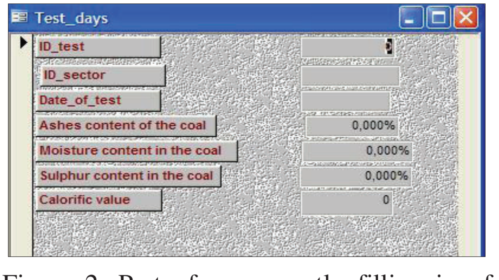

Figure 2. Part of permanently filling in of primary information about coal mining.

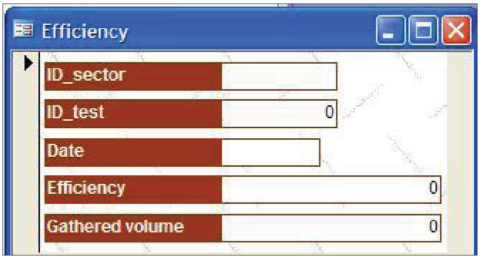

Figure 3. Part of permanently filling in of primary information about coal mining.

Another important indicator for monitoring of quality in coal mining is its massiveness or the so called fractions.

According to massiveness, the coals are separated into 4 or 5 fractions:

1. with massiveness between 0 and 20 mm;
2. with massiveness between 20 and 40 mm;
3. with massiveness between 40 and 100 mm;
4. with massiveness between 100 and 200 mm;
5. with massiveness over 200 mm.

The first fraction (the so called energy solid fuel) has the smallest market price and the fourth fraction is with the biggest market price.

The last fifth fraction are the so called extra dimension pieces which are subject to further destruction – that is to say, additional labour is needed so as for them to be good for sale.

The goal of this control is to receive maximum quantity of the fraction between 100 and 200 mm or of this one fraction which is on demand at the market.

The data about the received fractions from each production sector per day are filled in the table, shown in Figure 4.

Important information for the needs of quality management is also the demand and respectively the sales of particular fractions while searching optimum proportion between production costs and market price.

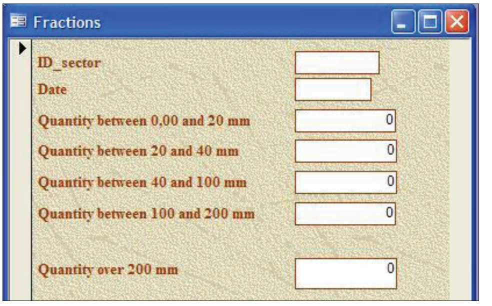

Figure 4. Primary data for quantity of received fractions from sector.

For that purpose, the respective department of the company opportunely fills in the data for the sold quantities from the fraction and their sales price on a daily basis, while at the same time the information about the market prices is observed incessantly. Part of the information being filled in is shown in Figure 5.

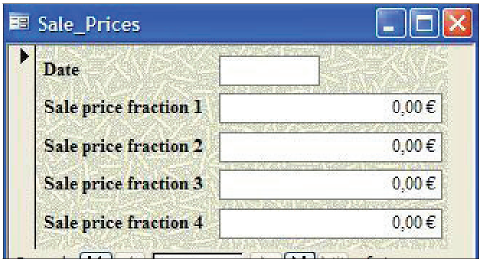

Figure 5. Information observing the market prices.

If the company has the respective measuring or communication equipment, it's possible for the different departments of the company to fill in the data into the information system automatically.

For the purposes of quality management during coal mining it is necessary, except the discrepancies between defined and factual indicators, to be also defined the reasons which cause them.

In order for this to be realized, it is necessary to research the tendencies and laws during coal mining as well as during the realization in the market.

Using statistical and analytical methods, managers are provided with information in the form of reports and charts, which allow them to analyze the situation at every one moment. On that basis, decisions can be made at tactical and operative level of management.

It's possible to create queries by certain criteria using all available information in data base.

The relation with the environment is an important module of the information system. In this case, it is basically the market of the production.

A big part of the coal mining companies have computers with standard software and therefore the information system is realized at two different platforms. One of them is for DB Access so as for the companies to work with it easily without being it necessary for the employees, responsible for filling in the primary information, to be additionally trained.

The other realization of the information system is based on database with Web interface and technology client-server that allows maximum usage of the opportunities of Internet programming.

The figures shown in the paper are a small part of the realization of the information system for DB Access.

3 CONCLUSIONS

The quality management of coal mining is a complex for formal description process, since to a great extent it depends on the natural deposits.

In spite of this, on the basis of the prognoses, which are set as natural deposits and using data from the information system, which shows tendencies and dependences in the long term, the manager can make important decisions directly targeted at the market realization of the production.

An essential advantage of the information system is the fact that it facilitates the making of decisions in successive processes as initial dressing or fragmentation of coals of a concrete fraction depending on the market needs.

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Ore Reserves and Mineral Production in European Union

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ABSTRACT Europe is rich in minerals, both metallic and industrial, and has an ancient tradition of mining and ore processing. Their impact on the geochemistry of the European environment is substantial, and is generally stronger for metallic than for industrial minerals. Some geological areas are richer than average in certain elements, and according to size they are called metallogenic provinces or districts. Also, some individual mineral deposits are so huge as to deserve world-class status.

Within Europe's relatively restricted land area, there exists a remarkable geological variety. It includes ancient cyristalline 'shields', massifs, complex younger fold belts penetrated by igneous intrusions, deep sedimentary basins containing coal and other minerals. As a consequence, Europe has a rich endowment of all the major group of minerals.

Metals production in Europe is insignificant by world production. The EU is a globally important producer of some industrial minerals. The European producers of industrial minerals operate more than 650 mines and quarries and 600 plants throughout Europe . The European industrial minerals sector is present in nearly all of the EU member states from the North of Scandinavia to Mediterranean Cost.

In contrast, the non-metallic substances extracted in the member countries are varied (sand, gravel, stone, calcium carbonate, slate, clays, gypsum, phosphate rock, salt, barite, fluorspar, kaolinite, bentonite, etc.). The building sector in particular reflects the abundance of natural non-metallic resources, while demonstrating the low value per tonne of product obtained and thus its restricted mobility.

In this study, the structure of the European Mineral Industry which has undergone fundamental changes has been discussed.

1 ORE RESERVES IN EU

Within Europe's relatively restricted land area, there exists a remarkable geological variety. It includes ancient cyristalline 'shields', massifs, complex younger fold belts penetrated by igneous intrusions, deep sedimentary basins containing coal and other minerals. As a consequence, Europe has a rich endowment of all the major group of minerals.

In general, the minerals can be classified according to the their source of origin, into two main groups (Anon, 2004):

- Endogenic : created during the crystallisation of magma,
- Exogenic : created through the dynamics of sedimantation.

Endogenic minerals are commonly found in the rocks of Precambrian and in the mountains of Alpine orogeny. Sulphides of lead, zinc, copper are associated with acidic volcanic rocks. They are principally found in Norway, Finland, Sweden, Central France, Germany and the Southern Iberian Peninsula. Iron ore deposits are particularly important in Kiruna region of Sweden.

A second group of endogenic minerals is formed in conjunction with granitic intrusions. It includes uranium, tin, tungsten, gold, antimony, arsenic and mercury. These minerals are found in the South West England, Central France and the Northern Portugal.

A third endogenic mineral group is found in association with basic rocks, including copper sulphides in Cyprus and Norway, and chromite, asbestos and talc found in Cyprus and Greece.

Exogenic minerals include a wide range of naturally occuring sedimantary rocks. They have been deposited in association with sedimentary rocks such as oolitic ironstone, found in central England, Brittany (France), Germany, Luxembourg and Portugal. Copper deposits have also been formed in this way in Germany. Displacement deposits of lead and zinc are common in sedimentary rocks. Thus, Europe is one of the most important parts of the world for these minerals.

Commercial deposits of lead and zinc have been mined in Austria, Finland, France, Germany, Greece, Italy, Poland, Sweden, Spain and Ireland. Other parts of Europe contain substantial deposits of bauxite (aluminium). The mineral occurs and is produced in Greece, Hungary, Italy, the Netherlands and Spain.

A third group of exogenic minerals consists of solid fuels and hydrocarbons. Hard Coal is deposited in Belgium, Czech Republic, United Kingdom, France, Germany, Hungary, Ireland, Poland, Portugal and Spain. There remain large reserves of Brown coal or lignite which are principally found in Austria, Czech Republic, France, Germany, Greece, Hungary, Poland, Slovak Republic, and Spain.

The metallic ore deposits mined in the European Union mainly concentrated in (Anon, 1999):

- The Mediterranean part of the EU (Portugal, Spain, Greece)
- Ireland which has become the leading country in Europe for the production of zinc and lead,
- The two countries that recently joined the European Union (Sweden and Finland), particularly in Scandinavia with the Baltic shield.

In Germany, France and Italy, and in the Benelux countries, nearly all the metallic mines have been shut down or anticipate closure.

In contrast, the non-metallic substances extracted in the member countries are varied (sand, gravel, stone, calcium carbonate, slate, clays, gypsum, phosphate rock, salt, barite, fluorspar, kaolinite, bentonite, etc.). The building sector in particular reflects the abundance of natural non-metallic resources, while demonstrating the low value per tonne of product obtained and thus its restricted mobility.

2 MINERAL PRODUCTION IN EU

The European minerals production and its proportion in total world production are shown in Table 1 (Weber and Zsak, 2002). As can be seen from the table that, metals production in Europe is insignificant by world production. The EU is a globally important producer of some industrial minerals. The European producers of industrial minerals operate more than 650 mines and quarries and 600 plants throughout Europe (www.uepg.org). The European industrial minerals sector is present in nearly all of the EU member states from the North of Scandinavia to Mediterranean Cost.

They offer direct employment to some 40,000 people and process an annual volume of some 100 million tonnes, contributing a value of around € 10 billion to the EU's Gross Domestic Product. If downstream industries such as glass, foundries, ceramics, paper, paint, plastic etc. are included these figures are several order in magnitude greater.

The European Aggregates Association (UEPG) estimates that the average annual production of primary aggregates in only 32 European countries is 3,039 million tonnes (www.uepg.org).

Table 1. Mineral Production in Europe (Weber and Zsak, 2002).

Ores	t (metal)	% Proportion World
Bauxite(Aluminum)	2,467,255	1.8
Chromite	288,343	5.6
Copper ores	715,689	5.2
Iron ore	11,878,949	1.6
Lead ores	271,190	8.8
Nickel ores	22,201	1.9
Zinc ores	843,810	9.5
Barite	398,936	5.8
Bentonite	2,586,585	24.7
Diatomite	128,387	12.0
Feldspar	4,684,413	52.1
Fluorspar	314,381	7.1
Graphite	21,479	3.6
Magnesite	2,649,830	19.0
Perlite	1,014,165	46.1
Salt	44,878,991	21.9
Talc	1,274,770	17.2
Potash	4,936,875	19.9

All European Countries gather annual statistics of national mineral production and trade. The symbol EU25 defines the 25 EU members (as May of 2004). EU associates Norway and Switzerland and the EU candidate countries Bulgaria, Romania, Croatia, Turkey and Macedonia , this group of 32 countries are referred to as EU32 in most of the literatures.

If some selected minerals production figures are studied for EU32 (Turkey is included), European production supplies fairly high proportion of the continent's requirements in contrast to the metallic ores. However, in the cases of several minerals, production is dominated by one country and the majority are still dependent on imports for all but a few of these minerals. The proportion of selected minerals produced by Europe is given in Table 2 (Taylor and Brown, 2009).

As known as, Turkey has got considerable reserves in some certain minerals such as boron, chromium, sephiolite in world scale and feldspar, magnesite, kaolinite and bentonite in EU scale.

Certain European countries are major mine producers of particular metals, for example; Poland (3.7% world total in 2004), Finland and Turkey for chromium (3.2% and 2.4% respectively) and Ireland for zinc (4.7%). The overall European position with regard to mine production of the major non-ferrous metals in 2004 is shown in Table 3 (Taylor and Brown, 2009).

Taking into consideration the large quantities of minerals required by the European society and industry, sustainable development in Europe will depend on actions at the European level, and at national level.

Turkey for chromium (7.0 per cent of the world total in 2007) Norway for titanium (6.8 per cent) and Poland for silver (5.9 per cent) but, the majority of European countries depend chiefly on imports from other continents. Between 2006 and 2007, the overall mine production for metals produced in the EU32 has generally increased slightly. Mine production of chromium has increased by nearly 38.9 per cent at mine production of nickel by 19.0 per cent. However, increases in mine production of other metals are more modest with a 3.8 per cent increased in mine zinc and a 5.6 per cent increase in iron are being typical. The overall European position of mine production of selected metals in 2007 is also shown in Table 3 (Taylor and Brown, 2009).

Table 2. Production of Selected Industrial Minerals as World Percentages (Taylor and Brown, 2009).

Ind.Min.	% World 2004	% World 2007	E31 countries with> 2% of world output in 2004	E32 countries with> 1% of world output in 2007
Feldspar	49.0	61.9	Italy (19%), Turkey, France, Spain, Czech Republic, Poland	Turkey (27%), Italy, Spain, France, Czech Republic
Magnesite	33.7	21.5	Turkey (16.7%), Slovakia, Austria, Spain, Greece	Turkey (8.9%), Slovakia, Austria
Kaoliniteite	25.7	22.1	UK (8.6%), Germany, Czech Republic, Turkey	UK (6%), Germany, Czech Republic, Turkey
Gypsum	23.5	21.0	Spain (10.3%), France	Spain (9.4%), France
Bentonite	23.5	22.1	Greece (5,9%), Turkey, Spain, Italy, Germany	Greece (5.6%), Spain, Italy, Turkey
Talc	15.7	15.9	Fınland (5,8%), France	Fınland (6.2%), France
Potash	14.7	13.7	Germany (11.3%)	Germany (10.0%)
Barites	7.5	-		
Fluorspar	7.2	9.2	Spain (3.1%), France	Spain (2.4%)
Salt		20.3		Germany (5.4%),Netherlands, France, UK
Mica		11.5		France (6.5%), Finland

Table 3. Europe Mine Production of Selected Metals as World Percentages (Taylor and Brown, 2009).

Metal	% World 2004	% World 2007	E31 countries with> 1% of world output in 2004	E32 countries with> 1% of world output in 2007
Silver	10.0	9.4	Poland (7.0%), Sweden	Poland (5.9%), Sweden,Turkey
Zinc	9.4	8.6	Ireland (4.7%), Sweden, Poland	Ireland (3.6%), Sweden, Poland
Titanium	8.7	6.8	Norway (8.7%)	Norway (6.8%)
Lead	7.3	7.5	Ireland (2.1%), Poland, Sweden	Ireland (2.1%),Poland, Sweden
Copper	6.0	5.3	Poland (3.7%)	Poland (2.9%)
Chromium	5.6	9.3	Fınland (3.2%), Turkey	Turkey (7.0%), Finland
Tungsten	4.0	3.5	Austria (2.6%), Portugal	Austria (2.0%), Portugal
Iron	2.2	1.6	Sweden (1.6%)	Sweden (1.2%)
Bauxite	2.2		Greece (1.5%)	
Mercury	2.0	3.2	Finland (2.0%)	Finland (3.2%)
Nickel	1.7	3.2	Greece (1.4%)	Greece (1.3%)
Gold	0.8	0.7	-	-
Manganese	0.6	0.4	-	-

Before 1975, European of both petroleum and natural gas was negligible but Europe produced 10.1 per cent of the world total for natural gas in 2007 and 6.1 per cent of petroleum. These percentages both show reductions on the 2003 figures which were 12.2 per cent and 8.0 per cent, respectively.

In 2007, Norway produced 3.2 per cent of total world output of oil and the United Kingdom produced 1.8 per cent. These two countries are the eleventh and eighteenth largest world producers respectively. Production in Norway has decreased by 18.1 per cent between 2003 and 2007, and in the UK by 27.9 per cent.

Coal output in Europe has been declining steadily for many years as deep mines become increasingly uncompetitive with sources in other continents. In 1981 production from the present EU32 countries was 1351 tonnes. In 2007 production from the same territories was 658 million tonnes, of which half was provided by Germany(3,2 per cent of the world total) and Poland(2.3 per cent).

In 2007, mine production of uranium in the EU32 countries amounted to 519 tonnes U3O8 equivalent, or 1.1 per cent of the world total, chiefly attributable to the Czech Republic. EU32 countries occupy the top four positions for nuclear share in electricity generation in 2007(International Atomic Energy Agency). These countries are France, Lithuania, Slovakia and Belgium with nuclear share ranging from 76.8 per cent to 54.0 per cent.

The overall European position with regard to production of energy minerals in 2007 is shown in Table 4 (Taylor and Brown, 2009).

Table 4. EU32 Production of energy minerals as world percentages (Taylor and Brown, 2009).

Mineral	% World	E32 countries with> 1% of world output in 2007
Coal	10.3	Germany (3.2%), Poland
Natural Gas	10.1	Norway(3.0%),UK, Nether.
Petroleum	6.1	Norway(3.2%),UK
Uranium	1.1	-

3 CONCLUSION

- Mining has played and continues to play a crucial role in the development of industrial and economic activity in Europe as well as in other parts of the world.
- The strong and growing markets for aggregates, metals and minerals and their downstream products in Europe will develop many EU products and Technologies which are leading and serve many wolrd markets.
- Within Europe's relatively restricted land area, there exists a remarkable geological variety.As a consequence Europe has a rich endowment of all major group of minerals.
- European Union is globally important producer of some industrial minerals. However, there are so many several minerals, production is dominated by one country and the majority are stil dependent on imports.
- In the last 5 years, Turkey is the main countryin the production of feldspar ahead of Italy and chromite in Europe.
- Germany and Poland are the main coal producers nearly fifty per cent of total EU coal production.

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Statistical Quality Control of Thicknesses of Plates Cutting from Diamond Circle Saw Block Cutter Machine

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ABSTRACT In this study, whether or not the thicknesses of the plates, which were randomly selected among the blocks cut using block cutting machine with diamond saw, are under control have been examined using range control charts. In order to achieve this, firstly, the thicknesses of the plates have been measured, which have been selected randomly. Range control charts belonging to two blocks have been drawn separately using the measured thicknesses. When the charts have been examined, it has been determined that the process belonging to both blocks are not under control. After that, in order to determine whether or not range values which were used during drawing these control charts represent the same main mass, Kolmogorov-Smirnov test has been conducted in two samples. In view of the test result, it has been determined that range values belonging to two different blocks represent the same main mass. Consequently, although it has been determined with the Kolmogorov-Smirnov test that the cutting settings used by the plant always remain the same, it has been determined with the control charts that these cutting settings have to be rearranged.

1 INTRODUCTION

Statistical process control is used to collect data among the statistical techniques and to classify, analyze, interpret and bring solutions to these techniques. This method bears a feature which ensures the compliance of the production to the designated quality properties and minimizing the production of defective products (Devor et al., 1992). Although statistical process control dates back to old times, the emergence of the quality control in a scientific sense goes back to the first quality control charts developed by W. Shewhart in 1924. Quality control charts is based on determining whether the source of the changes in the process depends on chance or on special reasons and eliminating the changes resulting from the special changes. Its second main purpose is to monitor the process variation limits by way of statistical control and to determine the continuance ability of the process (Aslan, 2001).

Quality control charts, which are commonly used in many fields, have started to be used in mining field. In this scope, some sample applications are as follows: determining the acceptable values of coal by designating the lower limit and upper limit values of the coal quality (Ankara and Bilir, 1995). Sizing the marble tiles which have been selected as quality characteristics via Statistical Quality Control techniques (Saraç and Özdemir, 2003). They conducted a study on the statistical analysis of chromite (Bayat and Arslan, 2004). They shaped the statistical process control analyses for the coals of Garp Lignite Plant Thermal Power Station (Aykul et al., 2005). The losses of the plates cut in circular saw block cutting machines with Shewhart control charts were examined and it was shown that they could be minimized

(Ankara et al., 2006). They determined a protective maintenance plan for ore-dressing facility using Kolmogorov-Smirnov test and average control chart (Yerel et al., 2007). It was found that range control charts are important in determining the change in the surface parallelism of the plates cut with circular saw block cutting machines (Ankara and Yerel, 2008). They showed that it is better to make the range control chart after the irregular data which cause the errors have been detected and removed from the process with the clustering method for the sampling errors in the natural stone plates (Ankara and Yerel, 2009).

In this study, range control charts belonging to two different blocks have been drawn and whether or not the process is under control have been examined. Moreover, whether or not range values which were used during drawing these control charts represent the same main mass has been determined using Kolmogorov-Smirnov test conducted in two samples.

2 STATISTICAL ANALYSIS

2.1 Range Control Chart

Grant and Leavenworth (1985) showed that, the range control chart should be prepared as follows:

- Statistical parameters related to characteristics are computed for each subgroup.
- The averages of ranges of the subgroups are computed as the parameters of a group.
- The three parameters for range control chart are calculated as Central Line (CL_R), Upper Control Limit (UCL_R) and Lower Control Limit (LCL_R).
- The range control chart is drawn.

2.1.1 Calculation of range for subgroups

The range of a subgroup (R_j) is calculated as the subtraction of the minimum measurement value (X_{min}) from the maximum measurement value (X_{max}) in a subgroup (Grant and Leavenworth, 1985; Elevli, 2006). R_j is equal to Eq (1).

$$R_j = X_{max} - X_{min}, \quad j=1, 2,\ldots,m \tag{1}$$

2.1.2. Calculation of parameters of range control chart

After the calculation of the ranges of subgroups, the average of ranges ($\overline{R}$) is found as the parameters of the group, which consist of the statistical parameters of subgroups (Grant and Leavenworth, 1985). $\overline{R}$ is the sum of ranges of subgroups divided by the number of subgroups in the group (m). Three parameters (UCL_R, CL_R and LCL_R) for a range control chart are computed from Eqs. (2), (3) and (4) (Montgomery, 1991; Yerel et al., 2007).

$$UCL_R = D_4\overline{R} \tag{2}$$

$$CL_R = \overline{R} \tag{3}$$

$$LCL_R = D_3\overline{R} \tag{4}$$

Where, D_3 and D_4 : The factors taken from table of factors for the control charts

2.2 The Kolmogorov-Smirnov Test for Two Independent Samples

The Kolmogorov-Smirnov test for two independent samples compares the cumulative probability distributions of two independent samples. If the two independent samples are derived from the same population, the two cumulative probability distributions would be expected to be identical. If there is a significant difference at any point along the two cumulative probability distributions, the two independent samples would be expected to derive from different populations (Sheskin, 2000).

Sheskin (2000) showed that, for determination of relationship between the two independent samples, the Kolmogorov-Smirnov test for two independent samples should be sequentially followed as;

- A cumulative probability distribution should be constructed for each of the two independent samples,
- The test statistic (D_{max}) is defined by the point that presents the greatest vertical distance at any point between the two cumulative probability distributions.

- The null (H_0) and alternative (H_1) hypotheses are derived.

- The test statistic (D_{max}) is compared with critical value (D_α) in the Kolmogorov-Smirnov test for two independent samples table.

- The test result is interpreted and decision making is realized.

The Kolmogorov-Smirnov test for two independent samples is used to determine whether two cumulative probability distributions are different or not. The first cumulative probability distribution for n_1 observations is defined as $F_1(x)$. The second cumulative probability distribution for n_2 observations is defined as $F_2(x)$. The test statistic of two cumulative probability distributions for any one of signs of positive or negative directions is given by (Ankara et. al., 2007; Yerel et. al., 2007);

$$D^+_{max} = \max(F_1(x) - F_2(x))$$

or

$$D^-_{max} = \max(F_2(x) - F_1(x))$$

The test statistic is obtained from the point that presents the greatest vertical distance at any point between two cumulative probability distributions. The test statistic based on the H_0 hypothesis claims that equality of two cumulative probability distributions is true. According to this assumption, the H_0 and H_1 hypotheses are established;

H_0: $F_1(x) = F_2(x)$

H_1: $F_1(x) \neq F_2(x)$

respectively. H_0 hypothesis claims that the two independent samples are almost the same because of equality of two cumulative probability distributions. H_1 hypothesis claims that two independent samples are quite different because of inequality of two cumulative probability distributions.

D_{max} is compared with D_α in the Kolmogorov-Smirnov test for two independent samples table. If D_{max} is less than D_α, H_0 hypothesis is accepted. In addition, D_{max} is equal or greater than D_α, H_0 hypothesis is rejected and H_1 hypothesis is accepted.

3 CASE STUDY

Range control charts, which have been drawn using the thickness measurements belonging to two separate Bilecik beige marble blocks cut using a diamond saw block cutting machine, have been given in Figure 1 and Figure 2. Range control charts, which belong to the 15 plates selected randomly among the plates cut from the first block and the thicknesses of which have been measured, have been given in Figure 1. Range control charts, which belong to the 12 plates selected randomly among the plates cut from the second block and the thicknesses of which have been measured, have been given in Figure 2. When these two charts are examined, it is seen that the process is not under control. That is because plate thickness deviations are high in first sections and show a tendency to decrease through the last sections. In other words, it is seen that there is no change resulting from chance (Değerli, 2009).

Whether or not range values which were calculated from the plates cut from two different blocks represent the same main mass has been certified with Kolmogorov-Smirnov test conducted in two samples. The following hypotheses have been formed in order to determine whether or not range values represent the same main mass.

H_0: Range values of the two blocks are identical.

H_1: Range values of the two blocks are not identical.

In order to determine the significance level;

They have been taken as follows:

$\alpha = 0.05$

and

n_1=12 and n_2=15.

D_α values which are formed from the difference between the frequency distribution of the $(n_1 \neq n_2)$ samples, the unit number of which are different, are found from the critical values table. D_{max} has been calculated as 0.167 for the Kolmogorov-Smirnov test in two independent samples. Kolmogorov-Smirnov has been determined as 0.05 from the critical values table for two independent samples.

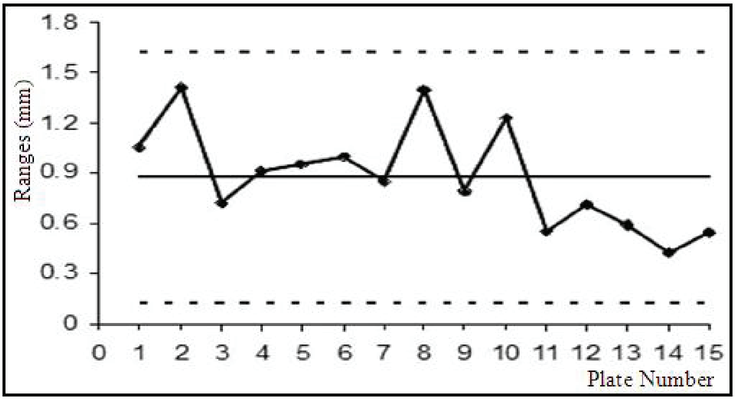

Figure 1. Range control chart for first block.

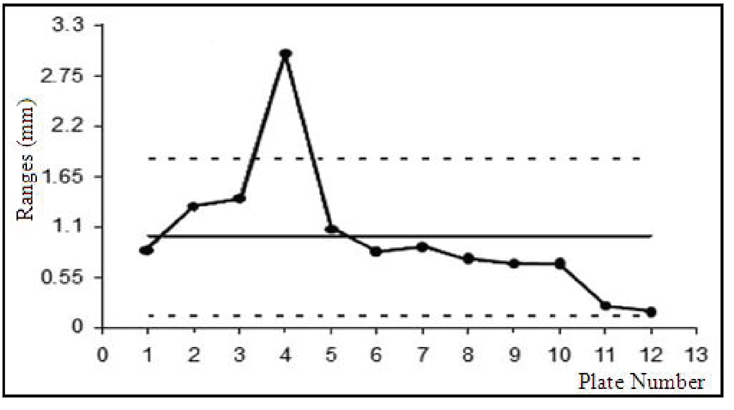

Figure 2. Range control chart for second block.

Due to the fact that the calculated value and table values are $D_{max} = 0.167 < D_{\alpha} = 0.50$; H_1 hypothesis has been rejected and H_0 hypothesis has been accepted.

It has been determined that range values of the plates obtained from the two blocks which were compared as two independent samples are identical to each other. In other words, it is seen that the settings of the block cutting machine used by the plant always remain the same.

4 CONCLUSION

In this study, range control charts of the plates cut from two different blocks using block cutting machine have been drawn. When the control charts have been examined, it has been determined that the process is not under control. Than, in order to support idea that the process is not under control due to the special reason resulting from the cutting settings, the identities of the range values belonging to two different blocks have been determined. In order to determine this, Kolmogorov-Smirnov test has been used in two samples. In view of the test result, it has been concluded that range values belonging these two different blocks represent the same main mass.

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Genleşmiş Kil Agregası Üretim Maliyetinin Hesaplanması
Cost Evaluation of Expanded Clay Production

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ÖZET Genleşmiş kil agregası üretiminde ısıl işlemin olması ve yüksek sıcaklıklarda üretimin gerçekleştirilmesi nedeniyle agrega maliyetlerinin çok yüksek olduğu düşünülmektedir. Bu çalışmada, ülkemiz şartlarında, birim agrega maliyeti yaklaşık bir değer olarak hesaplanmış ve sanıldığı gibi yüksek bir maliyetin olmayacağı ortaya konulmuştur. Yapılan hesaplamaya göre genleşmiş kil agregasının birim maliyeti 18,67 TL/m^3 olarak hesaplanmıştır.

ABSTRACT In production of expanded clay aggregates, costs are high because of heat treatment and high temperature during production. In this study, unit cost are calculated approximately in our country's condition and it's shown that, there will not be a high cost of production as supposed. According to making calculation, unit cost of expanded clay aggregates were found 18.67 TL/m^3.

1 GİRİŞ

Genleşen kil agregasının üretim maliyetinin hesaplanabilmesi için, nasıl bir üretim yapıldığı, üretim kapasitesinin ne kadar olduğu, hangi tür makineler kullanıldığı ve genleştirmede kullanılacak fırının özelliklerinin belirlenmesi gerekmektedir.

Genleşmiş kil agregası üretim maliyeti hesabı, gider türlerine göre ayrı ayrı hesaplanabilmektedir. Böylece ayrıntılı olarak giderler değerlendirilebilecektir. Üretim sırasında oluşan birim maliyet başlıca iki ana bölüme ayrılmaktadır. Bunlar, Sınai Maliyet ve Ticari Maliyettir. Bu maliyetlerde;

- A) Sınai Maliyet
  - a) Hammadde Giderleri
  - b) İşçilik Giderleri
  - c) Elektrik Giderleri
  - d) Yakıt Giderleri
  - e) Genel İdari Giderleri
- B) Ticari Maliyet
  - a) Amortisman
  - b) Finansman
  - c) Pazarlama

olarak kendi içinde ayrılmaktadır.

Genleşen kil agregasının üretiminin nasıl olacağı yanında üretim maliyetinin ne kadar olacağı da önemlidir. Ülkemizde henüz genleşen kil üretimi olmadığı göz önünde tutulduğunda yeni bir ürünün maliyeti daha da önem kazanmaktadır. Bu çalışmada genleşmiş kil agregasının üretimi için yaklaşık bir maliyet hesabı yapılmıştır (Özgüven, 2009).

2 ÜRETİM MALİYETİ HESABI

Genleşmiş kil agregası üretimi için yukarıdaki maliyet türlerine göre detaylı olarak incelenmiştir. Ülkemizde genleşmiş kil agregası üretim tesisi olmadığı için, hesaplamalarda benzer ocaklar ve fabrikalar göz önünde tutulmuştur. Hesaplanan değerler yaklaşık değerler olmakla birlikte üretim maliyeti için fikir vermesi amaçlanmıştır.

2.1 Hammadde Giderleri

Genleşen kilin ocaktan çıkarılarak tesise getirilmesi, stok sahasında depolanması, üretim tesisinde silolara yüklenmesinde kullanılan kepçe, kamyon, vb. makinelerden kaynaklanan giderlerdir. Bu gider miktarı genelde tüm fabrikalar için hemen hemen aynı kabul edilmiştir. Buna göre hammaddeden kaynaklanan birim maliyet 1,40 TL/m^3 olarak alınmıştır.

2.2 İşçilik Giderleri

İşçi ücret ve giderleri, işletme faaliyetlerini yürütmek, üretim ve hizmetleri gerçekleştirmek amacıyla çalıştırılan işçiler için tahakkuk ettirilen (esas işçilik, fazla mesai, üretim primleri, ikramiyeler, yıllık izin ücretleri, Sosyal Sigorta işveren primi, gece primi, hafta tatili ve genel tatil ücretleri, her türlü sosyal yardımlar ve işçilere ait diğer giderler gibi) her türlü tutarları kapsamaktadır (Coşkun, 2007).

Genleşmiş kil agregası üretimi için 2 vardiya ve 16 saat çalışılması düşünülmüştür. Üretim için gerekli personel sayısı yaklaşık 35 kişi olarak hesaplanmıştır. Çalışanların maaşları, giyimleri, vb. tüm maliyeti kişi başı ortalama 1500 TL olarak hesaplanmıştır.

Yıllık işçilik maliyeti ise (35 işçi x 1500 TL/işçi x 12 ay) 630.000 TL olarak hesaplanmaktadır.

Saatlik üretim miktarı 20 ton, günde 16 saat, yılda da 300 gün çalışıldığı düşünüldüğünde yıllık üretim 96.000 ton, diğer bir deyişle de 211.200 m^3 üretim yapılabilmektedir. Buna göre birim işçilik maliyeti (630.000 TL/yıl ÷ 211.200 m^3/yıl) 2,98 TL/m^3 olarak hesaplanmaktadır.

2.3 Elektrik Giderleri

Genleşmiş kil agregası üretimi için gerekli bir tesis için kurulu güç yaklaşık 1007,5 kW olarak hesaplandığında ve saatlik üretimin de 20 ton olduğu düşünüldüğünde ton başına 50,375 kW elektrik enerjisi harcanmaktadır. Diğer bir deyişle m^3 başına 22,9 kW elektrik enerjisi harcanmaktadır.

Tedaş (2009) verilerine göre sanayi için elektrik bedeli, günlük 16 saat çalışmanın puant dışı çalışıldığı göz önüne alınarak hesaplandığında ortalama olarak 10,2 kr/kW'dır. Buna göre üretimdeki elektrik gideri (22,9 kW/m^3 x 0,102 TL/kW) 2,34 TL/m^3 olarak hesaplanmaktadır.

2.4 Yakıt Giderleri

Genleşmiş kil agregası üretimi için hazırlanan tesis planında genleştirme fırını olarak döner fırın seçilmiştir. Döner fırının ısıtılmasında değişik yöntemler kullanılmaktadır. Bunların başlıcaları kömür tozu, doğalgaz, fuel oil ve elektrik enerjisidir.

Ülkemizde kullanılan döner fırınlar özellikle çimento üretiminde kullanılmaktadır. Çimento üretiminde kullanılan döner fırınlarda yakıt olarak kömür tozu kullanılmaktadır. Çimento üretiminde kömür/klinker oranı 0,1252'dir. Çimento üretimi döner fırınlarda ve 1450°C'de olduğu düşünülürse genleşen kil agregası üretimi için gerekli olan 1150°C için daha düşük bir orana ihtiyaç duyulacaktır. Basit orantı ile kömür/kil oranı 0,0993 bulunacaktır.

1 ton kilin fırında pişirilmesi için 0,0993 ton toz kömüre ihtiyaç duyulmaktadır. Toz kömür fiyatı yaklaşık olarak 85 $'dır. Buradan birim ton kil için yakıt gideri 10,55 TL/ton olarak hesaplanmaktadır. Kullanılan kömürün öğütme ve kurutma giderleri de yaklaşık 1,45 TL/ton olarak alındığında toplam gider 12 TL/ton, diğer ifadeyle 5,45 TL/m^3 olarak hesaplanmaktadır.

2.5 Genel Üretim Giderleri

Hangi mamule ya da gider yerine ait olduğu kesin olarak belirlenemeyen, ancak maliyet yerlerine yada maliyet taşıyıcılarına göre bölünebilen üretimle ilgi diğer dolaylı maliyetler Genel Üretim Giderleri olarak adlandırılır. Genel Üretim Giderleri çok çeşitli ve farklı özellikler taşıyan maliyetlerden oluşur. Endirekt isçilik giderleri, Endirekt madde (yardımcı madde-işletme malzemesi), ve diğer –direk madde ve direkt işçilik dışında- imalat giderleri Genel Üretim Gideri olarak adlandırılırlar. Genel Üretim Giderleri, mamul maliyetlerine doğrudan yansıtılamadığından bu giderlerin objektif kriterler yardımıyla mamullere yüklenmesi gerekmektedir (Coşkun, 2007).

İşletmede oluşan genel üretim giderler için 1,00 TL/m^3'lük bir giderin oluşacağı düşünülmektedir.

Ticari maliyetler üretim ile doğrudan ilgili olmayan, ancak birim üretime ilave edilmesi gereken amortisman, finansman ve pazarlama giderleridir.

2.6 Amortisman Giderleri

Üretimde kullanılan makinelerin amortisman giderleri tahmini tutarlar üzerinden üretimin maliyetine yüklenmektedir. Bu çalışmada da amortisman 2,80 TL/m^3 olarak alınmaktadır.

2.7 Finansman Giderleri

Finansman giderleri alınan krediler karşılığında kredi kurumlarına doğrudan doğruya ödenen faiz, komisyon ve benzeri isimlerle ortaya çıkabileceği gibi, işletmenin ilk madde ve malzeme satın alma politikasından özellikle vadeli alımlardan kaynaklanabilmektedir (Anonim, 2008a). Bu çalışmada finansman gideri yaklaşık 1,20 TL/m^3 olarak alınmıştır.

2.8 Pazarlama Giderleri

Pazarlama giderlerini; Pazarlama bölümü ücretleri, Pazarlama araçları giderleri, Stand giderleri, İlan reklam giderleri, Numune giderleri, Satış komisyonları, primleri, İhracat giderleri, Temsil ağırlama giderleri oluşturmaktadır (Anonim, 2008b). Pazarlama gideri olarak yaklaşık 1,50 TL/m^3 olarak alınmıştır.

Genleşmiş kil agregası üretiminde oluşan maliyetler Çizelge 1'de verilmektedir

Çizelge 1. Genleşmiş kil agregası üretim maliyetleri.

GİDER TÜRÜ	ÜRETİM MALİYETİ (TL/m^3)
Hammadde	1,40
İşçilik Gideri	2,98
Elektrik Gideri	2,34
Yakıt Gideri	5,45
Genel İdari Giderleri	1,00
Toplam Sınai Maliyet	**13,17**
Amortisman Giderleri	2,80
Finansman Giderleri	1,20
Pazarlama Giderleri	1,50
Toplam Ticari Maliyet	**5,50**
TOPLAM	**18,67**

Genleşmiş kil agregası üretimi için oluşan toplam sınai maliyet 13,17 TL/m^3, ticari maliyet 5,50 TL/m^3 olarak hesaplanmıştır. Toplam maliyet ise 18,67 TL/m^3 olarak hesaplanmıştır (Özgüven, vd., 2009).

3 SONUÇ

Genleşmiş kil agregası üretimindeki tüm maliyet aşamalarına göre ayrı ayrı hesaplamalar yapılmış ve birim üretim maliyeti 18,67 TL/m^3 olarak hesaplanmıştır.

Yapılan hesaplamalara göre genleşmiş kil agrega üretiminin sanıldığı gibi maliyetli bir üretim olmadığı görülmektedir. Bunun en önemli nedeni de ham peletlerin fırında kalma sürelerinin az olması ve fazla miktarda üretimin kısa sürede yapılabilmesi olarak gösterilebilir.

Genleşmiş kil agregasının ülkemizde yeteri kadar tanınmaması nedeniyle, yurtdışında olduğu kadar geniş alanda kullanımının zaman alacağı tahmin edilmektedir.

Özellikle kullanım yerine yakın üretim tesisi kurulması büyük önem taşımaktadır. Çünkü pomza üretim tesislerinin genelde Orta Anadolu'da bulunması nedeniyle nakliye giderleri bu fiyatlara olumsuz olarak yansımaktadır.

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Application of Input – Output Analysis in Corporate Enterprises of EPIS Thermal Power Sector

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ABSTRACT In modern business conditions, stimulated by intensification of EPIS thermal power sector restructuring process (TPS), an irrefutable requirement is imposed directed towards functional, wholesome, timely, corresponding methodological coverage, investigation and presentation of structure, manner and functioning process of corporate enterprise production system with timely settlement of essential issues with which they are faced in their existence and development.

EPIS thermal power sector is characterised by longstanding lack of investments into technological modernisation, expansion of existing and construction of new production systems as the joint effect of complex and unfavourable geopolitical movements, impact of sanctions, secession processes, restructuring, transitional trends, etc.

This paper is aimed at expanding the application area of input-output analysis of corporate enterprises within EPIS thermal power sector, with realistic consideration of mutually very complex production dependencies and impacts within corporate enterprises.

1 INTRODUCTION

Thermal power sector (TPS) of the Electric Power Industry of Serbia accounts for 66.6% of the overall amount of electric power produced. The main energy-generating product for the production of electric power in thermal power plant is coal-lignite.

Our coal industry is characteristic by longstanding lack of real investments into technological modernisation, expansion of existing and construction of new production systems as the joint effect of complex and unfavourable geopolitical movements, impact of sanctions, secession processes, restructuring, transitional trends, etc.

These features, the strategic importance of coal as the main energy source for this country now and the times to come, followed by far-reaching and consequential quality of decision-making, changeability and significance of open pit mine planning, technical and economic issues related to coal mining, processing and utilisation are features which had a decisive impact on the economic system of this country (Vujic et al., 2005).

The area of the input-output analysis (I-O) in public enterprises and its industrial companies is not sufficiently explored. This small number of examples of the I-O analysis application in mining industry is pointing out the possibilities of the more realistic view on the mutually very complex production dependencies and influences within the TPS and EPIS overall.

The network of I-O production models in TPS EPIS can serve for the purpose of comparison with the production structures of similar surrounding systems (Maksimović, 2009).

2 THE GROUND MODEL

Application of the general intersectoral model should ensure appropriate support to decision making in the integrated process, review the potentials, specifics (limitations), national importance, current position, and the longstanding importance of TPS in the social, industrial, business, and environmental ambience of EPIS and Serbia, and to realize the future development (Vujic et al., 2005).

The intersectoral analysis is having important position in the process of planning, monitoring and management over more and more complex social-economic and industrial development.

Intersectoral model, based on the intersectoral analysis application, can serve as a tool for finding out one of the possible solutions in the area of production, foreign trade, distribution and final production, and in particular for the conduction of the current financial policy (Stanojević, 1998).

The paper pays particular attention to the possibility of realistic review of mutually very complex productive dependencies and influences within the TPS.

3 INPUT-OUTPUT MODEL IN THE THERMAL POWER SECTOR

Thermal power sector consists of industrial societies of the Mining Basin "Kolubara" with the operational units (OU): OPM Field "D" (1), OPM Field "B" (2), OPM "Tamnava-East field" (3), OPM "Tamnava-West field" (4) and "Kolubara-Project" (5), Thermal Power Plant "Nikola Tesla" – TENT "A" (6), TENT "B" (7), TPP "Veliki Crljeni" (8), TPP "Morava" (9), Thermal Power Plant and Open Pit Mines Kostolac with OUs: OPM "Drmno" (10), OPM "Ćirikovac" (11), TPP Kostolac „A" (12) and TPP Kostolac „B" (13).

Ensuring the harmonization between the OUs listed or some other OUs appearing as supplies provider, or the buyers of final products is a very complex task.

There are two main assumptions to be met in application of I-O analysis in the TPS:

- The OUs should be a complete technological entity, where the production process is taking place in a certain manner;
- The size of all production expenditures for any of the OUs is a linear function of the production level of the corresponding OU.

3.1 The Structure of the I-O Table

With TPS I-O table (Tab. 1) as the starting point, we can ascertain the following:

Quadrant I in Table 1 is showing the structure of the reproductive expenditures of the TPS:

$$\sum_{j=1}^{n=13} x_{13,j} \ \text{i} \ \sum_{i=1}^{n=13} x_{i,13} \qquad (1)$$

Quadrant II is showing different components of the final consumption (Y_1, Y_2, ..., Y_{13}):

- General consumption management;
- Formation of the host fixed gross capital funds;
- Changeability of financing;
- Export.

Quadrant III contains elements of external expenditures necessary for overall production of particular OU (material, energy, services, etc.), social investments, salaries, amortization, funds, etc. From the quadrants I and III, technical coefficients can be calculated directly, and they are the base for calculation of Leonti's inverse coefficients and output multipliers. Quadrant IV is showing the primary input for final consumption.

Table 1. Simplified transaction tables.

	Medium expenditures	Final expenditures	Overall Outputs
Medium input Values of the primary calculated input. Input and components GDP Overall input	Quadrant I: Medium production and expenditures Quadrant III: Primary production input	Quadrant II: Final output for production units Quadrant IV: Primary input for the final expenditures	Sum: Medium expenditures and final expenditures

(Source: Clements and Greig (Stillwell, Minnitt, 2000))

4 TRANSACTION (ABSORPTION) MATRICES

Transaction values are shown in 13 rows and columns matrix (Tab. 2), marked with small a_{ij} with (i), (j) ranging from 1 to 13. To all the OUs at the beginning of every row, a subscript (j) was added. The overall input or total price of an OU is presented by a "sum" of all inputs into that OU:

$$X_i = \sum_{j=1}^{n=13} x_{ij} + Y_i \quad \text{where} \tag{2}$$

$\sum_{j=1}^{n=13} x_{i,j}$ ($i \neq j$, i, j = 1,2, … , 13), production quantity of an OU spent in other OUs, i.e.

$\sum_{j=1}^{n=13} x_{i,j}$ ($i = j$, i, j = 1,2, … , 13) production quantity of an OU spent in the very same OU.

Equation of costs for the very same OU is given by the following equation system:

$$X_j = \sum_{j=1}^{n=13} x_{i,j} + F_j \tag{3}$$

where:

$\sum_{i=1}^{n=13} x_{i,j}$ - production (material) costs created in internal realization (4)

F_j, (j=1,2, ... , 13) – all the costs and production elements, except for the expenditures created in internal realization.

Technical coefficient matrix is derived from the absorption matrix, and it shows the input coefficient or direct needs of each OU in relation to the OU output. Technical coefficients are basically production norms, representing the absorption matrix ratio and the sum of outputs of each OU.

In determining the technical coefficient in TPS, the starting point is that expenditures of each input are directly proportional to the production scope for which this input serve, marked with a_{ij}.

$$a_{ij} = x_{ij} / X_j \text{ or } x_{ij} = a_{ij} \cdot X_j \text{ hence:} \tag{5}$$

$$X_i = \sum_{j=1}^{n=13} a_{ij} X_j + F_i \tag{6}$$

It is a linear equation of the balance. A set of these linear equations of the balance and technical coefficients is marked as:

$$X = A_{nn} + F \tag{7}$$

Where;

A_{nn} – Technical coefficient matrix derived from general absorption matrix, while

F – Final consumption of each OU.

Since technical coefficients are dimensionless, they allow for the comparison between different economies, as well as deriving the inverse coefficients.

Realization of coefficient matrix (B=[$b_{i,j}$]) contains realization coefficients ($b_{i,j}$) showing that expenditures OU (i) (X_{ij}) are directly proportional to its overall realization (x_i), or that X_{ij} = bij x_i ,i.e.:

$$b_{ij} = X_{ij} / x_i \tag{8}$$

External expenditures coefficient matrix is related to the quantification of dependencies appearing in the process of general production of OU and TPS, contains direct coefficients of external (add. expenditures) showing direct dependencies of OU from external expenditures, (provider of appropriate material, services and other) (Popović et al., 1977).

Table 2. Intersectoral table of the thermal power sector (Maksimović, 2009).

Receivers		Reproductive consumption of operational units (OU)							Final consumption (external) realization					Overall distributed production (X)
Providers		*1*	*2*	.	*j*	.	*13*	Total	(1) Y	.	(j) Y	.	(n) Y	
Material costs by operational unit (OU)	*1*	X_{11}	X_{12}	.	X_{1j}	.	$X_{1,13}$	$\sum_{j=1}^{n=13} x_{1,j}$	(1) Y_1	.	(j) Y_1	.	(n) Y_1	X_1
	2	X_{21}	X_{22}	.	X_{2j}	.	$X_{2,13}$	$\sum_{j=1}^{n=13} x_{2,j}$	(1) Y_2	.	(j) Y_2	.	(n) Y_2	X_2
	.	.	.	.	.	.	.	.	.	.	.	.	.	.
	i	X_{i1}	X_{i2}	.	x_{ij}	.	$X_{i,13}$	$\sum_{j=1}^{n=13} x_{i,j}$	(1) Y_i	.	(j) Y_i	.	(n) Y_i	X_i
	.	.	.	.	.	.	.	.	.	.	.	.	.	.
	13	$X_{13,1}$	$X_{13,2}$	.	$X_{13,j}$	.	$X_{13,13}$	$\sum_{j=1}^{n=13} x_{13,j}$	(1) Y_n	.	(j) Y_n	.	(n) Y_n	X_n
Overall		$\sum_{i=1}^{n=13} x_{i,j}$	$\sum_{i=1}^{n=13} x_{i,j}$	.	$\sum_{i=1}^{n=13} x_{i,j}$	.		$\sum_{i=1}^{n=13}\sum_{j=1}^{n=13} x_{i,j}$	$\sum_{i=1}^{(1)} Y_i$	.	$\sum_{i=1}^{(j)} Y_i$	.	$\sum_{i=1}^{(n)} Y_i$	$\sum_{i=1}^{n} X_i$
Material expenditures	*M*	M_1	M_2	.	M_j	.	M_{13}	$\sum_{j=1}^{n=13} M_j$						
Electric power	*E*	E_1	E_2	.	E_j	.	E_{13}	$\sum_{j=1}^{n=13} E_j$						
Other	*O*	O_1	O_2	.	O_j	.	O_{13}	$\sum_{j=1}^{n=13} O_j$						
Material expenditures (E+M+O)	*T*	T_1	T_2	.	T_j	.	T_{13}	$\sum_{j=1}^{n=13} T_j$						
Amortization	*A*	A_1	A_2	.	A_j	.	A_{13}	$\sum_{j=1}^{n=13} A_j$						
Contractual obligation	*UO*	UO_1	UO_2	.	UO_j	.	UO_{13}	$\sum_{j=1}^{n=13} UO_j$						
Legal obligation	*ZO*	ZO_1	ZO_2	.	ZO_j	.	ZO_{13}	$\sum_{j=1}^{n=13} ZO_j$						
Gross profit	*BZ*	BZ_1	BZ_2	.	BZ_j	.	BZ_{13}	$\sum_{j=1}^{n=13} BZ_j$						
EFunds	*FO*	FO_1	FO_2	.	FO_j	.	FO_{13}	$\sum_{j=1}^{n=13} FO_j$						
Income UO+ZO+BZ+FO	*D*	D_1	D_2	.	D_j	.	D_{13}	$\sum_{j=1}^{n=13} D_j$						
External expenditures, E+M+O+A+D	*F*	F_1	F_2	.	F_j	.	F_{13}	$\sum_{j=1}^{n=13} F_j$						
Overall production available	*X*	X_1	X_2	.	X_j	.	X_{13}	$\sum_{j=1}^{n=13} X_j$						

5 INVERSE TECHNICAL COEFFICENTS MATRICES

Inverse coefficents are direct and indirect needs (in case of the open model) and direct-indirect needs and induced needs (in case of the closed model) by the monetary unit of final consumption. Opposed to technical coefficients (a_{ij}) showing direct needs by OU production unit for OU*(i)* products, the elements of the inverse matrix of technical coefficients express the manner of distribution of overall effects on the whole production system, created by an increase of the external realization unit of particular OU.

The linear homogenous function $X_{ij} = a_{ij} X_j$ can have the following matrix form:

$$X = AX + Y \tag{9}$$

where:

X – Vector of the overall OU production, of n dimension;

Y – Vector of external OU realization, of n dimension;

A – Square matrix of technical coefficients, of ($n \times n$) dimensions.

The product of matrix A and vector X is a column vector, of n dimension, showing the absolute scope of deliveries OU*(i)* OU*(j)* that corresponds to the given technology and the production scope in the OU*(j)* (i,j=1,2,…, n).

By solving the previous matrix equation by X, production for each of the OUs necessary to settle particular exogenous requests is obtained.

Previous equation can have the following form:

$$X = (1 - A)^{-1} \cdot Y \tag{10}$$

where:

$(I – A)^{-1}$ – Inverse coefficients matrix;

I – Singular matrix.

The inverse coefficient matrix can be presented as:

$$(1 - A)^{-1} = \left(I + A + A^2 + A^3 + ...\right) \tag{11}$$

where matrix A elements shows direct dependency between the OUs within the TPS, while elements of the matrix A^2, A^3,…, shows indirect connections between the OUs.

The inverse coefficient matrix can be presented as:

$$X = BX + F \tag{12}$$

where:

X – Vector of overall production, i.e. realization;

F – Vector of additional external costs;

B – Matrix of realization coefficients.

After solving the equation by X, the following equation is obtained:

$$X = (I - B)^{-1} \cdot F \tag{13}$$

where:

$(I – B)^{-1}$ – Matrix of inverse realization coefficients;

I – Singular matrix.

The previous equation expresses the expenditures of realization of a particular OU X_j as a function of „external expenditures“ F_j of all the OUs within the EPIS TPS.

The elements of the inverse matrix $(I – B)^{-1}$ shows the increase in realization expenditures OU *(j)* if the additional (external) expenditures OU *(i)* increase by a unit.

6 INTERSECTORAL MODELS WITHIN THE EPIS THERMAL POWER SECTOR

The MS models can be applied for different analytical-information purposes in the TPS. In the scope of econometric analysis in the TPS, models for testing the TPS harmonization structure were analyzed, as well as models for testing the degree of integrity of an OU within the TPS, and the rest of the MS models in the TPS.

6.1 Intersectoral Models for Harmonization Structure Tests within the TPS

The analysis of the harmonization structure of a certain production system is possible on the grounds of knowing the technical coefficient matrix, the inverse coefficient matrix (Tab. 3), and the equation system in particular (14).

The equation system 14 should fulfil three main approaches:

1) It can be determined for the given production level of all the OUs (X_1, X_2, …, X_n) which portion of the overall production remains for the fulfilment of the final consumption,

$$Y = X - AX = (I - A) \cdot X \tag{14}$$

This production function is pointing out the conclusion that the final consumption Y of the OU is the function of the overall production of all the OUs, and that this functionality is expressed via technical coefficients a_{ij}.

2) Since each OU has a task of achieving the previously determined level of final consumption, the previous MS model would not be applied in the TPS. The problem lies in determining the necessary level of production for each OU on the grounds of planned scope and the structure of final consumption of the same OUs.

Table 3. Parallel reviews of technical coefficients and inverse matrix coefficients for certain OUs (Maksimović, 2009).

Organizational units X_i, i = 1,2,…, 13	X_1 (j=1)		X_2(j=2)		X_3 (j=13)	
	a_{ij}	$(I\text{-}A)^{-1}$	a_{ij}	$(I\text{-}A)^{-1}$	a_{ij}	$(I\text{-}A)^{-1}$
1	a_{11}		a_{1j}		$a_{1,13}$	
2	a_{21}		a_{2j}		$a_{2,13}$	
.	.		.		.	
13	$a_{13,1}$		$a_{13,j}$		$a_{13,13}$	
Σ						

The system of equations expressing the functional dependency between the production scope OU (X) and the scope and the structure of the final consumption Y of the same OUs is expressed via the inverse coefficient matrix, and it has the following form:

$$X = (I - A) \cdot Y \qquad (15)$$

3) When the quantities of the overall production of certain OUs and the final consumption of the other OUs are known, the determination of the remained quantities' of the overall production and the final production is achieved by the combination of the equations (13) and (14).

The listed approaches to the solution of the equation system (14) are the grounds for all the MS models applied for different balancing and analysis of the production flows within the TPS.

6.2 Testing the Degree of Integration of the OU in the TPS

The application of MS model in the TPS enables the realization of a whole line of information relevant for determining the more rational organization of work, and of the business system in general. Through the analysis of technical coefficients a_{ij}, realization coefficients b_{ij}, external expenditures coefficients and the combination of these coefficients, it can be determined:

- How much is each of the OUs spending the products of other OUs $\left(\sum_{1}^{n} a_{ij}\right)$, and how much of external expenditures $\left(1 - \sum_{1}^{n} a_{ij}\right)$ for a unit of its own production;
- The level of fitting of the each OU into the particular integration unit, as well as the justification of such integration. At the same time, technical coefficients and realization coefficients i.e. arithmetic (mean) value of the sums of all coefficients for the k OU:

$$\frac{1}{2} \cdot \left(\sum_{i=1}^{n} a_{ij} + \sum_{j=1}^{n} b_{ij} \right) \qquad (16)$$

This relation can serve as an integration degree, i.e. level of fitting of particular OU into the TPS.

Table 5 gives the parallel review of the overall production vector and external realization vector for the existing and the planned condition.

If the arithmetic (mean) value is higher, the integrative fit of a particular OU is better. The complete analysis of the overall production vector X and the external realization vector Y, for the existing and the planned status with, for example, increased value of external realization of certain OUs, can affect the efficiency of these multipliers, with which a necessary production of the OU can be determined quickly, in order to ensure the increased realization within the OU [2].

Table 4. Parallel review of $\sum_{i=1}^{n=13} a_{i,j}$, $\sum_{i=1}^{n=13} b_{i,j}$ $\left(\sum_{i=1}^{n=13} a_{i,j} + \sum_{i=1}^{n=13} b_{i,j}\right)/2$.

$\sum_{i=1}^{n=13} a_{i,j}$	$\sum_{i=1}^{n=13} b_{i,j}$	$\left(\sum_{i=1}^{n=13} a_{i,j} + \sum_{i=1}^{n=13} b_{i,j}\right)/2$

Table 5. Parallel review of the overall production vector X and external realization vector Y for the existing and planned condition (Maksimović, 2009).

Existing condition		Planned condition	
External realization value $[Y]$	Overall production value $[X]$	Levelled value of external realization $[Y^*]$	Overall production expected $[X^*]$
Y_1	X_1	Y^*_1	X^*_1
Y_2	X_2	Y^*_2	X^*_2
.	.	.	.
Y_i	X_i	Y^*_i	X^*_i
.	.	.	.
Y_{13}	X_{13}	Y^*_{13}	X^*_{13}
ΣY	ΣX	ΣY^*	ΣX^*
%=100,00	*%=100,00*	*%=100,00±*	*%=100,00±*

7 CONCLUSIONS

Complex economic structure of the TPS in the EPIS has many dynamical variables, each unique for the particular part of the structure. Each of them holds the influence on the economy of mine, thermal power plants, TPS, EPIS as a whole. However, it is impossible to determine those influences by applying the existing accounting systems without the application of the MS analysis.

As already mentioned, from the MS tables provided, originated are: various wide and significant knowledge on direct and returnable production connection of subsystems into a system, two-way dependencies of the system and the environment, i.e. the nature and the intensity of the production system dependencies, deliveries to the environment, way of formation and distribution of subsystem production, i.e. the system and its value structure, the size and the structure of the final consumption, or external realization and the way and degree of its changes influence to the system production, structure of certain expenditures categories and the prices of the production accomplished, etc.

Vast number of countries uses the MS analysis for the assessment of leading the national economy. However, its application in the coal industry is not known well enough. University of Witwaterstand has shown that the MS analysis can be a powerful tool in supporting the management over large corporation (Stillwell and Minitt, 2000).

There are no obstacles to apply certain models of the MS analysis in business societies, TPS, EPIS, operational units, and furthermore in lower organizational forms (divisions, sections).

In relation to previous period, there are no particular limitations in obtaining the technical coefficients, i.e. norms of consumption and expenditures, since the documentation base is improved on the account of computer and appropriate software introduction. If the problem persists in some of the OUs, it can be diminished by the application of the appropriate methods.

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Efficiency Assessment of Cyclic Flow Conveyor System in the Assarel Mine, Bulgaria

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ABSTRACT This paper presents the analysis of the technology for mining of the Assarel ore deposit, presenting the basic structures of the complex mechanization, belt conveying system and their interconnection for two periods – months November and December, respectively for years 2005 and 2007. A brief analysis is made in the selection and delivery of new equipment for mine planning, mining and dispatching of the mining fleet. The types of delays of the cyclic flow conveyor system, loading machines and haulage trucks were studied, compared and analyzed for both periods. It was concluded that the increase of efficiency of the belt conveyor system for mining and waste transportation in the Assarel Mine is dependent on the mine planning and dispatching in the cyclic part of the "loading equipment – haulage truck" system and it is of a particular significance for the operation and economics of the Mine.

1 INTRODUCTION

The operation efficiency enhancement in open pits is achieved through the use of state-of-the-art mining equipment and complexes. They are arranged on certain structures in a single technological system. The efficient operation of this system depends on a multitude of conditions and factors. The relation between the situation on the International Metal Market and the final product price cost is important and determining.

Before the Cyclic Flow Conveyor Technology was commissioned in 2000, the waste from Assarel pit used to be hauled by mine trucks with loading capacity from 40 to 120 tons to an external bulldozer dump. The gradual increasing of haulage distances due to indepth mining in the pit and handling the dump under heavy mining conditions made truck haulage inefficient. Tires, fuel and spare parts consumption significantly grew.

The considerable transportation distances for waste haulage (more than 4,000 m), due to geological and environmental reasons, led to finding of highly efficient transportation systems which had to replace the classical and so far used way of waste haulage with trucks and reduce transportation costs.

The technical and economic studies made by experts from Assarel-Medet JSC showed that it was cost effective and environmentally friendly to haul waste by the use of Rubber Transport Belts (RTB) applied by the Cyclic Flow Conveyor Technology (CFCT).

Based on the decision made by the Board of Directors, an agreement was signed for design, equipment delivery, construction and installation activities and commissioning of a Cyclic Flow Conveyor Technology for waste haulage from the pit to the dump with capacity 3,000 tonnes per hour and "turnkey" performance.

In the middle of 2000 the project was implemented and the facility was commissioned.

During the first year of the Cyclic Flow Conveyor Technology operation in Assarel

Mine it was shown that the planned project indices could not be achieved without a constant relation between the cyclic and conveyor units of the system. Continuous coordination and dispatching of the operation of trucks and shovels is also indispensable. The insufficient number of trucks also had a negative impact upon the system efficiency.

Mine modernization had to be continued in order to improve the operation efficiency. During the period from 2002 to 2003 a feasibility study, was conducted in Assarel-Medet JSC. The study was assigned to an international consulting company who together with experts from Asarel-Medet reviewed the activities in the mine and the concentrator plant and then outlined and suggested improvements which were necessary in order to reduce the cost price of the final product – copper-pyrite concentrate.

Since the end of 2003 an improvement program started.

The major part of the investment program, implemented by the company, was replacement of the mining equipment. The major task when selecting and replacing mining equipment by the end of the deposit life was defining and accepting the objectives to be accomplished:

-Achieving the lowest possible cost price, fuel and lubrication materials consumption, consumption of electric power and spare parts per unit of work under equal technical and technological capacities of the machines;

-Equipment of optimal size corresponding to the nature peculiarities under mining conditions, annual production capacity and life of mine;

-Unification and reliability of equipment by types, makes and units in order to reduce the inventory of spare parts and materials;

-Enduring service maintenance, service facility and personnel training for marinating availability of the equipment;

-Price of the investment.

For the delivery of mining equipment a scrutinized study was made between the most reputable companies producers of mining equipment like CATERPILLAR, LIEBHERR, KOMATSU, Terex - O&K, Sandvic, Ingersoll-Rand, Reedville, Hitachi, BelAz, Reedville , Atlas Copco, etc.

In the process of the study based on own experience and world trends some major criteria were outlined when making a final decision like:

-The machines involved in the complex automation have to match the mining and technical, weather and mining and geological conditions of the pit. They have to be produced in series with corresponding capacity parameters;

-The mining equipment has to be optimal in conformity with the conditions, size and capacity of the mine, the mining technology used and life of mine taking into consideration that the more highly productive equipment has lower operation costs per ton of produced material;

-For the loading units – when a constructed infrastructure and long experience are available electric shovels have significantly lower operation costs which come as a result from the difference between the price of diesel fuel and electric power, lower maintenance costs for the engines and longer life of major components. The limitation of flue gas sources in the mine pit is also an advantage. The disadvantages are lower mobility, longer delivery period, lower usage of this type all over the world;

-For the drilling equipment – diesel engine machines were preferred due to the lack of practice to produce electric machines and the necessity of higher mobility for the Assarel pit;

-The geological complexity of the mined deposit and the requirements for achieving average values of ores based on several indices was assessed and it was found out that the use of shovels with bucket capacity larger than 15-17 m^3 was inefficient. Trucks with loading capacity from 136 to 180 tons correspond to these types of shovels based on technical and operational parameters.

-A comprehensive comparison was made between diesel mechanical and diesel-electrical trucks and advantage to the diesel-electrical trucks was given;

-The importance of mine planning was evaluated and software was purchased and implemented in the Mine Planning Sub-department;

-The necessity of state-of-the-art and reliable systems for mining activity management was also assessed. The purchased mining dispatch system completely met our requirements. Applying the most advanced achievements of GPS technologies, wireless communication systems and computer systems it contributes to the most efficient management of mining equipment by performing automatic dispatching of empty and loaded trucks, real-time management and control in the process of mining activity, including the state of the equipment in the whole mine and quality control of mined ore observing several indices.

Significant attention when preparing the contracts was given to the availability guarantees of the equipment and the related activities on the delivery, life and prices of major components and spare parts, operators, service and maintenance personnel training, instrumentation, etc.

A conclusion can be made that Assarel-Medet JSC confirmed again its endeavoring to maintain its activity at a world level and following the latest scientific and technical achievements in the mining industry.

2 CURRENT STATE IN THE ASSAREL MINE

2.1 General Data about the Mine and Mining Technology

Assarel-Medet JSC Mining and Processing Complex covers an area of 20 thousand decares close to the town of Panagyurishte, to 100 km eastwards from the city of Sofia in the ore region of Panagyurishte at about 1,000 m altitude above sea level.

Assarel-Medet JSC Mining and Processing Complex is a leading producer of copper and pyrite concentrates in Bulgaria and one of the biggest mining companies in Europe.

Assarel Mine is an open pit mine based on the deposit for porphyry copper mining with the same name. The deposit is of altitude-depth type. The accepted mining system is a transportation one with a fan-like carrying out of mining works and hauling waste and leach ore to externally located dumps.

The deposit is characterized by a low grade of copper components (the average value is about 0.40%) with significantly hydrothermally modified rocks.

At the current mining stage the bottom level of the pit is at elevation 735 and according to the staged working project it has to reach a final elevation of - 555 (445m total depth).

Mining activities are carried out at 15 m high benches and a longitudinal haulage road gradient of 10% as the final slope angles are designed to be 28÷34,5°.

Activities like drilling, blasting, loading, haulage and stacking of about 140,000 tonnes of material per day participate in the mining technology. About 36,000 tonnes of copper ore are processed in the concentrator plant daily.

2.2 Structures of the Complex Automation Applied in the Assarel Mine

As it was mentioned before, Assarel deposit is characterized with a very low average metal grade and a high variability of ores by grade, type and sorts. In order to be competitive on the international market constant study and implementation of the most modern achievements in the area of mining automation and process management systems is indispensable.

Based on the classification made by Prof. W.W. Rzhevski (1980) the used complex equipment in Assarel mine can be systemized in the following technological processes:

-A technological preparation unit for rocks digging. The unit carries out drilling and blasting activities with Driltech D-75KS drills, Atlas Copco ROC L8 drills, Volvo mixers for blasting substance and high technology Nonel emulsion blasting substances and means produced by the joint venture company Dyno NitroMed JSC together with the world famous company Dyno-Nobel-ASA, Norway;

-A technological unit for digging and loading of mined material. The unit has at its disposal O&K-RH 120 and LIEBHERR-R 992 diesel hydraulic shovels with $13m^3$ buckets, LIEBHERR -R 994B electrical hydraulic shovels with $17m^3$ buckets and 992D and 994 D CATERPILLAR rubber-tired dozers with bucket capacities – respectively – 10 and $17m^3$;

-A technological unit for primary preparation, continuous waste haulage and stacking. The unit includes a bin, a crushing and screening complex, belt conveyor system and a stacker. The system's capacity is 3,000 tonnes per hour;

-A technological unit for cyclic transportation, operating with diesel-electrical BelAZ 75131 trucks with loading capacity of 130 tonnes;

-A technological unit for construction and auxiliary activities, operating with LIEBHERR crawler bulldozers, CATERPILLAR rubber-tired dozers, CATERPILLAR graders and other auxiliary equipment.

The structure of complex automation for waste haulage in the Assarel Mine is of haulage and stacking type. It is characterized by the use of cyclically used shovels, haulage trucks and conveyor rubber belt transportation with stacking. The interaction between the cyclic and conveyor units is immediate without any intermediary bunker. When one of the connected in parallel lines stops, the complex productivity is respectively reduced. The entire complex stops functioning if one of the connected in series lines stops.

3 CURRENT STATE OF THE ISSUE

Mining industry development at a world scale is characterized with a significant growth of mined volumes of mineral resources and rock types. The constant reduction of metal grade in ore is compensated mainly by increasing the mined volume.

The increase of the mined ore volume also determines the necessity for mining and transportation equipment modernization. However, the implementation of powerful expensive equipment in open pit mines is efficient only when its intensive use is ensured by establishing technological, organizational and technical prerequisites for continuous and faultless operation of the automated system.

A major trend in the mining industry development is the creation of powerful ore mining complexes characterized with a high concentration of operations, using a rational type of transport at minimal haulage distances and low production costs. In order to achieve this aim, it is necessary to increase the mining activity efficiency through improvement and development of the complex automation and intensification of mining and digging processes.

The mining efficiency of open pit mines is determined by the proper selection of technology and equipment. The use of Cyclic Flow Conveyor Technology (CFCT) turned out to be a priority for many mines and its efficiency is proved by the operation of the CFCT system in Assarel Mine.

Research in the area of economic and technical assessment of the indices, characterizing the efficiency of a mining company have been made by many authors and groups – Lukyanov A.N. (1998), Holman P. (1995), Granitzki V.I. (1983), Novozhilov M.G. (1998), Matbarchuk V.A. (1985), Chetverick M.S. (1979), B. Brankova (1976), Gavazov S. (1976), Dzobov I. (2001), Geotechmin (2000), etc.

Efficiency criteria of individual units involved in the mine process from the design stage up to comparing functioning technologies, cyclic, cyclic-conveyor and conveyor, were formulated. Analysis shows that the most used authors' criteria for efficiency assessment are the following:

-Productivity of automation;

-Labor productivity and production cost price;

-Equipment availability and utilization;

-Planned economic effect and relative profit, cost effectiveness and capital investments;

-Electric power consumption and automation depreciation.

In general, all authors accept that the mining efficiency of a certain deposit can be increased only as a result from a significant growth of labor productivity per worker and it depends mainly on the productivity and the use of state-of-the-art mining automation.

4 EFFICIENCY ANALYSIS OF THE USE OF THE CYCLIC FLOW CONVEYOR TECHNOLOGY FOR WASTE HAULAGE FROM ASSAREL MINE

4.1 Current State

The Cyclic Flow Conveyor Technology (CFCT) for waste haulage is applied for the first time in our country and the interest in it by the researchers is fully justified.

The main facilities in the system comply with the particle size and hauled material grade and include:

-A crushing and screening sector consisting of a receiving bin with 300 tonne capacity, apron feeder, vibration screen and a cone crusher with capacity 1,300 tonnes per hour;

-Conveyors No. 1, 2 and 3 with total length of 3,333 m, belt width of 1,200 mm and motion speed 4 m/sec.

-A stacker and tripping car on a crawler with stacking boom length of 50 m;

-Power supply of the complex, provided by Mine Substation through an underground power line (6.3 Kv).

The complex control is fully automated and computerized. It is run by operators, as there is one operator in each command panel located in the screening and crushing sector and the stacker.

The technological circuit is designed for an annual throughput of 13 to 14 mln. tonnes at an hourly throughput of 3,000 tonnes and availability of 52%.

For the last eight years more than 100 millones tones of waste have been hauled with the Cyclic Flow Conveyor Technology in the Assarel Mine and a significant effect has been achieved from transportation activity.

The technological circuit for waste haulage in the Assarel mine is characterized with the following:

- The Mine Planning Sub-department presents a schedule for waste mining by pit levels, blocks and loading units to the mining dispatcher on a daily basis;
- The mining dispatcher manages and controls mining through the mining dispatch system. The latter involves three modules:

-Mine Vision – a positioning system for mine maps application and real time equipment statuses. Using Mine Vision one can see the mine map in the Real-Time Window as well as the position of all units functioning in the mine, the loading locations, beacons, roads, dumping locations and blast fields.

-Fleet Control – a software application for truck [Scheme Shovel – Dumping Location]. Under normal working conditions of the equipment, the application allows automatic dispatching.

The interface between the dispatch system and truck onboard computer provides information about the loading of the trucks, tire pressure, engine oil temperature and pressure. Using this feature of the application, the mining dispatcher manages and controls the mining processes. He observes the equipment state, their status, reasons for delays, truck loading, tonne kilometers, movement routes and dumping location.

-DB Home – a software application for reports and references like Configuration Editor, Report or Reference (in tables and graphs) and blocks import. The import blocks are files in ‘csv’ format which are processed in advance and contain surveyor and geological information. The system is able to generate reports in the form of graphs and tables. From the tool window one can obtain a reference about production, throughput and mining equipment delays as well as about travel time analysis of mine road sections.

The receiving bin at the Cyclic Flow Conveyor System is included as a dump location.

- The mining dispatcher has an access to the Computer and Information System for the Cyclic Flow Conveyor Technology through the systems for information service;
- In most cases the servicing of loading machines loading waste on trucks is performed in a closed system as a definite number of trucks is assigned to each loading unit.

4.2 Analysis of Delays and Equipment Utilization by the Complex Automation in Assarel Mine

Two randomly selected periods, the months of November and December respectively in 2005 and 2007, were used to make short analysis of the complex automation operation and it the relation between them. The periods were related to the implementation of the Program for Technical and Technological Improvements which started in 2005 and its first stage was completed in 2007. Summarized data from the mining dispatch system was used for the purposes of analysis. Detailed analysis of the complex automation operation by registering of "bottlenecks" and undertaking initiatives for improvement are subject of a constant discussion.

- Loading units

The major data about delays, utilization and productivity of loading units for the two reviewed periods is presented in Tables 1 and 2.

The references are summarized and they refer to the total volume of mined material in the mine. It was found out that loading units digging waste (generally with bucket capacity of 17 cubic meters) achieve significantly better indices with regards to availability, utilization and productivity. Their monthly throughput varies within 800,000 – 100,000,000 tonnes. The total result is reduced by the necessity to give an average value of mined ore based on several indices and therefore there are at least three loading units with decreased productivity.

Table 1. Downtimes and operation of the loading units.

Period	Calendar time	Technical servicing	Mecha-nical	Electrical breakdowns	Technolo-gical	Availa-bility Time	Delays Total	Time in operation	Availability Coefficient	Utilization coefficient
2005 November	4 680	166	1 242	74	716	3 198	2 198	2482	68.33	77.61
2005 December	4 834	232	1 078	68	664	3 456	2 042	2792	74.49	80.79
Total - 2005	9 514	398	2 320	142	1 380	6 654	4 240	5274	69.93	79.26
2007 November	5 040	228	862	30	723	3 920	1 843	3197	77.78	81.56
2007 December	5 088	151	986	15	692	3 936	1 844	3244	77.35	82.42
Total - 2007	10 128	379	1 848	45	1 415	7 856	3 687	6441	77.57	81.98

The results show that there is an improvement of the indices which impacts the price cost of the mined material.

- Trucks

The renovation and increase of the truck fleet significantly improved the indices for truck utilization, presented in Tables 3 and 4.

The constant increase of the average haulage distance due to going indepth inexerts a negative impact upon the average throughput per truck. The long delays occurring during shift changes was reduced by the requirement "change on place" within the pit.

There was a significant improvement of the truck fleet servicing. Planned replacement of units coordinated with the unit life time stated by its producer was introduced which resulted in reduction of breakdowns.

Table 2. Average productivity of the loading units.

Period	Mined volumes, tonnes				Time in operation (hours)	Productivity of 1 unit	
	Ore	Leach ore	Waste	Mined material		Tonnes per month	Tonnes per hour
2005 November	1 015 000	253 186	1 588 378	2 856 564	2 482	439 471	1 151
2005 December	1 140 000	74 608	1 787 518	3 002 126	2 792	461 866	1 075
Total - 2005	2 155 000	327 794	3 375 895	5 858 689	5 274	450 668	1 111
2007 November	1 084 000	138 810	2 413 676	3 636 486	3 197	519 498	1 137
2007 December	1 112 000	473 739	2 497 715	4 083 454	3 244	583 351	1 259
Total - 2007	2 196 000	612 549	4 911 391	7 719 940	6 441	551 424	1 200

Table 3. Downtimes and operation of the trucks.

Period	Calendar time	Technical servicing	Mechanical	Electrical breakdowns	Technological	Availability Time	Delays Total	Time in operation	Availability Coefficient	Utilization coefficient
2005 November	9 250	731	454	331	1 132	7 772	2 648	6 564	84.02	83.92
2005 December	9 652	490	632	544	1 268	7 986	2 959	6 413	82.74	80.10
Total - 2005	18 902	1 221	1 086	875	2 400	15 758	5 607	12 977	83.37	82.35
2007 November	16 510	559	1 222	369	1 901	14 360	4 299	12 261	86.98	85.38
2007 December	16 862	753	1 057	467	2 318	14 585	4 845	12 767	86.50	87.54

Table 4. Average productivity of trucks.

Period	Mined volumes, tonnes			Transportation distance		Tonne kilometers		Productivity of 1 unit	
	Waste CFCT	Total Waste	Mined material	CFCT	Km / month	Tonnes / month	Mined material	T/ truck	Tonne km/truck
2005 November	982 620	1 588 378	2 856 564	1.241	2.217	1 192 954	6 333 002	204 040	452 357
2005 December	1 384 550	1 787 518	3 002 126	1.172	1.892	1 703 127	5 680 022	214 438	405 716
Total - 2005	2 367 170	3 375 895	5 858 689	1.199	2.050	2 896 081	12 013 024	209 239	429 037
2007 November	1 272 220	2 413 676	3 636 486	1.407	2.642	1 787 624	9 607 596	158 108	417 722
2007 December	1 430 400	2 497 715	4 083 454	1.287	2.569	1 814 823	10 490 393	177 541	456 104
Total -2007	2 702 620	4 911 391	7 719 940	1.326	2.603	3 602 447	20 097 989	167 825	436 913

- Cyclic Flow Conveyor Technology (CFCT);

Improvement of loading units and truck productivity and dispatching reflected to a great extent on the Cyclic Flow Conveyor Technology (CFCT) system.

Data presented in Table 5 and Figures 1 and 2 show that the trend for hourly loading and utilization coefficient of the CFCT is kept.

The improvements made for removing the “bottlenecks” in the facility operation allowed reduction of delays for the performance of a number planned maintenance and technological activities.

The accumulated experience throughout the years for work with the facility also lead to a positive outcome.

Table 5. Trend of the main parameters of CFCT over the past years.

	Period under review, month, year					
Index	XI - 2005	XI - 2007	XII - 2005	XII - 2007	2005	2007
Planned delay, hour	269.55	97.62	76.44	89.77	1779.19	1285.9
Breakdown and emergency delay, hour	92.51	171.70	150.37	158.84	1884.93	1483.8
Total delay, hour	362.06	269.32	226.81	248.61	3664.12	2769.7
Total time in operation, hour	357.94	450.68	517.19	495.39	5095.88	5990.3
Productivity according to scales, tonnes	982 620	1 272 220	1 384 550	1 430 400	2 254 840	3 685 240
Hourly loading, tonnes/hour	2745	2823	2677	2887	2673	2919
Availability, %	49.71	62.59	69.51	66.58	58.17	68.38

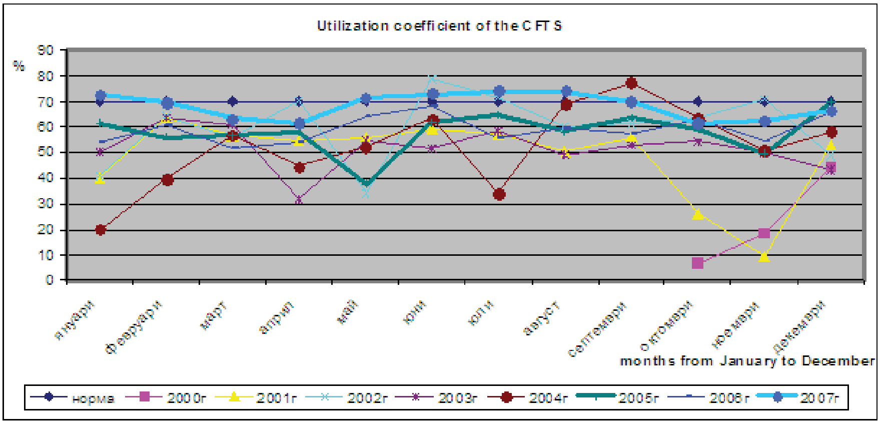

Figure 1. Utilization coefficient of the CFTS.

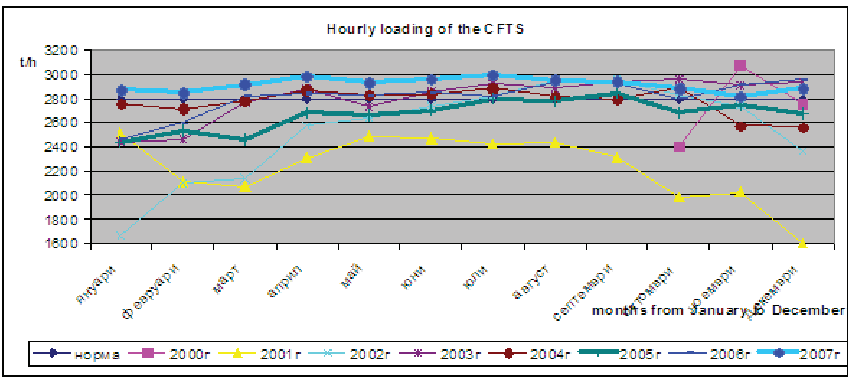

Figure 2. Hourly loading of the CFTS.

5 CONCLUSIONS

The prepared analysis of the efficiency of the Cyclic Flow Conveyor Technology (CFCT) for waste haulage from the Assarel pit enabled us to make the following important conclusions and decisions:

The use of the Cyclic Flow Conveyor Technology (CFCT) for waste haulage and stacking in open pit mines is efficient.

The analysis made for the change of the price cost of hauled and stacked waste with CFCT for the period 2005-2007 shows a decrease of the price cost of 2.5 times compared to the average price cost of the waste transported with CFTS. During the years there is the following change in the price cost: 2005 – 0.276 leva per tonne, 2006 – 0.401 leva per tonne and 2007 – 0.332 leva per tonne. In the costs structure the largest relative share is for the spare parts and other material costs - 53%, followed by electric power - 12%, depreciations - 10%, wages and insurances - 11%, hired services - 7% and other costs - 7%.

In the comparison made between the costs for waste haulage from the Asarel mine with trucks and respectively with the Cyclic Flow Conveyor Technology (CFCT) it was found out that at transportation with CFTS the mining costs are reduced with 30%. It becomes obvious from the data that the waste hauled by trucks has a higher price cost and the difference in time is increased as well.

Waste haulage with trucks at a constant increase of the haulage distance imposes the maintenance of a larger number of trucks in operation which requires additional costs for road maintenance (auxiliary equipment), higher number of personnel (operators and maintenance personnel) and last but not least it exerts an adverse impact upon environment.

In order to optimize the transportation circuit in a long-term plan based on the above stated and following the best world practice, preparation of new ways for ore and waste haulage from the Assarel Mine is needed. This will enable a higher loading of mining equipment and, in general, long-term maintaining of lower operation costs.

Taking into consideration the experience with the operation of the currently existing Cyclic Flow Conveyor Technology (CFCT) for waste haulage various transportation circuits for waste and ore haulage in the Assarel Mine were reviewed. A scenario, involving delivery and construction of a new Cyclic Flow Conveyor Technology (CFCT) for waste haulage with capacity of 5,000 tonnes per hour and reconstruction and relocation of the operating CFCT for ore haulage was discussed, assessed and accepted.

The prepared economic analysis and investment returns evaluation enabled us to find out that the investment for a new CFCT facility with included costs for reconstruction of the currently operating CFCT will pay back for 121% and 159%.

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Evaluation and Actions that can Assure a Stable Management of the Open Pit Mine “ELLATZITE”-Main Elements, Measures of the Success.

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ABSTRACT The mine “ELLATZITE” is one of the biggest active open pit mines on the Balkan peninsula for copper and ore extraction. The annual production rate for 2008 reached 56 million tonnes of material mined. The developments of the mine works in 2009 conforms to the state of the world metal markets and to the possibilities for the development of the mine without discredit its existence until 2022 – the deadline of the concession. The need to maintain a balanced level of extraction imposed the elaboration of a methodology that helps the decisions taking process and the fulfilling of procedures for improving the organization of work. As basics are taken the indicators for the main mining equipment: Availability, Utilization, Production Efficiency, Overall Equipment Effectiveness, that are formed by processing the information, related to the main production equipment, provided by the GPS-based Fleet management system of the mining complex “Sky Links”. The results of the taken decisions are seen in: stable increase of the extraction in 4 consequent years, reduction of the fuel consumption, increase of the productivity of the main mining equipment and reduction of the cost of mining.

1 INTRODUCTION

The copper–porphyry deposit “ELLATZITE” is situated in the south peripheries of the west Balkan structure –metalogene zone near the Srednogorie zone.

The mine “ELLATZITE” is known as one of the biggest functioning open-pit copper mines on the Balkan Peninsula.

The annual production rate for 2008 reached 56 million tonnes of material, from which approximately 12.8 million tonnes of low-grade copper ore.

The project for development of the mine for 2009 conforms to the beginning of the economical crisis in the last three months in 2008 that became deeper. In this sense it stipulates reaching production levels at about 42 million tones of material mined, keeping the same level of ore production. Those production levels aim to ensure the life of the mine for the next 7-14 years.

The methodology created in 2004 for the evaluation of the production process, on the base of which are planned and fulfilled the procedures for improvement of the work is ratified as efficient to be able to maintain a constant, balanced level of extraction.

The procedures that created the possibility to reach to a successful determined production tasks are a complex of measures in the following areas: organization of work, innovation in the work /providing new mining equipment/ and others.

In the basis of the taken decisions are indicators that are as follows: Availability, Utilization, data for them are formed after processing the information for the main production equipment from the GPS-based Fleet management system of the mining complex “Sky Links” in use since

01.11.2003.

The results are visible as increase of the extraction compared to every previous year, reduction to fuel consumption, increase of the productivity of the main mining equipment and the labour productivity.

2 PRINCIPAL ELEMENTS OF THE PRODUCTION

While analyzing the production activity of the complex is taking in mind the contact between the following elements:

1. Organization and state of the production technique and mechanization
2. Organization and state of the maintaining technique and mechanization
3. Organization of the repair activities
4. Plan and control of the production processes
5. Organization and hierarchic structure in production management

Their state is determined from the values of the following indicators: Availability, Utilization, Production Efficiency, and Overall Equipment Effectiveness, which are reviewed in details below.

2.1 Availability, Utilization

2.1.1 Availability

The proportion of time the equipment is in condition to be used according its intended purpose.

$$= \frac{TotalHours - MaintenanceHours - DowntimeHours}{TotalHours - MaintenanceHours} \times 100 \quad \% \tag{1}$$

2.1.2 Utilization

The proportion of the time that the equipment is available that it is used according its intended purpose.

$$= \frac{TotalHours - DowntimeHours - StandbyHours}{TotalHours} \times 100 \quad \% \tag{2}$$

The traditional view of Availability and Utilisation maintains that achieving high equipment Availability is a Maintenance responsibility, while achieving high utilisation is a Production responsibility. By maintaining both high equipment utilisation and high equipment availability, maximum output will be achieved from the equipment.

Consider, however, the situation where a haul truck is operating, but, because of a problem with the engine, it can only haul at 80% of its normal speed. The truck is available, and being utilised, according to our definitions, but clearly maximum output is not being achieved.

Another situation: a shovel trips, causing a 15 minute delay while it is reset.

During this time, the haul trucks queue at the shovel. Once again, those trucks are available, and being utilised, but maximum output is not being utilised.

Clearly, we need a better measure of overall equipment performance.

This is achieved with the introduction of a new indicator, named Production Efficiency.

2.1.3 Production efficiency

The ratio of actual output from a machine (which meets the required quality standards) to its rated output, during the time that it is operating

$$= \frac{ActualProduction}{TotalHours - DowntimeHours - StandbyHours} \div RatedCapacity(units/hour) \times 100 \quad \% \tag{3}$$

Poor reliability, while having some impact on equipment availability, is likely to have a bigger impact on Production Efficiency, due to the inefficiencies associated with starting up and shutting down equipment, and the time and effort that it takes to get the production operation back to a steady state situation. It is fair to say that the costs of poor reliability generally show up in lower production efficiency. This is a measure that is often not given the same emphasis as Availability or Utilisation measures, and in any case is generally considered to be a Production responsibility, with the impact of Maintenance on this figure generally being ignored.

The Overall Equipment Effectiveness is the ‘next step measure’ to assess the effectiveness of the utilization of the asset and can be defined using the following formula:

2.1.4 Overall equipment effectiveness

$$= Availabilty \times Utilization \times ProductionEfficiency \qquad (4)$$

Overall Equipment Effectiveness is closely linked to the accounting measure, Return on Assets, and provides us with an indication of how well we are using our investment in Plant and Equipment.

If Availability, Utilisation and Production Efficiency were all equal to 90%, we might be tempted to think that we are doing a pretty good job, but in fact, the Overall Equipment Effectiveness for this example only equals 73%. This means we are only getting 73% of the total potential output out of this equipment.

Increasing this figure will mean that we can produce more with the same equipment, or potentially, could produce the same amount with less equipment – with an investment of in excess of $1m required for a large haul truck, the savings could be considerable and could be used in other directions.

3 PRODUCTION RESULST OF THE COMPLEX FOR 2008

The idea of introducing the above mentioned indicators in the management of the production process came with the implementation on-site of the GPS-based Fleet management system of the mining complex "Sky Links" since month of November 2003, which gave us all the necessary data for helping the decision making process. We collected enough information to manage to give an impartial assessment of the usage of the main equipment (shovels and trucks) and the distribution of its non-production time.

The previous year, 2008, is characterized as the best year in the 20th history of the mining complex, taking into consideration that the initially discovered and approved for extraction ore reserves were actually mined in the year 2004. We where able to attain those results thanks to:

1. Technical measures: It was approved and financed a new Project for the development of the mine called: "140+20 million tones of ore". That gave us the possibility to create a high productivity working areas with preliminary determined indicators for the exploitation of the equipment.
2. Innovation: During the last two years were bought two shovels HITACHI EX-2500 and the fleet of trucks was basically renewed with: BELAZ 75131 /130 tones/ and HITACHI EH -1700.
3. Organisation: Increase of the productivity rate of the equipment and at the same time reduction of the operational costs, especially for the fleet working in the overburden area.
4. Market: Favourable prices of the copper on the London Stock Exchange for the metals.

As a result of all mentioned above in 2008 was created an organization for stable mining for more than 4 000 000 tones of material per month. Those factors, as well the accomplishment of the working project for 2008, contributed to achieve the main target of the management to assure the mining activities until 2022.

In Table 1 is presented the summary of the achieved mining production in "Ellatzite" for the previous four years 2005, 2006, 2007 and 2008.

Table 1. Mining production.

Year	2005		2006		2007		2008	
	tonnes	%	tonnes	2005-2006	tonnes	2006-2007	tonnes	2007-2008
Ore, kt	11 857	100	12 584	106%	13 005	103%	12 756	98%
Waste, kt	21 864	100	26 606	122%	38 981	147%	43 089	111%
Metal production, kt	45,6	100	44	97%	46,1	105%	44,5	97%
Waste:Ore	1,844		2,114		2,997		3,378	

Shovels – below are presented the relative distribution of the reasons for downtime standby (Fig.1) and the values of the indicators for Availability and Utilization (Fig.2) for 2004, 2005, 2006, 2007, 2008 and the first 5 months from 2009.

Trucks – below are presented the relative distribution of the reasons for downtime standby (Fig.3) and the values of the indicators for Availability and Utilization

(Fig.4) for 2004, 2005, 2006, 2007, 2008 and the first 5 months from 2009.

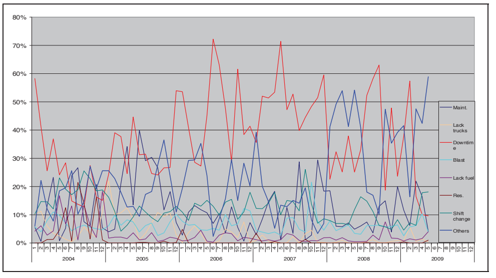

Figure 1. Relative distribution downtime/standby.

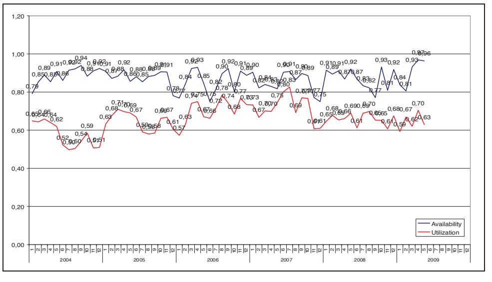

Figure 2. Availability and utilization.

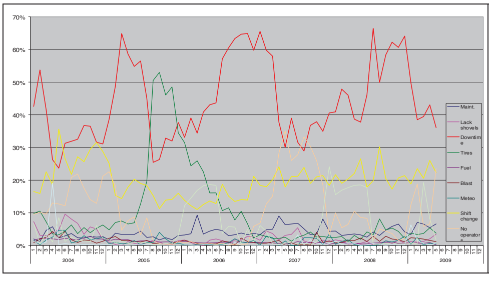

Figure 3. Relative distribution downtime/standby.

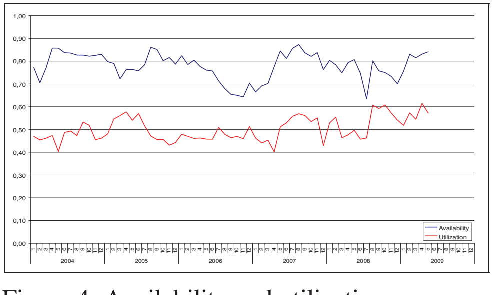

Figure 4. Availability and utilization.

The presented figures give us good initial information about the main reasons for the downtime/standby of the equipment. On them we can see the results from the undertaken measures related to the organization, as well as, introduction of new equipment. Those periods are clearly defined and reflected in the increase of the Availability in the corresponding months. The results of the organizational measures, related to the shift-change in place can be clearly seen in the beginning of the year.

4 PERSPECTIVES

The management of the mining company "Ellatzite" is facing the great challenge to keep maintaining a stable production policy in the dynamically changing environment of the world-spread economical crisis by achieving its production targets of: Availability – 90 % and Utilization – 75 %.

5 CONCLUSION

The presented methodology gives an idea for the assessment of the management of the mine, while trying to introduce the best available techniques, regarding the utilization of the main mining equipment. It gives a possibility to analyze every factor, influencing the observed processes, and to define the cracks, weak points in the management and the organization of the mine.

It ensures a consecutive control over the results from the determined procedures, related to the improvement of the production process.

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Production information from "ELLATZITE –MED" AD.

Developing Sustainability

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ABSTRACT The concept of sustainability has sparked much debate since the issue of The Brundtland Commission Report on environment and development in 1987. Indeed, sustainability has become a widely used concept in development literature. It has also become a buzzword for global corporations in their attempts to maintain profitability in the face of mounting environmental pressures. As it is currently used, sustainable development does not refute growth: it only introduces limits to natural capital use. Yet growth cannot be considered sustainable because we live in a finite global ecosystem which does not grow although it develops (Daly 1990:1). Even so, present efforts toward sustainability revolve around growth, aiming for economic sustainability. Consider addressing the problems of global warming and energy demand with biofuel production, which has already been shown to be neither ecologically benign nor economically viable. This paper proposes that environmental and social wellbeing should get priority over economic growth.

1 INTRODUCTION

The concept of *sustainability* has attracted worldwide attention since the issue of The Brundtland Commission Report on environment and development in 1987. This report defined sustainable development as that which "meets the needs of the present without compromising the ability of future generations to meet their own needs" (WCED 1987:43). The chair of the World Commission on Environment and Development (WCED), Gro Harlem Brundtland, stated that a new era of economic growth was required: "growth that is forceful and at the same time socially and environmentally sustainable" (p. xii). This statement implies that growth and sustainability are not in conflict. Indeed, over time, economic interpretations of sustainable development have taken precedence over ecological and social concerns. As it is commonly used, sustainable development does not refute growth: it only introduces some limits (Daly 1996). Yet experience to date has shown that economic growth and ecological wellbeing are irreconcilable. About twenty years ago, Herman Daly (1990) pointed to important differences between two uses of sustainability: sustainable development and sustainable growth. He rejected the concept of sustainable growth outright as being implicitly contradictory, hence an oxymoron. While growth refers to quantitative increase in physical realm, development refers to qualitative improvement of existing potentials. Growth cannot be considered sustainable because we live in a finite global ecosystem which does not grow although it develops (Daly 1990:1).

The Brundtland definition of sustainable development also fails to clarify the concept of *needs*. The needs of the poor are obviously quite different from those of the rich. While the majority of the world population struggles to meet the most basic human needs (food, water, shelter), a privileged minority lives in conspicuous consumption. Per capita material and energy consumption in the North is beyond the horizon of most people in the South. Then, whose needs are going to be met by sustainable development?

This paper emphasizes that the concept of sufficiency should be given a decisive place in sustainability debates, along with efficiency and effectiveness. Also, Northern countries must initiate genuine efforts to lower their consumption since their economy is responsible for squandering most of the Earth's natural capital.

2 SOME DEFINITIONS

2.1 The Problem

The current economic system considers consumption to be the sole requirement to maximize human wellbeing (Schumacher 1973:42). Ever larger stocks of materials and energy are consumed to meet this end. Yet survey studies have shown that more consumption does not necessarily lead to a lasting sense of wellbeing and happiness. More importantly, this material and energy intensive lifestyle is unsustainable.

Human beings are part of a global ecosystem. Their survival depends on essential services from nature, which include environmental source and sink functions and life support. Ecosystem services also provide the basis for all economic activity. Yet the human economy is consuming natural capital (resources and ecosystem services) at an alarming rate.

The contemporary lifestyle has led to heedless exploitation of natural capital both as source (depletion) and as sink (pollution), causing irreversible environmental degradation around the world. Natural resources are being exploited aggressively to meet ever increasing human wants. Nonrenewable resources are recklessly extracted, without considering that they are irreplaceable. Renewable resources are exploited beyond their regeneration rates while the environment is polluted beyond the assimilative capacities of ecosystems (Daly 1990, 1994). The decrease in biodiversity, for example, is perplexing. The average rate of extinction has reached over 3,000 species per year and is accelerating rapidly (Meyer 2006).

Natural capital consumption has also a broader scope of social, political and economic dimensions. On one level, the human population is growing exponentially, increasing the demand for natural resources. On another level, there exists an expanding inequality in ecological and economic exchange between developed countries and less developed countries. The richest fifth of the world's population reaps the benefits of global economic growth, consuming more and more resources while the poorest fifth carries much of the burden of this plunder. Consumers in the affluent global North externalize the environmental requirements of their consumption by exploiting the natural capital of the South (Hornborg 2007). The North imports the products of polluting extractive industries (mining and agriculture) from the South and disposes of its garbage in the landfills of the South. Environmental, social and cultural fabrics of the South are destroyed during this exchange, expanding world inequalities. Schumacher (1973:45) asserted that exploitation of natural capital almost inevitably leads to violence between men. All wars are resource wars, as the recent war in Iraq and civil wars in Nigeria, Angola and Sierra Leone exemplify.

In sum, the human species has altered the environment for every living being on Earth, including themselves, their descendants and other species. As Giddens (2000:45,61) rightly summarized, today people live after the end of nature and after the end of tradition, that is, with the risks they themselves created. Global environmental risks, such as the stratospheric ozone hole and global warming, threaten all life forms on Earth, and they may lead to irreversible disastrous consequences. Given the magnitude of these problems, there should have been a race to create an environmentally sustainable and socially just global society. Yet, as the following case of biofuels exemplifies, efforts toward sustainability revolve around unsustainable practices.

2.1.1 A Recent unsustainable act

Biofuels have been promoted as a panacea to addressing problems of global warming and the looming energy crisis in transportation. Paradoxically, the production of these fuels may emit more greenhouse gases through land use changes and during conversion processes. Consider, for example, corn ethanol. Corn needs to be grown, harvested, transported, and converted to fuel. All steps require energy inputs – which, at present, come mostly from *nonrenewable* fossil fuels. For this reason, corn ethanol cannot be considered a renewable energy source.

When all energy inputs for corn production and processing are evaluated, about 1.4 gallons of oil equivalents are required to produce 1 gallon of ethanol (Pimentel 2007). Moreover, ethanol has lower energy content than gasoline: a unit volume of ethanol has 2/3 of the energy contained in the same volume of gasoline (DOE report, 2005). In other words, ethanol would deliver 2/3 of the distance traveled compared to that delivered by the same volume of gasoline in the same vehicle.

Besides, as millions of hectares are transformed by multinational corporations to grow biofuel crops, populations around the world face severe food shortages and higher prices. Indigenous peoples suffer the most: they lose their food sovereignty, food security and human dignity. Today the Sumatran rainforests of Indonesia are being cut down, burnt and turned into oil palm plantations. The indigenous Orang Rimba community, which lived sustainably for ages in these forests, has lost the ability to sustain itself. Community members suffer from hunger and malnutrition. They either beg for food or take it from plantations at the risk of being subjected to violence. On September 17, 2007, Indonesian police opened fire on members of the Orang Rimba community because they were collecting palm fruits on an oil palm plantation (Ernsting, 2008).

Even in rich countries, more people are suffering the consequences of biofuel production. In the United States, for example, large scale ethanol production in 2007 increased food prices, bringing low-income individuals to the edge of hunger (Runge and Senauer, 2007).

All is done to endow a wealthy minority with the freedom of an inefficient mobility.

Replacing nonrenewable resources with renewable resources constitutes a right step toward sustainability. Yet sustainability should be attained without forcing ecological limits or increasing social inequities. As it is currently administered, biofuel production is economically, environmentally and socially unsustainable. The production process as a whole embodies inefficient and ineffective resource use.

2.2 Sustainable Development

Efficient and effective resource use is a prerequisite for sustainable development. Efficiency requires obtaining the highest productivity possible per unit of resource while effectiveness requires obtaining optimal utility from the resource used (Ruth 2006:337). To date, the major theme in sustainable development rhetoric has been efficient resource management. "Getting more with less" has become the mantra of most industries. Nevertheless, efficient resource use has not prevented nor decreased environmental deterioration: resource depletion and waste accumulation continue to increase. Examples abound. Fuel efficiency, for example, has reduced the cost of driving and consequently increased both the number of cars and the miles driven per year.

A recent article in *The Economist* (2008) stated: "The green message—use less energy—is not going to solve the problem [of climate change] unless economic growth stops at the same time. If it does not (and it won't), any efficiency saving will soon be eaten up by higher consumption per head." This approach is not new; it is called the Jevons' paradox or rebound effect. The paradox was named after the British economist William Stanley Jevons who maintained, in his 1865 book *The Coal Question*, that energy efficiency improvements in coal-powered engines increased the consumption of coal rather than reducing it.

The Jevons' Paradox is an observation that increased efficiency decreases the cost of resource use and subsequently accelerates its consumption in the long run. In a paper based on a comprehensive review of the evidence for rebound effects, Sorrell (2009:1468) concludes that "while it seems unlikely that all energy efficiency improvements will lead to backfire … rebound effects matter and need to be taken seriously."

Focusing on efficiency alone does not lead to sustainability, unless it goes hand in hand with effective resource use and more importantly with sufficiency. Ineffective resource use epitomizes the present economic system. Allocating vast areas of farmland to raise biofuel crops instead of food crops, watering golf courses instead of farmland, burning petroleum in race cars instead of saving it for future generations are some examples of ineffective resource use. All signify waste.

The most fateful error of the current economic system is treating natural resources as free goods and their consumption as income even though they are indisputably capital items (Schumacher 1973; Daly 1994). To spend capital is not a rational economic act. This also holds true for natural capital. Therefore, as Schumacher (1973:4) emphasized, "we should do everything in our power to try and minimize [its] current rate of use." First, nonrenewable resources are irreplaceable: once they are consumed, they are gone forever. For this reason, nonrenewable resources should be used only if they are indispensable (Schumacher 1973:44). Renewable alternatives must be created to replace them. Depletion of nonrenewables can further be reduced by increasing resource productivity and through recycling whenever possible. Renewable resources must also be used with utmost care, allowing them to regenerate. Second, the environment's capacity to absorb and eliminate waste is limited. Environmental sustainability requires natural capital to be consumed less than nature's regenerative and assimilative capacities.

Efforts toward sustainability, then, need to start with respect to ecological limits. Living in sight of these limits requires the endorsement of the principle of sufficiency. The concept of sufficiency is the most overlooked issue in the consideration of sustainability, albeit the key criterion (Schumacher 1973; Sachs 1993; Princen 2005). Sufficiency is a simple and realistic idea that indicates *enoughness* (Princen 2005:6). The concept refers to the ethic of consuming less, without threatening the welfare of other beings.

Present consumption habits entail the satisfaction of endless unnecessary human wants fabricated and imposed on people as if they were necessary. Fulfillment of these wants involves substantial ecological destruction and social suffering. It also threatens the wellbeing of future generations.

Schumacher (1973:42) emphasized more than three decades ago that "consumption is merely a means to human wellbeing" therefore, "the aim should be to obtain the maximum of wellbeing with the minimum of consumption." Today, facing the impending global environmental crisis, the human economy needs this wisdom more than ever.

Sustainable development can be defined as a process which aims to attain maximum human and ecosystem wellbeing today and in the future, with minimum consumption of natural capital. Any endeavor to develop sustainability must address issues of equity, sufficiency, effectiveness and efficiency. A new pattern of resource use is required that cultivates ecological, social and economic wellbeing. This calls for a new wisdom, hence new approaches to education, discipline and organization.

3 CONCLUDING REMARKS

Humankind faces today a world of diminishing resources, decreasing biodiversity, global environmental risks, social disintegration and impoverished prospects for future generations. These problems are all interrelated, all created by the material and energy intensive contemporary lifestyle. Tragically, this lifestyle is only enjoyed by a minority of humankind. The rest suffer from the negative externalities imposed on them.

The developed Northern economy is responsible historically for the current environmental degradation. The South has subsidized Northern consumption for ages, by providing raw materials, labor and dumping grounds. For sure, a small minority in the South enjoys the same consumption patterns as Northern consumers. Yet the majority in the South, which constitutes the majority of the world population, cannot even meet the most basic human needs. They suffer the worst consequences global resource plunder.

The powerless position of Southern countries in the international arena does not allow them to pressure the North to take up responsibility for corrective action. On the contrary, some industrial Northern countries have tried to hold developing countries accountable for their *future* impacts as a result of their population growth rates.

Developing sustainability poses multifaceted ethical problems which involve (1) ethical issues relating to the stewardship of nature; (2) equity issues across nations: between those that have historically consumed more resources across the globe and those that have suffered from the consequences of this consumption; (3) equity issues among social classes within nations; and, (4) equity issues across generations.

In view of these ethical issues, sustainable development has three guiding principles for political action (Sachs et al. 1998). First, the environment needs to be protected from irreversible damage (stewardship). Second, every human being should have the same right to globally accessible resources provided that these resources are not overexploited (global justice). Third, current needs (not wants) should be satisfied without endangering the needs of future generations (intergenerational equity).

Daly (1994:185) asserts that "sustainable development must be achieved in the North first." Northern nations have an ethical obligation to promptly encourage behavioral change in their countries toward less consumption. To do so requires public education about the environmental, economic and social impacts of excessive consumption, in which they indulge. Including the cost of environmental and social externalities in the price of commodities is a rational precaution to impede overconsumption. Moreover, governments of these countries must reformulate their domestic and international policies to discourage polluting industries, develop affordable and appropriate technologies and encourage ecologically responsible means of production and consumption to re-establish the harmony between humans and nature.

In the meantime, less developed countries should be provided with the same technological, financial and educational means as developed countries, *no strings attached*, to sustain their livelihoods for themselves without further damaging the environment.

Society at large is not fully aware of the magnitude of the environmental and social degradation inflicted by our consumption driven lifestyle. Education and public dialogue are essential to increase awareness of this reality. We do not live in a reality show: there is no escape to an outside world. Human wellbeing can not be thought of in isolation from the ecosystem. We need to abide by nature's laws to protect our welfare and that of future generations.

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Assessment of Quality of the Plates Using Single Linkage Cluster Method

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ABSTRACT Circle saw cutter machines are commonly used in the natural stone facilities in Turkey. Measuring the length and width of the natural stones plates sized using circle saw cutter machines and determination of those who are different from those measurements holds a great importance in terms of controlling the process. In this study, similarities and differences among the natural stone plates have been determined using single linkage cluster method which is one of the cluster analyses. General purpose of the single linkage cluster method is to classify the ungrouped data according to their similarities and to help obtain summarizing information. In this paper, lengths and widths of the natural stones plates, the edges of which were smoothened using a circle saw cutter machine, have been measured. Then, these measurements have been subjected to single linkage cluster method. After the method, it has been observed that the similarity between some natural stone plates is low. Consequently, it has been concluded that the quality of the natural stones plates, the edges of which were smoothened using a circle saw cutter machine, can be determined using single linkage cluster method.

1 INTRUDUCTION

In today's competition conditions, it is seen that the product quality, apart from the price, also affects the buying decision of the purchasers to a great extent. It is evident that the quality of the manufactured product is a very significant factor in determining the market share of the product. Since obtaining a sellable and serviceable product is the main purpose of the production, it is obvious that there is a requirement to be in a constant development in terms of achieving and controlling the quality in order to offer the purchasers higher quality, more reliable and economical products (Akçakoca et al., 2006).

Improving quality is possible by reducing the change in process and products. The quality level of a product can be measured with the help of quality characteristics, weight, length, etc. which is dependent on the target value determined by the customer prior to production. When the product interested is marble or natural stone, sales appeal and mechanical properties of the marble type comes to mind in terms of quality. Apart from the own quality of the marble type used, whether or not the product is cut in sizes requested by the customer also considered as a quality factor. For instance, if a countertop in 2.05 meter or 1.94 meter length is delivered to a customer who wanted a countertop in 2 meter length, the customer will not be satisfied regardless of which type of marble is used, and accordingly, it is considered that a poor quality product has been delivered to the customer (Saraç and Özdemir, 2003).

In this study, similarities among the natural stone plates, the edges of which were smoothened using a circle saw cutter machine, have been determined using single linkage cluster method, and it has been shown that marble and natural stone plates in different qualities can be determined by making use of these similarities.

2 SINGLE LINKAGE CLUSTER METHOD

Cluster analysis is an exploratory data analysis tool for solving classification problems. Its objective is to sort cases into groups or clusters, so that the degree of association is strong between members of the same cluster and weak between members of different clusters. Each cluster thus describes, in terms of the data collected, the class to which its members belong; and this description may be abstracted through use from the particular to the general class type (Einax et al., 1998; Kowalkowski et al., 2006; Yerel, 2009).

The steps in the single linkage cluster method are given as follows (Everitt, 1993; Sharma, 1996; Johnson and Wichern, 2002);

- The number of clusters and the number of observations are determined.
- The Euclidean distances between the pairs of cluster are calculated.
- N-dimensional symmetric Euclidean Distance Matrix (DM_E) is formed from using Euclidean distances.
- The Similarity Matrix (SIM_E) is computed by using DM_E. SIM_E is represented by the similarity values between pairs of cluster.
- Maximum similarity value between all pairs of cluster is selected and this cluster pair is joined to form a new cluster.
- Above steps are repeated n-1 times, until all clusters are combined in a single cluster.
- After, clusters are represented graphically in what is called a dendrogram.
- The dendrogram is examined and the erratic data caused by sampling errors are determined.

2.1 The Calculation of the Euclidean Distance

The Euclidean distance amongst the pairs of cluster is calculated as Eq. (1). Then, DM_E is formed by using the Euclidean distances (Johnson and Wichern, 2002).

$$d_{it} = \sqrt{\sum_{k=1}^{n}\left(c_{ik} - c_{tk}\right)^2} \qquad (1)$$

where;
i=1, 2,.., m,
t=1, 2, 3… m
d_{it} : The Euclidean distance between ith and tth clusters
c_{ik} : the value of the kth variable for the ith cluster
c_{jk} : the value of the kth variable for the tth cluster
n : the number of variables

2.2 The Calculation of the Similarity Value

The similarity value $(Sim(i,t))$ is calculated using Eq. (2). Similarity matrix (SIM_E) is formed for similarity values (Özdamar, 2004).

$$Sim(i,t) = 100\left(1 - \frac{d_{it}}{\max(d_{it})}\right) \qquad (2)$$

where $\max(d_{it})$ defines maximum Euclidean distance on DM_E.

3 CASE STUDY

In this study, lengths and widths of the plates which were released from the gang saw have been smoothened using a circle saw cutter machine available in a natural stone facility. Determining whether or not the length and width values of these plates are similar holds a great importance in terms of determining the quality of marble and natural stone. Twenty of the plates, the lengths and widths of which were smoothened using a circle saw cutter machine, have been selected randomly in order to determine this similarity. Then, length and width measurements have been performed on four different points of these plates. In order to determine the similarities of the plates with each other using length and width measurements, single linkage cluster method has been used. The main purpose of this method is to measure the similarity among the length and width values measured from different plates and to determine the plates, the quality of which is different.

In order to study the similarity among the plate lengths and widths using single linkage cluster method, distance matrices have been formed separately for plate lengths and widths. The number of plates (the number of samples) has been taken as twenty to form this distance matrix. Then, the cluster pair which has the closest similarity has been taken as a single cluster and this process have been repeated until all clusters gathered in a single cluster. Newly formed clusters and the similarities between the cluster pairs have been given in Table 1 and Table 2. Dendrograms have been drawn and interpreted for the lengths and widths using the values in Table 1 and Table 2.

Table 1. The similarities between the plate lengths.

Step	**Cluster number**	**Similarities (%)**	**Cluster pairs**	**New Cluster**
1	19	100.00	1–9	1
2	18	100.00	3–7	3
3	17	92.96	4–12	4
4	16	92.96	3–16	3
5	15	90.05	17–19	17
6	14	90.05	3–15	3
7	13	90.05	10–13	10
8	12	90.05	3–6	3
9	11	87.81	4–5	4
10	10	87.81	4–10	4
11	9	87.81	1–4	1
12	8	85.93	2–14	2
13	7	85.93	1–11	1
14	6	85.93	1–2	1
15	5	84.27	3–17	3
16	4	84.27	1–8	1
17	3	82.77	1–18	1
18	2	82.77	1–3	1
19	1	80.10	1–20	1

When the Table 1 and Figure 1 are examined, it is observed that the similarity values of the lengths belonging to the plates are very high. Accordingly, it can be stated that all plates have the same quality in terms of length. When the Table 2 and Figure 2 are examined, it is observed that the similarity ratio of the fourth and eleventh clusters with the other clusters is 63.77% in terms of widths belonging to the plates. After this cluster, it is seen that the cluster with the lowest similarity value combines with other clusters with a ratio of 76.28%. For that reason, it has been determined that the similarity ratio of the fourth and eleventh clusters with the other clusters is very low, and accordingly, they are of different quality than the other plates.

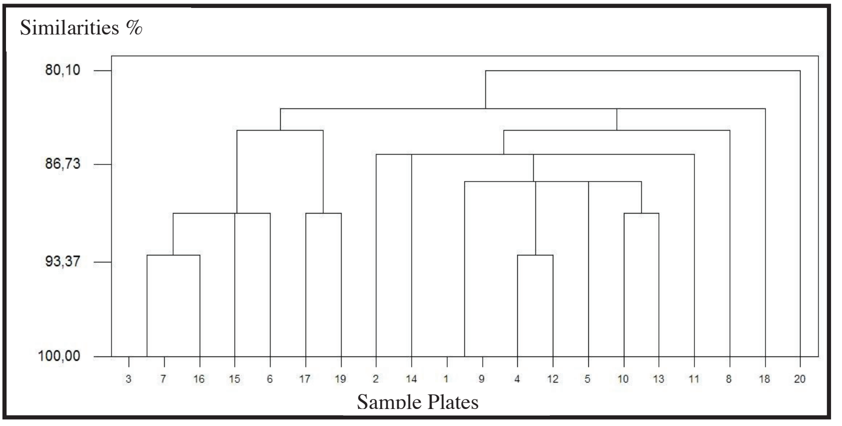

Figure 1. Dendrogram for the plate lengths.

Table 2. The similarities between the plate width.

Step	Cluster number	Similarities (%)	Cluster pairs	New Cluster
1	19	100.00	9–17	9
2	18	100.00	13–16	13
3	17	100.00	7–13	7
4	16	97.50	9–20	9
5	15	97.50	9–15	9
6	14	97.50	1–14	1
7	13	97.50	9–12	9
8	12	97.50	7–10	7
9	11	97.50	5–9	5
10	10	97.50	3–5	3
11	9	97.50	1–3	1
12	8	95.67	2–8	2
13	7	95.67	6–7	6
14	6	94.41	2–6	2
15	5	88.82	1–19	1
16	4	87.01	2–18	2
17	3	80.80	4–11	4
18	2	76.28	1–2	1
19	1	63.77	1–4	1

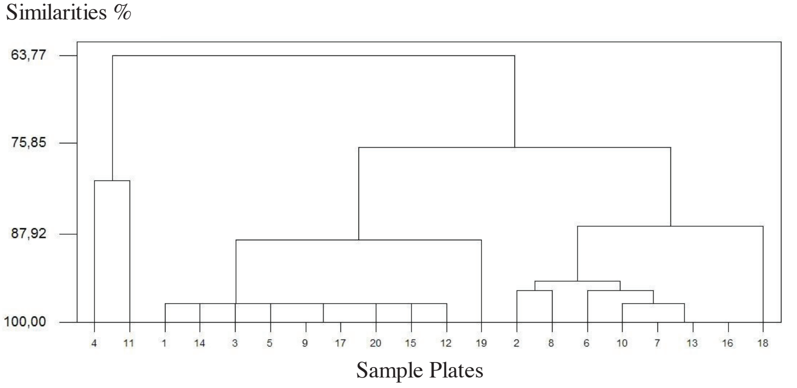

Figure 2. Dendrogram for the plate widths.

4 RESULTS

Single linkage cluster method, which is one of the cluster analysis, has been used in this study to determine whether or not the lengths and widths of the plates cut from the natural stone blocks using circle saw cutter machine in a natural stone facility are similar and to determine the plates, the quality of which is different. When the similarity among these plates, the lengths and widths of which have been measured using this analysis method, are examined, it has been determined that the similarities of the plate lengths are high. When the similarities of the plate widths are examined, it is observed that the similarity ratio of the fourth and eleventh clusters with the other clusters is 63.77%, in other words, they are of different quality than the other plates. As a consequence, it has been concluded that single linkage cluster method can be used in order to determine the natural stone and marble plates of different quality among the plates in the further studies.

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Availability Analysis of Mining Machines-A Case Study of a Dragline

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ABSTRACT Availability is one of the most commonly used performance metric of mining machines due to the easiness interpretation of its numeric results and can be expressed as a function of Mean Time between Failures (MTBF) and Mean Time to Repair (MTTR) which are the measures of equipment reliability and maintainability characteristics. In this study, electrical system availability of a dragline used in Tunçbilek Coal Mine (TCM) has been researched. In order to represent the failure and repair data probabilistically, the gathered data has been checked whether it is independent and identically distributed (iid) or not. Since both data has been found to be iid, Kolmogorov- Smirnov Test has been carried out to select the best-fit distribution. MTBF and MTTR values have been then calculated on the basis of the best fit probability distribution. Finally, electrical system availability of the dragline has been estimated and evaluated.

1 INTRODUCTION

Draglines are comprised of multiple electrical and mechanical systems that must be integrated together to allow the digging and dumping cycles. A dragline is in operational state only when both subsystem is in operational state and is in failed state whenever at least one of them is in failed state. The annual output of a dragline is a function of its availability. The loss of revenue for each day a dragline is out of action is estimated to be roughly $1 million (Townson *et al.,* 2003). For this reason, availability is a critical performance indicator of great significance to a mining company operating the dragline.

Availability is generally measured in terms of uptime and downtime. Increased reliability contributes directly to system uptime, while improved maintainability reduces downtime. Therefore, maintainability and reliability are two major system characteristics which are combined to form the commonly used effectiveness index availability.

In last two decades, mining industry has realized the effect of availability on economic sustainability of the activities. In this scope, efforts of collecting and analyzing the maintenance data to compute availability, reliability and maintainability have been increased considerably for the last two decades. Some examples are given by Kumar (1989), Kumar & Klefsjö (1992), Kumar & Huang (1993), Kumar & Vagenas (1993), Paraszcak & Perreault (1994), Majumdar (1995), Vagenas *et al.* (1997), Samanta et al (2001), Hall & Daneshmend (2003a, 2003b), Vagenas *et al.* (2003), Chorente *et al.,* (2004), Samanta *et al.* (2004), Barabady & Kumar (2005, 2007, 2008) and Elevli *et al.* (2008). Despite the fact that draglines are an important element of an opencast mine and their failures can have an important impact on the overall performance of mines with respect to production, there has been an inadequate attention on analyzing maintenance data of draglines in this scope.

The main objective of this study is to show how to estimate the availability of mining machines such as draglines. This kind of study will enable the mine managers better understanding the performance and maintenance need of their machine.

2 AVAILABILITY

Availability is a performance criterion for repairable systems that accounts for both the reliability (the probability that a given system will perform its intended function satisfactorily for its intended life under specified operating condition) and maintainability (the probability that a failed system will be restored to a satisfactory operating condition within a specified interval of downtime) of a system.

The term availability is used to indicate the probability of a system or equipment being in operating condition at any time t, given that it was in operating condition at t=0 (Rao, 1992). If repairs are always carried out upon occurrence of a failure, then the duration of the mission is the sum of MTBF and MTTR. In this case, availability can be calculated by equation 1.

$$A = \frac{MTBF}{MTBF + MTTR} \tag{1}$$

Where MTBF= Mean Time between Failures (Mean time that a system works without failure) and MTTR= Mean Time to Repair (Mean time taken to restore a system after a failure).

MTBF and MTTR are measures of equipment reliability and maintainability characteristics respectively and random variables. According to Equation (1), if MTBF is very large compared to the MTTR, then you will see the high availability. Likewise if MTTR is minor, then availability will be high. As MTBF becomes smaller (i.e., reliability decreases), better maintainability (i.e., shorter MTTR) is needed to achieve the same availability. That is, even a system with a low reliability could have a high availability if the MTTR is short. The reverse is also true. Thus, whether the availability improvement should be based on increasing the MTBF or decreasing the MTTR should be analyzed carefully. Table 1 briefly displays the relationship between reliability, maintainability and availability.

Table 1. Relationship between Reliability, Maintainability and Availability

Reliability	Maintainability	Availability
Constant	Decreases	Decreases
Constant	Increases	Increases
Increases	Constant	Increases
Decreases	Constant	Decreases

3 METHOD

Since MTBF and MTTR in Equation (1) are random variables, it is necessary to analyze these two characteristic in availability studies. In this respect, the main steps of this study are as follows:

1. Collection, sorting and classification of Time between Failures (TBF) and Time to Repair (TTR) data
2. Data analysis for verification of the identically and independently distributed (iid) assumption (If the data is found not to be iid then other models such as Non-homogeneous Poisson model or Power Law model could be used) ,
3. Fitting theoretical probability distribution given in Table 2 to the TBF and TTR data
4. Assessing the goodness of fit of a theoretical probability distribution to the data
5. Estimation of the reliability and maintainability parameters
6. Estimation of availability

Table 2. Theoretical probability distributions.

	Distribution			
	Exponential	Lognormal	Normal	Weibull
Probability Density Function, f(t)	$\lambda e^{-\lambda t}$	$\frac{1}{\sigma\sqrt{2\pi}}e^{-\frac{(\ln t-\mu)^2}{2\sigma 2}}$	$\frac{1}{\sigma\sqrt{2\pi}}e^{-\frac{(t-\mu)^2}{2\sigma 2}}$	$\beta\alpha^{-\beta}t^{\beta-1}e^{-\left(\frac{t}{\alpha}\right)^{\beta}}$
MTBF	$1/\lambda$	$e^{\left(\mu+\sigma^2/2\right)}$	μ	$\alpha\Gamma(1+1/\beta)$
MTTR	$1/\lambda$	$e^{\left(\mu+\sigma^2/2\right)}$	μ	$\alpha\Gamma(1+1/\beta)$
Reliability, R(t)	$e^{-\lambda t}$	$1-\Phi\left(\frac{\ln t-\mu}{\sigma}\right)$	$1-\Phi\left(\frac{t-\mu}{\sigma}\right)$	$e^{-(t/\alpha)^{\beta}}$
Maintainability, M(t)	$1-e^{-\lambda t}$	$\Phi\left(\frac{\ln t-\mu}{\sigma}\right)$	$\Phi\left(\frac{t-\mu}{\sigma}\right)$	$1-e^{-(t/\alpha)^{\beta}}$

4 DATA ANALYSIS

The data over a period of 5 years (2004-2008) have been obtained from Tunçbilek Coal Mine (TCM), which is a state-owned corporation of Turkish Coal Enterprises (TKI). Overburden removal operations rely on dragline stripping in this mine. Two draglines operate on different panels. Average yearly operating times are 4500 h and 5000 h for D18 and D26, respectively. Dragline maintenance data is regularly recorded in TCM under main headings of Electrical and Mechanical Subsystem. In this study, electrical subsystem data of D18 has been used. The main motivation behind the selection of electrical subsystem is related to the existence and quality of data which are Time between Failures (TBF) and Time to Repair (TTR).

4.1 Trend and Serial Correlation Test

Data are independent if there is no association between them. Independence among TBF (TTR) values generally means that large TBFs(TTR) are not necessarily followed by larger or smaller TBFs (TTR) and vice versa. Identically distributed means the probability distribution from which the TBFs(TTR) are derived is not changing (Jones, 1995). That is, the chronological order of data values does not contain any information. Analysis of availability data is based on the assumption that the failures/repairs are independent and identically distributed (iid). Because of this, an important task before the application of theoretical probability distributions is to determine if there is the existence of any trend and serial correlation in the TBF and TTR data.

The trend test and serial correlation test can be accomplished by both graphical and analytical. In this study, analytical methods have been utilized. Table 3 shows serial correlation results of TTR and TBF data. The strength of the relationship between present (t_2, t_3, ……t_n) and past values (t_1, t_2, ……t_{n-1}) is indicated by the Pearson Correlation (lag-1). The significance of the relationship is expressed in probability levels. Since the larger the correlation, the stronger the relationship is and the smaller the p-level, the more significant the relationship is, TTR and TBF data are independent at the 5% significance level.

The purpose of a trend test is to determine whether the values of a series generally increase or decrease. In this study, Unit Root Test has been used for trend analysis. A unit root means that the observed series is not stationary. If the Dickey-Fuller statistical value is smaller than the Mackinnon critical value then the null hypothesis of a unit root is accepted. Table 4 shows that both TBF and TTR data is stationary at the 5% significance level. That is, the values of future TBFs or TTRs are not influenced by previous values.

Table 3. Serial correlation results of TTR and TBF data.

	Pearson Correlation (lag-1)	Sig. (2-tailed)	Decision*
TTR	0.189	0.055	No Serial Correlation
TBF	0.012	0.904	No Serial Correlation

* Reject null hypothesis that sample comes from a population with zero correlation at lag-1, if p-level<5%

Table 4. Unit Root Test Results ($\Delta TTR_i \quad \beta_0 + \delta TTR_{i-1} + \varepsilon_t, \Delta TBF_i \quad \beta_0 + \delta TBF_{i-1} + \varepsilon_t$)

	β_0	δ	τ (Dickey-Fuller Test Statistics)	MacKinnon Critical Values	Decision*
TTR	14.6085	-0.8171	-8.3533	-3.4536	No Trend
TBF	209.7526	-1.0207	-10.2347	-3.4536	No Trend

*Reject null hypothesis that trend exist, if $|\tau| > |\text{MacKinnon Critical Value}|$

Since both tests indicate no trend and no serial correlation, then the assumption of iid data can be accepted and consequently the data can be fitted to theoretical probability distributions for availability calculations.

4.2 Best-Fit Distribution

Since the assumption of the iid is not contradicted, probability distributions given in Table 2 have been tried to describe the TBF and TTR data. For estimation the parameters of distributions, Statgraphics, statistical software package, has been used. Kolmogorov-Smirnov (K-S) test has been used as a measure of goodness of fit. According to the results given in Table 5 and Table 6, Lognormal and Weibull distributions provided the best fit for the TTR and TBF data respectively.

Table 5. Best-fit theoretical distribution for TTR.

Distribution	Estimated Parameter	K-S Test [DN, (P-value)*]	MTTR (h)
Exponential	λ=0.07721	0.3565 (0.0000)	12.9512
LogNormal**	Mean=1.0004 (Log Scale) Std.dev.=1.8196 (Log Scale)	0.0932 (0.3285)	14.2374
Normal	Mean=12.9512 Std. dev.=31.0605	0.3395 (0.0000)	12.9512
Weibull	Shape (β)=0.553986 Scale(α)=6.86518	0.1037 (0.2139)	11.5754

*P-values less than 0.05 would indicate that TTR does not come from the selected distribution with 95% confidence.

**The best distribution for TTR data is lognormal distribution.

Table 6. Best-fit theoretical distribution for TBF.

Distribution	Estimated Parameter	K-S Test [DN, (P-value)*]	MTBF (h)
Exponential	λ=0.0031	0.1313 (0.0554)	321.208
LogNormal	Mean=5.0050 (Log Scale) Std.dev.=1.4252 (Log Scale)	0.0786 (0.5581)	411.840
Normal	Mean=321.208 Std. dev.=387.653	0.2058 (0.0003)	321.208
Weibull**	Shape (β)=0.838568 Scale(α)=291.805	0.0642 (0.7847)	320.451

*P-values less than 0.05 would indicate that TBF does not come from the selected distribution with 95% confidence.

**The best distribution for TBF is Weibull distribution.

4.3 Availability Calculation

Based on the selected theoretical distributions indicated in Table 5 and 6, MTTR and MTBF have been found as 14.2374 and 320.4514 hours respectively. Based on these values, inherent availability of the dragline's electrical subsystem can be calculated by using equation 2.

$$A = \frac{320.4514}{320.4514 + 14.2374} = 0.9575 \quad (2)$$

Since the larger the availability, the more operational the dragline is, there exist a potential improvement area in effective usage of the dragline. As mentioned before, availability can be increased by reducing the MTTR and/or increasing the MTBF. To determine which of the two parameters have a greater effect on the availability is an important issue. In this scope, comparison of $\Delta A/A_0$ ratios can help to determine which parameter should be improved first. Since $\Delta A/A_0$ for a 1% decrease in MTTR is higher than the value for a 1% increase in MTBF, it can be concluded that decreasing MTTR provides greater marginal benefit for the electrical subsystem. For this reason, it is suggested that the implementation of preventive maintenance policy should be reviewed, adequate stock level of critical spare parts should be maintained, and the maintenance crew should be trained regularly in order to increase the maintainability of the dragline.

5 RESULTS

Since availability is a good performance indicator of overall equipment condition, there is a clear need to analyze availability of sophisticated, high capacity and capital intensive machines used in mines. Availability studies should be an integral part of mine maintenance management for the effective utilization of the machines such as draglines. This study presented the details of an availability study for a dragline used in TCM. Following studies should include the mechanical subsystem for evaluating the dragline as a whole.

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The World War Two Turkish Chrome Sales

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ABSTRACT The influence of strategically important materials on the global power relations is not a new phenomenon. During the 20th century production and trade of such materials played a significant role in the world politics. This work will be focusing on the case of Turkey to emphasize the significant role that chrome trade played in the Second World War.

Turkey's chrome policy during the Second World War was full with critical decisions taken , critical both for her own destiny and for the destiny of the belligerents. The Turkish chrome being a vital and insubstitutable product for the German war industry, caused the allies to give utmost attention to the 'pre-emptive war' regarding Turkish chrome.

By the 1942, both Axis and Allies started to search new solutions to the exhausting raw material stocks and they applied to Turkey more desperately then ever. On the other hand Turkey had her own calculations, first and foremost keeping war out of her borders.

Partly due to the chrome policy that Turkey followed and partly due to other specific reasons, one can talk about the failure of the Allied Economic Warfare regarding chrome in Turkey. More importantly, Turkey didn't yield to the pressure coming from the belligerents, she was even capable of making such strategic moves that would have changed the destiny of the war.

1 ECONOMIC WARFARE IN TURKEY REGARDING CHROME

1.1 Definitions

Alloys with diverse percentages of chrome are indispensable for metals that require strength, toughness, hardness, and resistance to corrosion or abrasion. Chrome is an alloy with a very high fusion point, and a remarkable ability to withstand corrosion from basic and acid slags, molten metals, and gas. It is used in the manufacture of steel and can be combined with other elements to produce a variety of chemical combinations many of which are industrially useful. There are three main uses for chrome: metallurgical, i.e. for the production of ferrochrome and other chrome alloys and for chrome plating: for the production of refractories for furnace linings, and chemical, for paints etc. Chrome is used in great quantities for making alloy steel. It enhances the hardness and the tensile strength of the steel and brings in properties such as rust-proof and heat-proofness.

Chrome is an essential constituent of all stainless steels and chrome-nickel and chrome-molybdenum alloys are required for armor plating, gun barrels and many types of shell Chrome is one of the seven major metals strategic for warfare: antimony, chrome, manganese, mercury, nickel, tin and tungsten. As Kemp mentions, Chrome is to modern industry as yeast is to bread; only a little is required, but without it there is no bread. It is a typical example of a strategic material that's dependence on foreign countries for supplies, for efforts to control and to stimulate production and imports during war time. It has a qualitative rather than quantitative importance.

More importantly, chrome is used in military production especially in tank manufacturing, production of tank munitions, cannon barrels, armaments, aircraft engines, important submarine parts such as crank shaft, periscope, navigation instruments, torpedoes,etc. Chrome is often mixed with other alloy metals. For instance Chrome-Nickel- Steel combination is used for the armor plates and gun barrels as well as Chrome-Nickel-Wolfram-Steel for machine gun parts. Chrome is used in the chemical industry as well. Among them chrome plating of the base metals for corrosion prevention and manufacturing of light sensitive films and preparation of leather are the most important ones. Due to its high melting point, chemical resistance and endurance to change of temperature, chrome is used in the fireproof construction industry.

"Without chrome there would be no modern arm industry." Although it was possible to substitute several of the steel-alloying metals by others, this was less true for chrome. Alternative steels, carrying less of the accepted was possible but since it was these alloys which made possible the high performance weapons of the second world war, any change in basic analyses from the complex high alloy to the simply low alloy steels, meant a decrease of performance and reliability.

Europe was poor in chrome reserves. Chrome production was far from supplying its own demands. Russia, with its pre-war efforts, managed to cover the one fourth demand of whole Europe. However, the greatest producer of chrome was consuming her whole production inland. The only chrome sources of Europe were laying in the Balkan peninsula namely in South Serbia and North and Mid Greece however, with low ore contents.

1.2 Germany's Dependency on Chrome from Abroad in World War Two

During the Second World War Germany was entirely dependent on imports of chrome ore and her stocks probably did not exceed 5 or 6 months consumption in the beginning of the war so that chrome must be considered as one of the most important German deficiency commodities ranking probably immediately after oil, iron nickel and copper

Germany's chrome import has risen tremendously from the beginning of the National Socialist era to the beginning of the war.

The import of chrome has doubled from 1933 to 1935. Chrome imported in 1938 was 85.000 tons more than the amount of chrome imported in 1935 and in the first 8 months of 1939 the amount of chrome imported was 64.000 tons more than the first 8 months of the previous year and 5 times more than the same period of 1933. Increase in the German chrome import rates is remarkable despite the extraordinary increase in chrome prices due to the world armament race. In the last years before the outbreak of the war, Turkey was taking the role as the main supplier of chrome compared to the British influenced Union of South Africa. One can talk about a decreasing dependency of Germany to the Allied controlled states since Turkey was becoming the major supplier of chrome.

1.3 Turkey and Chrome

High grade chrome ore produced in Turkey is used largely for metallurgical purposes and most of its applications in the field are very definitely of military character:

Chrome was first discovered in Turkey in 1848 and from 1860 to 1893 she was the principal source of the mineral. Owing to increased competition from other parts of the

World, chiefly in New Caledonia and Rhodesia, Turkey's output rapidly fell to a point where in 1921 only some 2.000 to 3.000 tons were produced. In 1938 Turkey regained its former position as the leading chrome producer with an output of over 200.000 tons mainly owing to the Turkish

Government's increased interest in the base metal industry and the exploitation of the Guleman Deposits. Chief producers of chrome in Turkey during the Second World War were: Societe Anonyme Turque de Minerals, Societe Miniere de Fethiye, Societe Anonyme Turque des Chromes de l'Est a Guleman. Majority of the companies were working in the Bursa region in north-western Turkey. In the district it was reported that the World's greatest known chrome deposit was discovered. The Fethiye Company exploited the deposits in the Makri region in south-west of Turkey. Four mines were opened and exploited.

Some lump ore 44-48% Cr_2O_3 was produced but the larger part of the output was in the form of concentrates running 50-52%. The Guleman Company exploited the chrome ore in the Elazığ region in the eastern portion of the country. The deposits were developed and production commenced in 1936.

1.4 Turkish Suppliers Value to Germany

The value to Germany of Turkish supplies was even more important than the figures suggested, since they had high chrome oxide content and the supplies of similar grades in Axis Europe were only sufficient to meet about two-thirds of the ferro-chrome requirements of 55.000 tons. Ferro-chrome could be produced from lower grade ores but such processing was expensive both in labor and power. In addition to that, certain of the really low grade Greek ores were not useable for military production.

1.5 Chrome Ore Pre-emption in the Second World War

It was believed that in spite of the Four Year Plan, Germany, by the outbreak of war would not be indefinitely self-sufficient in all raw materials and foodstuffs. In 1938, according to the British intelligence, estimated German stocks would last after the outbreak of war in 1939 not for more than a few months in most cases. Chrome ore was one of the most important of all raw materials of the British pre-emptive program.

Chrome ore was on the list of commodities which were believed to be of critical importance to Germany. German consumption of chrome under war conditions was estimated at about 150.000 tons per year, and stocks were believed to be equivalent to about 7 months supply. The supplies from Russia were uncertain, but the Russian transport system was weak and securing the whole or most of the output from Turkey, would mean exhaustion of the German stocks.

1.6 Allied Economic Warfare and the Turkish Chrome in the Second World War

With the beginning of German aggression and German-Italian rapprochement, Turkey felt the necessity to look for a new alliance that would counterbalance Italy and Germany. It also means search for a new trade partner, modernizer for the military and pioneer for industrialization. Great Britain was a candidate and the two states were getting very close just before the beginning of the Second World War. Attempts by Turkey and Britain to unite the Balkans against German aggression was recognized by the Germans as a hostile act. Hitler ordered that war material waiting to be delivered to Turkey was to be suspended. Including two submarines.

As a result, Turkey decided not to renew the German Turkish agreement on the exchange of goods and payments which was signed in 25 July 1938 for a period of one year and to be extended in July 1939. Delivery of chrome was also suspended.

After the conclusion of German-Soviet pact of 23rd August 1939, Turkey felt herself in a threat however she decided to remain neutral on the other hand refrained from renewing the Turkish-German trade treaty expired on 31st August which hampered the Turkish-German trade to a great extent.

These events affected the economic relations between Germany and Turkey but by the late 1939 Germany was still influential over Turkey as a result of her purchasing policy and clearing system. It was very difficult for Britain to take the role of Germany at once, she failed to do.

Britain, believing that she can substitute Germany as a trade partner of Turkey, attempted to make an agreement under which she should purchase the entire Turkish output of chrome and obtain a guarantee that chrome exports to Germany would be prohibited.

It soon became apparent that Allies should not achieve their object unless they give a certain amount of support to Turkish markets in other directions. The collapse of the dried fruit market in particular hit Turkey severely and Allies therefore agreed that any contract giving Britain and France the right to purchase the Turkish chrome output should have as a corollary an agreement for the purchase by them of Turkish dried fruits.

Negotiations for a chrome contract started after this good will gesture Turkey first insisted on signing a 20 years chrome agreement which might be exceedingly advantageous for Turkey as the price of the chrome after the war naturally sank. Chrome, therefore would turn into an insurance for the long term debts of Turkey.(Transferred to Turkey from the Ottoman Empire)

Turkish proposal for signing a 20 year contract was that if Allies and Turkey signs a war time contract turning Turkey in to a war time supplier, it means for Turkish side the end of neutrality therefore being obliged to defend herself against German antagonism. On the other hand, if the contract is made for a 20 years period, Turkish side would be able to show to the Axis that she is not a war time supplier and still a neutral country.

Britain and France and Turkey in December 1940 agreed to buy, in proportions 11 to 4, all chrome up to production (limited to 250.000 tons). Other neutrals were to obtain chrome only by receiving the consent of the United Kingdom or France. Anglo-Franco-Turkish Financial and Commercial Agreement was also signed in Paris, January 8, 1940. The effect of the Agreement was that Turkey was prevented from selling chrome to any destination except the U.S. without the consent of the British and the French Government. Agreement has been reached on the basis that British and French Government buy Turkish chrome for 2 years with option on either side to renew for 1 year, the price of chrome was to be that of the London Metal Exchange when Germany occupied France, Germans applied to the Turkish decision makers to get the share of France which was 4/15 of the total chrome output.

Britain argued that she was ready to buy the share of France and she had priority. Turkey agreed to sell the share of France to Britain and Britain resold it to US.

Meanwhile price negotiations lasted so long due to the huge price gap between the supplier and the buyer. As a result:

Turkish side started to believe that Britain was behaving reluctantly to buy Turkish chrome. Besides that, Germany was offering 100% higher than the price offered by the British. Mine owners wanted to sell chrome to Germany as the prices were higher.

British side was not willing to rise the prices arguing that she was offering the prices of the London Metal Exchange . On the other hand Turkish side believed that London Metal Exchange prices were manipulated by the British. British later on confessed that they were manipulating the London Metal Exchange prices.

Chrome stocks were piling up in Turkey as Italy's entry into the war was making the sea transport between Britain and Turkey almost impossible so that neither chrome could be transported to Britain nor Britain could ship anything from Great Britain to Turkey.

Turkey had no choice but to go back to the old ally Germany.

All these events made the rapprochement between Germany and Turkey inescapable.

Britain could not replace Germany as a trade partner: Turkish products were attainable from the British colonies.

On the other hand, Germany was a natural market for the Turkish surplus raw materials and foodstuffs.

When Germany was cut off from overseas markets by the sea blockade, Turkey's chrome, cotton, tobacco, dried fruits, oil seeds, valonea and wool became more crucial for Germany, especially in terms of attainability.

Finally the disappointment of Turkey by not being able to sign an agreement with Britain on chrome and other Turkish commodities such as wool, cotton, mohair and olive oil with Britain, led on June 25, 1940 to a one year commercial and payments agreement with Germany. Turkish-German trade volume was fixed to 21.400.000 TL. Germany was supplying thirty-nine locomotives, wagons, factory equipment, spare parts, pharmaceutical and cigarette paper.

In the late 1940, the Allied purchases of chrome became more than a merely pre-emptive purchase, due to the increasing chrome demand on the side of the Allies as a result of increasing military production.

US lost Philippines -the main chrome supplier- to Japan. Stocks of the allies exhausted as the production increased tremendously.

German chrome stocks exhausted as well, German calculations about the end of the war didn't come true. Long lasting war and increase in German military production forced the Germans to knock the door of the Turkish statesman for a new era in the German Turkish relation.

Turkey agreed to supply chrome to Germany with the Turkish-German trade agreement of June 10, under which half of the total Turkish production would go to Germany. During the negotiations with the Germans Turkish side didn't mention anything about the time of the chrome deliveries to be started.

On the other hand Turkey stayed liable to her agreement with the allies and didn't deliver any chrome until January 8th 1943. (Turkish-British Agreement was due to January 8th 1942 with an option to extend for one year.)

By the end of 1942, months before the end of British chrome contract, Britain started the evacuation of chrome. According to the contract all the chrome mined before the 8th of January 1943 belonged to Britain. So Britain increased the price paid for chrome in order to prevent the mine owners stockpiling. She made bonus payments to the mine owners to fasten the production.

The idea behind this was to buy all the chrome produced before 8th January 1943 so that Germany could not get any chrome before the mid 1943.

Mining and transportation was taking at least 4-5 months especially during the harvest season and in winter transportation of chrome cease was suspending due to the lack of wagons and bad weather conditions.

First party of chrome to be delivered to Germany was 45.000 tons and first chrome delivery took place in mid 1943. Second party of delivery which was 45.000 tons as well was dependant on the delivery of war materials to Turkey until March 1943. As this happened, Turkey delivered started the second party delivery and by the end of 1943 and beginning of 1944 great amounts of chrome was delivered to Germany despite of British and US protests and threats.

While Turkey's entry into the war against the Axis was discussed heavily in the conferences of Casablanca and Tehran, Churchill was meeting in Adana (Southern Turkey) with Turkish President İnönü, and discussing the involvement of Turkey in the war on the side of the Allies, Turkey kept selling chrome to Germany.

Allies protested the increasing deliveries of chrome to Germany. US and Britain threatened to apply blockade measures to Turkey immediately if she does not stop the delivery of chrome to Germany. Turkey resisted these demands due to the fact that she was afraid of a retaliation to come from Germany in the form of bombing of the cities such as İstanbul and İzmir if the chrome deliveries to Germany ceased.

As the German campaigns in Soviet Russia failed and Russian campaign towards Berlin started, Turkey ceased the delivery of chrome to Germany in April 21st 1944 and declared war on Germany in February 1945, months before the end of World War II, in order to be involved in the conferences in which the new world order will be decided.

Germany didn't get any chrome officially from Turkey after this date but she kept receiving chrome from unofficial ways. German personnel smuggled chrome to

Germany until the end of war in various ways.

2 WHAT WERE THE REASONS FOR THE FAILURE OF THE ALLIED ECONOMIC WARFARE REGARDING CHROME

Economic Warfare of the Allies didn't manage to cease the supply of war materials of excellent quality to Germany completely, and Germany's war production and military operations were never seriously hampered by a shortage of any essential raw material or industrial products, with the single exception of petroleum. The allied economic warfare was so ineffective and disappointing.

2.1 Failure of the Tripartite Alliance

Aim of Great Britain was to help the Turkish economy and while doing this, preventing the supply of chrome to Germany. Great Britain was not able to avoid Turkey signing three trade agreements with Germany, the only thing she managed to do was to curb the volume of Turkish-Germany trade. Turkey was attached to the allies with the tripartite alliance. The malfunctioning of it was caused by the uncertain place that Turkey occupied within the strategy of Great Britain. At first, Great Britain followed an ambiguous policy torn between Italy and Turkey. British strategy was not to disturb Italy who pursued an expansionist policy during the mid 1930's.

Britain was misinterpreting the Turkish view. According to her, although Turkey was not participating in the war, she was an ally and was therefore under a moral obligation not to supply chrome to Germany. It was recognized that this would entail some hardship, but it was her duty to be prepared, as an ally, for shouldering her share of the burden of the war.

Turkish side, on the other hand, argued that: they must have arms and equipment for their military; and machinery, locomotives, rolling stock and other supplies essential for the maintenance of their economy. Britain was unable to supply Turkish demands, and Germans were willing to do providing that Turkey supplies chrome.

Allied governments had been short sighted in limiting their chrome agreement of January 1940 to two years. Turkish Secretary General Numan Menemencioğlu was trying to sign a long term, 20 years, chrome agreement with the British and the French, and was trying to make chrome into a security for the creditors, which might be exceedingly advantageous for Turkey as the price of the chrome after the war naturally sinks.

On the other hand Allies were arguing that any chrome contract signed between the Allies and Turkey would turn Turkey into a war time supplier, which means the end of the neutrality. More importantly she would be ina situation to defend herself against German antagonism. On the other hand, according to Menemencioğlu, if a 20 years contract was made, the Turkish side would have a proof showing that their chrome contract with the Allies was a long term one for the compensation of the depths and has nothing to do with the recent disputes between the axis and allies.

2.2 Great Britain not Being able to Replace Germany as a Trade Partner

Turkey's economic predicament and shortages during the war were created by its prior dependence upon Germany. The war conditions forced Turkish statesmen to achieve closer economic ties with the Allies but they faced with the fact that Turkish trade routes went under the control of the Germans which prevented the Turkish export from going to the Allied destinations. As a result, Turkish government went back to the old ally.

2.3 Piling Chrome Stocks in Anatolia

Huge stocks of chrome piled up in Turkey during 1940, it was not only dangerous in the sense that they could be stolen by Germany but also there was not enough storage places for chrome masses. There was variety of reasons causing the piling of chrome in Turkey. Firstly, U-Boats provided dominance in the seas to the German war machine. German U-Boats not only guaranteed Germany a military superiority in most of the sea routes but also made transport of Turkish chrome to Great Britain risky. U-Boats patrolling in the English Channel were making chrome deliveries from Turkey to Great Britain to a great extend impossible. Secondly, infrastructure of the Turkish railways was not in a good condition. Most of the railways routes were either too busy with the Turkish priority deliveries or not in a good condition. Thirdly the Greek-Italian war was making the sea transportation in the Aegean Sea not safe. Most of the local ship owners did not want to carry chrome to Great Britain, and
British ships due to demand in other parts of the world were not most of the time ready for the transport of the ore. Lastly, the price negotiations were too tough and long lasting for both sides and caused long delays.

2.4 Ineffectiveness of the Pre-emptive Buying

The efficiency of pre-emptive buying was very limited. It was causing the mine owners to put more production on the market with the hope that all their production will be taken, therefore it was making the control of the chrome very hard. On the other hand Agreements between Allies and Turkey to restrict exports to Germany was not working efficiently while the Allies were not capable of preventing smuggling or under-declaring. For instance, up to 30th November, 1943, 281.232 tons of chrome ore was declared by the Eti Bank as having been shipped from Turkey, but of this amount only 262.440 tons have arrived in the United States. The difference was 18.885 tons. It is possible that the Germans made a clandestine arrangement with the Masters of the vessels to off-load some of the ore at one of the German occupied islands on the route. By this means they may have acquired chrome for movement to Germany in addition to the Clodius amounts.

The mine owners were very influential indeed. They were vetoing the prices offered to the Turkish Government by Great Britain. During the war, Great Britain attempted to bring the prices under control but didn't manage to do. On the other hand, although Allies had influence over the Turkish Government and Etibank, they had little influence over the private mine owners, who were not declaring their production to the authorities. Throughout the war, mine owners worked for the Germans to a great extend until the U.S. representatives came and offered higher prices. American market was more profitable than the British one for the Turkish chrome. The individual mine owners, such as Mr Paluka, were preferring to sell his production to the U.S. market but Etibank was contented with Paluka's individual relations while, the aim of Etibank bureaucrats was to enter the American market as the profit was much more, but they wanted to make their own connections and didn't want to involve the mine owners in to the game.498

2.5 The Trade Routes Being Out of the Reach of the Blockade

Turkish chrome was going to Germany through Bulgaria. Burgaz was the main port of entry for Turkish imports destined for German use. From Burgaz, the chrome was transported along the Danube via the important river port of Ruschuk, or along the main railway line from Burgaz through Sofia, Nis, and Belgrade to Zagreb. When this line was cut in Croatia, as it was often done, through partisan action, the chrome was routed straight from Belgrade to Budapest. There were other ports along Danube, connected by rail to the Bulgarian railway system, Lom, Somovit and Svishtov. These port facilities were also used by the Germans in Emergency. When Burgaz was denied to Germany, the chrome traffic was routed through Constanza, in Romania. This route became very congested after the Russian advance into Northern Romania started, it was the main line of supply for the German Southern Front. Germans made every effort not to use it for chrome traffic. For the Allies, the capability of interrupting the chrome transfer to Germany was very limited. The only way to interrupt the railway traffic was bombing but when Allies dropped bombs to the railways Germans shifted to trucks and continued transportation. Partisan attacks to the railways were another method used by Great Britain to interrupt the traffic but partisans were working with the Russians.

Under these circumstances Great Britain could not use the most effective feature of the Economic Warfare, simply the contraband control and also could not cut the chrome traffic between Turkey and Germany permanently.

2.6 British Plan to Buy Turkish Chrome at a Lower Price and Change the Conditions of the Agreement

The price negotiations between Turkey and the Allies was half the battle, and most of the time failed due to various reasons. Turkish side being a haggler was not the only reason behind the prolonging negotiations. The British side was as guiltier as the Turkish side for the delay.

After signing the chrome agreement with the allies, price negotiations began immediately. The British side found the price proposal of the Turkish side so high that they said 'Unless Turkey modifies her price ideas, she will not sell a ton.'501 According to them, the offer received was supposed to be rejected. A counter offer was made, which would be approximately the same as the price at which supplies can be obtained elsewhere, i.e. 125/- basis for 48% c.i.f. (cost insurance freight) UK and 142/- for 48% c.i.f Marseilles, which were the present prices for the British ore, on the basis of today's sea freights from Rhodesia. These prices were London Metal Bulletin prices. Great Britain was planning to drop the negotiations for the time being if their offer was rejected by the Turks. They were planning to cover chrome requirements from Greece or own mines of France and Great Britain, if special grades were not available from the Turkish production. The price of chrome was defined in the Chrome Agreement as that of the London Metal Exchange, Turkey however, demanded a higher price, using an offer made by Germany to supply sorely needed armaments in return for chrome. British aim was to maintain the present London Metal Bulletin prices, so that it would not encourage increased production from British and other sources and reduce the liability under the Turkish agreement. According to the plan, Turks were expected to realize soon that they could not sell their ore and in their anxiety for foreign exchange would meet the market prices. MEW was thinking that the present attitude of the Turkish side was encouraged by the high prices paid for the Greek ore. On the other hand British bureaucrats believed that 'in the past the British have learned to expect hard bargaining from the Turks, but where their bluffs have been called, they always paid dearly. It is for this reason that Great Britain should reject their offer and give a counter offer at open market prices.

Allies were thinking that the right to preempt chrome was their principle reward for buying the raisins.

2.7 Inadequate offers made by the British Government for Turkish Chrome

The prices offered by Great Britain for the Turkish chrome was refused most of the time by the mine owners. The Turkish Minister, responsible for the mining was furious to such an extent with the British negotiators that he told Etibank that he was ready to break off negotiations with England and sell Germany. On the other hand, although the reaction of the Minister and the mine owners to the British price was very severe, British officials believed that president İnönü and Prime Minister Şükrü Saraçoğlu had more to say on this than a simple cabinet member. This was partly true but diplomatically at fault. On every occasion both the British and the Turkish side were accusing each other of having ‘a rather undignified state of haggling’ The British side suffered for her price policy on the later stages of war, she conceded to pay 270 shillings for the Turkish chrome in a time when Germany was getting the same amount or even more for the same price. On the other hand if Great Britain had paid the same amount or accepted the 20 years contract without taking into consideration the economic burdens, she wouldn’t have found herself in such a serious situation.

2.8 The British side Blackmailing Turkey to Agree Upon The Chrome Price

The British plan was to force Turkey to limit its chrome production to an amount that the acquiring of the whole output would be lot more easier. In order to achieve this goal, they step by step limited the market of Turkey. Along with the allied powers, selling of chrome to the Scandinavian countries was also denied. However, this was only a plan and it was hard to make Turks to assent on such a ban, as long as the buyer countries kept neutral.

The Turkish side believed that there was a commercial intrigue against them. According to them the London Metal Exchange price was not the real price and was manipulated by Great Britain.

There was enough reason for The Turkish side to believe in this conspiracy because they made a recent sale to the USA at 28.50 dollar(138.2 shillings) a ton which was equivalent to 22.75 fob dollar. London Metal Exchange price was 125/ less 52/- freight, or 72/fob which was 14 dollar at the present free rate of exchange. Besides that the delivery of chrome with Turkish ships was also out of question while SS Refah, a Turkish ship, carrying chrome to Great Britain sunk in Mediterranean, believed to be sunk by the Germans. Americans misinterpreted this case, by thinking that Turkey was not willing to deliver chrome to Allies because she was pro-German. On the 26th of July 1941, Great Britain applied to United States to make the deliveries to Turkey through Suez, which was still a safe route. British couldn’t persuade the Americans on this issue, as the Americans were insisting on the delivery of chrome as a precondition to deliver any good to Turkey. Besides all, the harvest season was coming most of the wagons were to be sent for the transportation of the harvest. Not many wagons could be spared for the transportation of chrome during the harvest season. There was flood as well. Some of the transportation routes had to be moved. All these factors were making the delivery of chrome to the allied destinations more problematic. British at the end said: If the Americans insist on carrying this policy to its logical conclusion we cannot maintain our position in Turkey and the American attitude may force Turkey more into the German economic orbit. Based on the diplomatic dispatches between the allied authorities, one can come to the conclusion that Allied approach to the Turkish State was contemptuous. The diplomatic staff did not hesitate to insult the Turkish behavior in general and hint at the hatred that they feel against Turkey in general. The ill feelings they maintain were from time to time having its effect on the decisions taken.

2.9 Attempts to Bribe the Turkish Authorities

Most of the dispatches were away from diplomatic propriety, For the British

Authorities Turkish bargaining strategy was a result of greediness and giving concessions to Turkey was described as ‘bribing the Turkish authorities’. The most moderate tone within these dispatches were defining the bargaining process as ‘making some minor concessions to Turkey’.

2.10 Turkey Being Against the Unconditional Surrender of Germany

The allies demanded the unconditional surrender of the Germans in January 1943.
According to the Turkish Government, The doctrine of unconditional surrender was the

greatest failure of the Allied policy. The fate of Europe and Germany were connected to each other. The decision of the Allies to destroy Germany's power unconditionally was perceived as a threat by the Turkish statesmen. In the case of an unconditional surrender, Russia was going to be left unchallenged in south east Europe if Germany would surrender unconditionally and United States would not intervene to the South Eastern Europe that would lead to the Bolshevization of Turkey and Europe. It was already possible that Soviet Union might occupy Turkey if Turkey participated in the war.

Besides that, when Germany began to retreat from the Soviets territories, Hitler persistently ordered the defense of Crimea. When his generals wanted to retreat from the

Crimea, he overruled them on the assumption that as long as Germany was present on the Black Sea, Turkey would continue to resist Allied pressure and sell chrome. When Germany retreated from Crimea (May 8, 1944), Turkey ceased selling chrome to Germany (April 21, 1944) the last delivery of chromium to the German was on June 1, 1945.

3 CONCLUSION

The Allied Economic Warfare regarding the chrome sales failed in Turkey as a result of a series of facts derived from the Allies, the Axis and Turkey herself. Among many critical mistakes of the Allies, the most important of all was her reluctant behavior towards Turkey as a result of having the monopoly over the Turkish chrome in the first phase of the war. Allied reluctance caused the Turkish economy to suffer to a great extent. Great Britain did little to take the role of Germany as Turkey's major partner. Besides that her intelligence did not work effective regarding the German deficiencies and fed the British Foreign Office with wrong information which caused the British Ministry of Economic Warfare to focus on the wrong aspects. On the other hand, Turkey followed some simple principles and hardly deviated from them. The most important of these principles was keeping out of the war no matter what, not being too much dependent to any of the belligerents and being against any unconditional surrender of the belligerent parties which would destroy the balance of power in Europe. The Turkish statesmen realized the chrome export as the most important asset in hand, and used it in every occasion as a bargaining chip. It took a while for the British side to realize this. It was due to the fact that during the early years of the war, buying the Turkish chrome was nothing but a pre-emption and once the roles changed and the chrome demand of the Allies increased, they strived to understand Turkish point of view, but it was too late. Once the Turkish point of view was realized, Germany was already entitled to chrome.

Germany on the other hand benefited from the deeply rooted economic relations, that's foundations were laid during the 1930's. The dependence of Turkish economy to Germany was to such an extent that the attempts of the Turkish side to decrease this dependency.

Beginning from the last days of the peace failed and at the end in the early 1940, Turkish Government realized that their attempt to substitute Germany with Great Britain failed. It would be a failure to explain the failure of the Turco-British alliance with one fact; it was both the German 'lebensraum' policy that rendered Turkey dependant on Germany, and the British inability to substitute Germany as a trade partner that led to the rapprochement of Germany and Turkey in 1940. It would be also necessary to mention the role of German military success in the South East Europe and its effect on the Turkish public and politicians. In the eyes of the Turkish people and specifically the Turkish ruling elite, Germany was the old ally against whom not a single war was fought. On the other hand despite the improving relations the credibility of Great Britain was poor in the beginning of the war and decreased each passing day.

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IT Application for the Occupational Health and Safety Integrated Management System

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ABSTRACT This paper intends to present the design and the production of a software for the Occupational Health and Safety Integrated Management System with the view to a rapid drawing up of the system documents in the field of occupational health and safety. This software has been produced in compliance with the provisions stated by NUCLEU program, Project no. PN-07-45-01-03.

1 INTRODUCTION

History has shown that a correct implementation and maintenance of a Occupational Health and Safety Integrated Management System can bring about several benefits both to the employers and tot o employees:

- an easier way for an organized evaluation of risks, hazards and incidents;
- a better awareness of hazards and of risks;
- improving the transparency of internal processes;
- improving communication among employees;
- improving the motivation and identification of the employees with the interests of the undertaking where they work;
- a suitably integrated approach of the working environment;
- a better quantisizing of the OHSAS.

In order to provide a good implementation of the Occupational Health and Safety Integrated Management System, a software specific for diagnosis of occupational safety could prove its usefulness; there have taken into consideration the notice on the implementation guide for the model of the Occupational Health and Safety Integrated Management System and the tools used for auditing this type of system with the view to drawing up the system documents in compliance with the principles and practices used by the quality system.

Consequently, there had been drawn up 15 system documents (checklist with the management requirements in the field of OHSAS, a handbook for the management of OHSAS and 13 system procedures); these documents were transposed into an expert software that can provide certain major benefits derived from the facilities determined by the use of professional computers: high capability for the storage of information, rapid search of information, high data, processing speed etc.

2 THEORETICAL AND PRACTICAL APPROACH OF THE OCCUPATIONAL HEALTH AND SAFETY INTEGRATED MANAGEMENT SYSTEM

An integrated management system represents a logic and organized managing – type approach; it allows to take the best possible operating and strategic decision that takes into consideration all the essential aspects that lead to an efficient operation of an undertaking, both from a qualitative point of view and from the point of the surrounding environment, or of food safety and hygiene etc

The integrated management systems display several advantages:

– they provide a higher coherence;
– they provide an optimization of operations;
– they provide a diminution of overlaps and of dissipation of responsibilities;
– the common requirements are being fulfilled in an efficient manner;
– a joint approach of the future development with the diminution of costs related to the certification process.

An integrated management system is made of at least two different management systems:

– OMS+EMS (quality management system + environment management system);
– OMS+OHSAS (quality management system + occupational health and safety management system);
– OMS+EMS+OHSAS (quality management system + environment management system + occupational health and safety management system).

An advanced management system shall have to have a modular structure so as to allow the entrance of new elements into the current system.

OHSAS covers a set of elements bearing a decision like, organizational, information and motivator character within the undertaking; it is being used for the implementation of the processes and relations inside the occupational health and safety integrated management system with the view to getting the required occupational health and safety level.

The implementation of an OHSAS in accordance, with SR OHSAS 18001:2008 provides several advantages that strengthen the idea that the economic agents:

– shall observe the legal and regulating requirements;
– shall possess an OHSAS at a very high standard, included in the general management of the company.

Consequently, the following aspects shall have to be settled for a suitable development of the activities at the level of the national economy:

– accident risks;
– manner used to diminish the occupational diseases as much as possible;
– manner used to measure, monitor and evaluate the implementation of the OHSAS.

An improved occupational health and safety shall cover:

– a well planned and well-documented approach of the issues related to the OHS;
– procedures related to OHS that define the areas of responsibility;
– a familiarization with the issues related to the OHS and a stable social climate favorable to high performances in the undertaking;
– issues related to the improvement of the OHSAS with quantized results;
– a diminution of risks related to occupational diseases and accidents;
– inspections and operations bearing a preventive type character;
– exercises and practical operations that have proven their efficiency in emergency situations;
– analyses audits and recordings of the OHS management;
– a periodical evaluation regarding the application of the legal and regulating requirements that also provides a diminution of probability for the payment of legal and compensatory penalties;
– a good motivation of the working personnel and a diminution of operating costs;
– a dissemination of the efforts with respect to the compliance with SR OHSAS 18001:2008 to all the interested parties;
– an improvement in the image of the economic agent by increasing

competitiveness and improving the relations with the public authorities.

3 DOCUMENTATION OF THE QUALITY SYSTEM IN THE FIELD OF OHSAS

There have been devised the models of the documents that belong to the quality system in the field of OHSAS for a suitable implementation of the principles listed in the model of the OHSAS proposed in application guidelines.

The production of the system documents in the field of OHSAS has taken into consideration the tailoring of the requirements stated for the proposed model to the organizational particular aspects of the production specific to economic agents that carry on their activities in atmospheres with hazard of explosion and for in toxic atmospheres and the requirements stated by the legislation in force in the field of OHS, especially the Law no. 319/2006 of OHS and the Methodological Norms for the implementation of this law.

Basically, the documentation of the OHSAS includes the following items:

– checklist with the requirements of the OHSAS;
– handbook of the OHS management; the main document of the system that is the basis for all the other documents;
– the procedures of the system that provide details on certain provisions stipulated by the handbook; there are also established concrete aspects on responsibilities, functions, competences, actions, information flows, recording of information so that the system should work in compliance with the requirements stated by the legislation in force and by the model and all the general objectives related to OHSAS should be reached.

3.1 Checklist with the Requirements of the OHSAS

This checklist is a working document that comprises the description of the requirements stipulated in SR OHSAS 18001:2008 and provides the possibility to check the observance of the corresponding requirements from the above mentioned standard, with the view to a correct and complete drawing up of the system documents (MC-SSM, PS-SSM).

3.2 Model for the Management Handbook of OHS

The model of the basic document for the OHSAS has been drawn up by taking into consideration the following issues:

– the requirements of the legislation in force related to the OHS;
– the requirements valid for the model for OHSAS proposed in the paper;
– correlation between the OHSAS and the quality management system existing at the offices of economic agents;
– particular aspects related to the organization of the economic agents.

The handbook starts with the "Political statement of the managing board in the field of OHS"; this document covers the approach and the objectives of the managing board.

The introductory part of this model comprises a general presentation of the economic agent, the scope and the targets and the drawing up and the management of the handbook.

The subsequent clauses in the handbook present the structure of the model of OHS and basic issues that correspond to each element of the system that provides the observance of the requirements for the model, together with the implementation, maintenance and improvement of the system.

3.3 Models of the Procedures for the OHSAS

The system procedures provide details on certain provisions stipulated by the handbook; there are also established concrete aspects on responsibilities, functions, competences, actions, information flows, recording of information with the view to providing compliance with the requirements of the model and efficiency in the operation of the OHSAS.

The drawing up the models for the system procedures has taken into consideration two main requirements for the model that might seen in opposition one from another:

– the volume of the documentation shall be maintained to necessary minimum in order to provide the efficiency of the system;
– the documents shall cover a general and detailed description of the basic elements of the system and of the relations from among them.

To this end, there have been determined the models of the following system procedures: PS - SSM – 01 “Identification of hazard, risk evaluation and establishing the verification”; PS - SSM – 02 “Evaluation of compliance with the legal requirements and other requirements”; PS - SSM – 03 “Competence, training and raising awareness”; PS - SSM – 04 “Communication”; PS - SSM – 05 “Participation and advisory operations”; PS - SSM – 06 “Control of documentation”; PS - SSM – 07 “Preparedness for emergency situations and capability for answering back”; PS - SSM – 08 “Monitoring and measuring the performances of OHSAS”; PS - SSM – 09 “Investigating the incidents”; PS - SSM – 10 “Non-compliances, corrective and preventive actions”; PS - SSM – 11 “Control of recordings”; PS - SSM – 12 “Internal audit”; PS - SSM – 13 “Management Analysis”.

4 SOFTWARE FOR THE OCCUPATIONAL HEALTH AND SAFETY INTEGRATED MANAGEMENT SYSTEM

This software is in EXE format and it has been designed in Visual Basic. Figure 1 shows the logical diagram of the mathematical algorithm proposed for the drawing up documents related to the OHSAS.

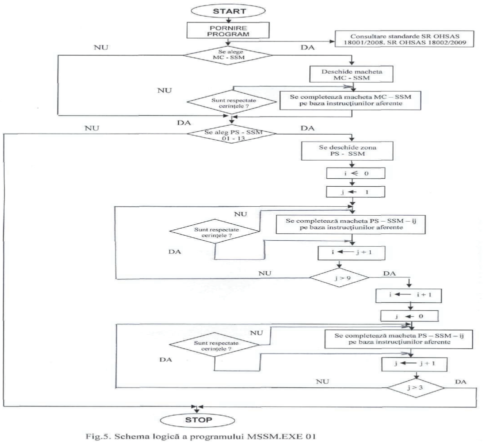

Figure 1. Logical diagram of the mathematical algorithm.

MSSM.EXE 01 software represents a working tool that is being used for the drawing up the system documents for the OHSAS. The input data covers the following issues: objectives of the OHS, the targets, the scope, the responsibilities with respect to the drawing up and the management of the handbook for OHS, the implementation, maintenance and improvement of the system related to OHS, action means, information flow, means used to record the information with the view to providing compliance with the requirements of the model and an efficient operation of the OHSAS. Figure 2 shows the start page of MSSM.EXE 01 software.

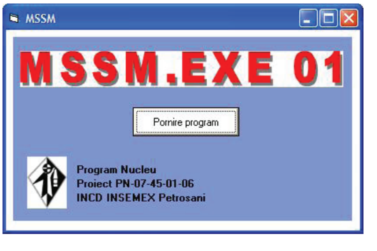

Figure 2. The start page of software MSSM.EXE 01.

By pressing the button "**Pornire program/Start software**" the application is being launched and the main window that contains 4 distinct areas is displayed. Figure 3 shows the working window where the user shall select from among the 4 areas:

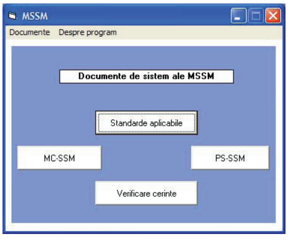

Figure 3. Working window in MSSM.EXE.01 software.

These areas provide the following facilities:

1. The area " **Standarde aplicabile/ Applicable standards**", comprises the applicable standards for the domain of OHSAS. They can be visualized as pdf. These standards are the following ones: SR OHSAS 18001 :2008 "Occupational health and safety management systems. Requirements" and SR OHSAS 18002 :2009 " Occupational health and safety management systems. Guidelines for the implementation of OHSAS 18001 :2007" (Figure 4).

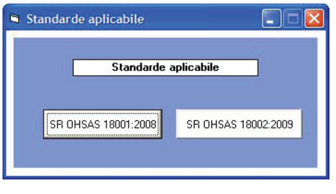

Figure 4. Working window for the visualization of the applicable standards.

2. The area of "**MC-SSM**", is used to visualie the model of the handbook for the management of OHS. It comprises the instructions for the filling in of each clause in this structure. The model of the system document has been produced with the help of hyperlinks which help the browing of the working areas of the document with the view to reading the instructions and to filling in correctly the handbook for the management of the OHS;

3. The area of "**PS-SSM**", allows the acces to the list with the 13 system procedures, making easier the acces to these models, which also include instructions for their filling in. The reading and the filling in of these models are similar to those valid for the above said handbook for the management of the OHS. By taking into consideration the design of the software, the 13 models for the system procedures can be opened/filled in both one by one or randomly, depending on the option of the user (Figure 5):

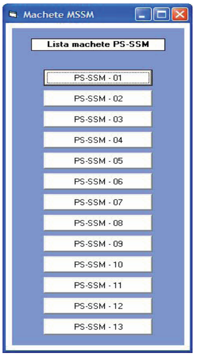

Figure 5. Working window with the list of the models for the 13 system procedures of OHSAS.

4. The area for the "**Verificare cerințe/Verification of requirements**", provides the possibility to verify/self-check the observance of the requirements stated by the SR OHSAS 18001:2008, with the view to getting correct and full system documents (MC-SSM şi PS-SSM). This area comprises the document with the description of the requirements stipulated in the above-said standard. It has been designed based on the same system with hyperlinks that help the visualization of the corresponding requirements.

For providing the safety of information introduced into MSSM.EXE 01, the access to the application and to the models of the system documents is made with the help of a password (Figure 6).

Figure 6. Window with the access password to MSSM.EXE 01 software.

Subsequently, all data and information comprised by the documents are provided protection as it follows:

- during the reading of the standards applicable for the OHS domain, i.e: SR OHSAS 18001:2008 and SR OHSAS 18002:2009, available in pdf format;
- during the reading and typing in the checklist with the requirements for the OHSAS and the models of the system documents (MC-SSM, PS-SSM-01 ÷ PS-SSM-13).

5 CONCULUSION

MSSM.EXE 01 software can provide rapid and accurate system documents in the field of OHS based on the evaluation of the specific features displayed by all the occupational health and safety and manages by the economic agents involved in hazardous and/or toxic activities.

The use of this software is not limited and it needs a minimum configuration, together with the use of the suitable operating system. All data and information which are to be introduced are stipulated by the requirements of the guideline for the implementation of the model for the occupational health and safety integrated management system.

This software is the first version and it allows the updating of information and the upgrading of the application whenever significant changes are being made to its documents.

Subsequently, MSSM.EXE 01 can represent a reliable solution for a rapid and accurate solving of the issues in the field of OHS, more exactly through:

- a well-planned and well-documented approach of OHS issues;
- underilng the domains of responsibilities;
- becoming more familiar with the aspects related to OHS and increasing awareness on the issues raised by the OHS;
- providing a stable, favorable and highly performant climate for the assignet humain resources ;
- improving the OHSAS by quantized results.

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The Effect of Atmospheric Pressure on Methane Release: A Case Study

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ABSTRACT Release of methane in active underground mine rooms is continous with many factors. One important factor is atmospheric pressure. Release of methane changes with changes in atmospheric pressure. It is especially true on new section (advanced galleries).

In this paper a relation between atmospheric pressure and release of methane in active uderground mine rooms is defined for the mine 'Podzemna eksploatacija' RMU 'Banovići'd.d.

1 INTRODUCTION

Methane is the symbol of mining and is the first association on mining danger. Accidents in underground coal mines in the world are mainly connected to methane explosions. Since coals contain methane in the pores and in the slips of deposits, methane is always danger for miners' safety and for planned work operations.

In underground mining rooms methane can be released during all mining operations. The intensity of the release depends on various factors: coal exploitation's dynamics, dimensions of faults and fractures in coal deposits and a value of atmospheric pressure.

In this paper the effects of atmospheric pressure on the release of methane has been studied in the mine "Podzemna eksploatacija".

2 THE RELEASE OF METHANE IN THE MINE "PODZEMNA EKSPLOATACIJA"

The occurrence and release of methane in the mine "Podzemna eksploatacija" are connected to all work operations during coal exploitation. Higher release of methane is present on advanced galleries, near hollows and near faults. Near faults, where is more slips and more pores in coal layer, aggregate quantity of methane.

Methane from the coal deposits and the hollows in the mine "Podzemna eksploatacija" can be released:

- constantly and continuously
- spontaneously and suddenly.

Constant and continuous release of methane is present in the hallows and at the massive coal deposits. The intensity of released methane has almost constant value and depends mostly on the value of atmospheric pressure and coal exploitation's dynamics.

Sudden and spontaneous release of methane is typical for underground mining work near the fault. In these cases methane is released spontaneously and its intensity is directly connected to the dimensions of the pores and slips in the coal deposit and the value of atmospheric pressure.

In the Figure 2 we can see the release of methane from a shot hole 1.25 m deep, near the fault of the deposit. The concentration of methane is to 5%.

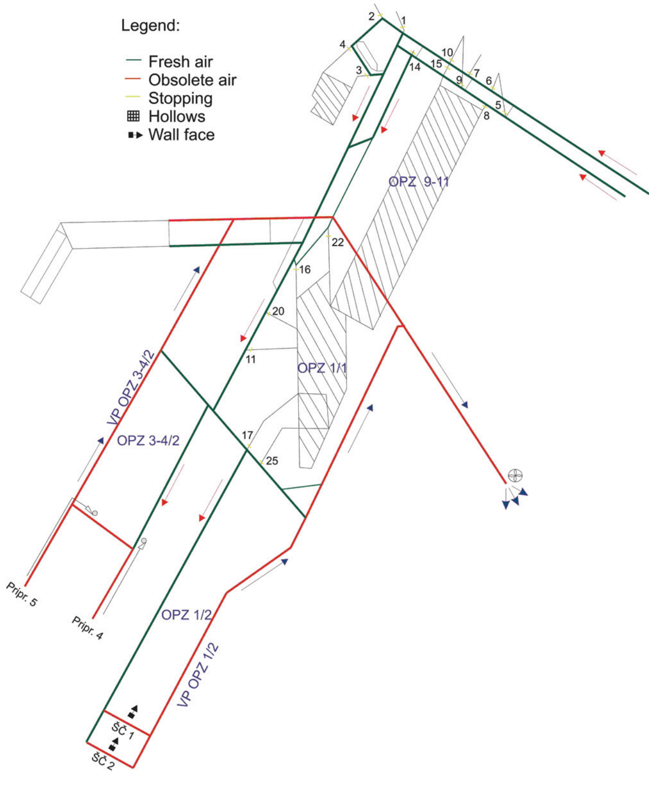

Figure 1. The scheme of ventilation in the mine “Podzemna eksploatacija”.

Figure 2. The release of methane from a shot hole (1.32 % CH_4).

The exploratory drillings which are done in order to characterize coal deposits can cause the sudden release of methane. These cast holes can be 150 m long. In the case when a cast hole comes across a fault and some other sectors rich with methane, more methane is spontaneously released and is often followed with occurrence of underground water. In these cases the concentration of methane is often over 80% (Figure 3).

Figure 3. Sudden release of methane and occurrence of underground water (>80 % CH_4).

3 THE INFLUENCE OF THE ATMOSPHERIC PRESSURE ON THE REALEASE OF METHANE IN THE MINE “PODZEMNA EKSPLOATACIJA”

Methane, which is released during coal exploitation, was formed during the phase of forming coal deposits and carbonization of coal. This way formed methane fills the pores and fragments of the coal deposits. It is under the pressure and has a permanent tendency to move to mine rooms and hollows.

During coal exploitation in the underground mine “Podzemna eksploatacija” it is noticed that a change of atmospheric pressure leads to a change of the level of released methane. The air in the underground mine's rooms is under certain pressure. When the atmospheric pressure falls, the pressure in the mining rooms also falls and methane from the pores and fractures in the coal deposits moves to mining rooms.

The influence of the atmospheric pressure's change on the level of released methane in the mine “Podzemna eksploatacija” has been established by measuring the concentration of methane in the mining rooms, two wall faces in $OPZ_{1/2}$ and advanced galleries in $OPZ_{3\text{-}4/2}$, in the case when atmospheric pressure is variable.

Table 1 gives the measured values of released methane on the wall face in OPZ.

Table 2 gives the measured values of released methane in advanced galleries.

Table 1. Concentration of methane on the wall faces.

Atmospheric pressure (Pa)	Date	Q (m^3/min)			CH_4 (l/min)		
		ŠČ 1	ŠČ 2	VP $OPZ_{1/2}$	ŠČ 1	ŠČ 2	VP $OPZ_{1/2}$
98,100	11.08.2005.	310.0	280.0	592.0	31.0	56.0	177.6
98,200	24.04.2005.	320.0	275.0	600.0	32.0	27.5	120.0
98,400	26.10.2005.	330.0	290.0	622.0	0.0	29.0	124.4
98,500	24.07.2005.	300.0	290.0	592.0	30.0	87.0	177.6
98,600	11.09.2005	320.0	280.0	605.0	32.0	28.0	181.5
98,800	23.05.2005.	310.0	275.0	592.0	31.0	27.5	118.4
98,900	12.06.2005.	290.0	260.0	562.0	58.0	0.0	112.4
99,000	26.08.2005.	320.0	285.0	607.0	0.0	0.0	60.7
99,300	26.09.2005.	324.0	290.0	615.0	0.0	0.0	61.5
99,500	09.10.2005.	319.0	280.0	607.0	0.0	28.0	60.7
99,600	08.05.2005.	320.0	270.0	592.0	32.0	0.0	59.2
99,700	10.04.2005.	320.0	280.0	605.0	0.0	0.0	0.0

Table 2. Concentration of methane in advanced galleries.

Atmospheric pressure (Pa)	Date	Q (m^3/min)			CH_4 (l/min)		
		Prip. 4	Prip. 5	VP $OPZ_{3-4/2}$	Prip. 4	Prip. 5	VP $OPZ_{3-4/2}$
98,100	11.08.2005.	255.0	176.0	546.0	25.5	422.0	819.0
98,200	24.04.2005.	205.0	185.0	602.0	41.0	518.0	782.0
98,400	26.10.2005.	144.0	152.0	567.0	129.6	197.6	623.7
98,500	24.07.2005.	221.0	176.0	532.0	0.0	246.4	425.6
98,600	11.09.2005.	187.0	168.0	525.0	37.4	84.0	420.0
98,800	23.05.2005.	136.0	136.0	592.0	0.0	95.0	414.0
98,900	12.06.2005.	136.0	136.0	581.0	13.6	272.0	406.7
99,000	26.08.2005.	212.0	192.0	546.0	21.2	384.0	436.8
99,300	26.09.2005.	170.0	144.0	574.0	17.0	14.4	229.6
99,500	09.10.2005.	161.0	136.0	560.0	0.0	40.8	224.0
99,600	08.05.2005.	153.0	152.0	592.0	0.0	76.0	118.0
99,700	10.04.2005.	195.0	195.0	595.0	0.0	39.0	59.5

Analysing the measured values, we can conclude that the change of atmospheric pressure effects the concentration of methane in the mining rooms. Lower atmospheric pressure mostly causes higher concentration of methane. This phenomenon is less noticeable on wall faces and the reason for that is bigger working zone and possibility of realising methane from a bigger area.

Higher concentration of methane caused by lower atmospheric pressure is especially present and noticeable on advanced galleries. Any change of atmospheric pressure causes a change in concentration of methane. Variable concentration of methane is mostly present in advanced galleries. It is almost sure that the fall of atmospheric pressure will cause the concentration of released methane in advanced galleries. The reason for this can be found in degasification of new parts of coal deposits.

On the basis of these measurements, to define this phenomenon we used the GRAF4WIN. This computer programme enables definition of the relation between these two valuables with coefficient of determination (R).

In this the diagram corresponds to logaritmic curve

$$y = B \log(x) + A$$

with coefficient of determination R = 0.735162.

Figure 4. Relation between atmospheric pressure and concentration of methane on the wall face.

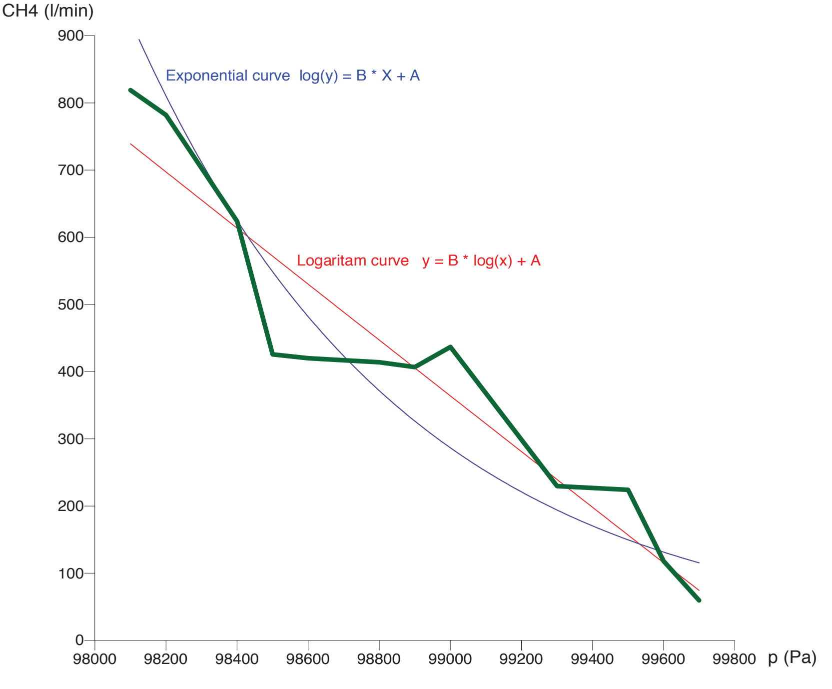

Figure 5. Relation between atmospheric pressure and concentration of methane on the advanced gallery.

In this diagram the best conform with logaritamic curve

$$y=B\ log(x) + A$$

with coefficient of determination R=0.906844 and exponential curve

$$log\ y=B\ x + A$$

with coefficient of determination R=0.850556.

4 CONCLUSION

The release of methane in underground mining rooms is always present. Variable atmospheric pressure causes the changes in concentration of methane. Knowing these effects we are able to prevent mining accidents and improve coal mining safety.

Also, we have possibility foreseen release of methane in case different atmospheric pressure in some parts of the mine.

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Preventing Spontaneous Combustion Using Zinc Chloride Inhibitor at Lonea Mine Unit

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ABSTRACT Spontaneous combustion represents major risk during coal mining. It can trigger fires, with adverse effects both from a material point of view and with human losses. This paper presents the conveyance of zinc chloride sprayed particles into coals and into fissures in the coal back. These particles diminish coal tendency to self-ignition with more then 90% thus diminishing the possible occurrence of spontaneous combustions.

The researches carried out by the experts of INCD – INSEMEX Petrosani and CNH Petrosani have materialized into a method that is being used to diminish coal tendency to self – ignition. The tests carried out on the stand at INCD-INSEMEX Petrosani with coal samples from the Jiu Valley coalfield, have underlined a cease of oxidation before reaching coal self-ignition. This method was tested at Lonea mine based on a working agenda approved by the management of CNH Petrosani – Lonea mine is in its subordination.

1 INTRODUCTION

Coal still is and shall be one of the main power generating sources. Coal has been integrated into the strategies that have been designed and implemented with the view to promoting a sustainable energy development both within the EEC area and in other areas of the world.

In Romania, the sector of the "energetic coal" must be competitive, with a diminution of the production costs. These aspects involve both reorganization and a modernization of mining operations. There has been increased the coal output in the viable mines when implementing highly efficient mining methods, such as coal bed undermining.

Unfortunately, this method amplifies certain risk factors, such as the occurrence of spontaneous combustions.

The present situation (a diminution of the financial support given for mining activities) requires an increased mining efficiency. Consequently, one way to reach this aim is to prevent the occurrence of spontaneous combustions in coal mines; consequently, there diminishes the period during which coal deposits are blocked, we have a continuity in coal mining operations, a diminution of accidents, all mining equipment are being used in good conditions and the expenses for an active, passive or mixed fight against fires are being cut down.

Beside the classical preventive means (mudding, treatment with anti-pyrogenetic substances, treatment with chemical foams and sealing) new engineering methods that use inorganic inhibitors, there have been performed pilot-tests and *in situ* tests.

2 SCIENTIFIC EXPLANATION OF COAL SELF-OXIDATION INHIBITING THE SELF-OXIDATION PROCESS

2.1 Coal Chemical Structure

According to the researches carried out in the field of coals chemical structure, they are considered macromolecule compounds made of compressed aromatic nuclei with side hydrocarbons chains included, which show an increased reactivity with respect to the compressed aromatic nuclei, in compliance with the structures shown in Figures 1, 2 and 3.

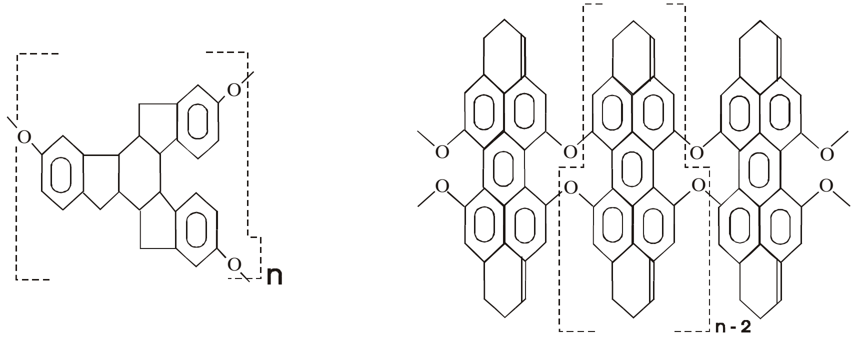

Figure 1-2.

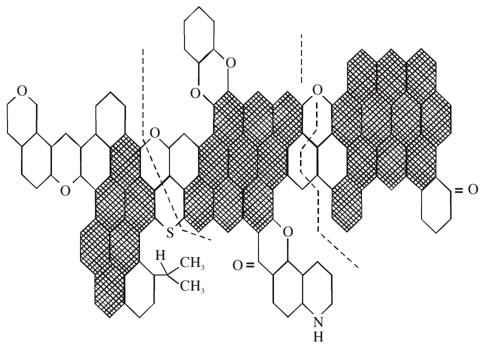

Figure 3.

An increased metamorphism increases the content of aromatic nuclei, to the prejudice of side chains; as a result, there is developed a more arranged structure that comes near the crystalline network of graphite.

2.2 Self-Oxidation of Organic Compounds

Considering coal chemical structure, one can say that coal is a complex organic compound that observes the same rules valid for all the organic substances, so including the one of self-oxidation.

Self-oxidation is a reaction between the organic compounds with the molecular oxygen, in relatively mild conditions of temperature and pressure.

Generally, the reaction is triggered by dimmers or ions of the transient metals and this proves the radicalic character of the process.

2.3 Inhibiting the Self-Oxidation Process

Self-oxidation of hydrocarbons polymers and of other organic compounds is a chained process. For inhibiting this process, one can take the suitable measures either during the stage of chain formation, or during the chain spreading and branching.

The inhibitors that are used during the chain formation may be compounds of screened phenols or sulphurs type.

According to literature, the theory of the ”pyretic-oxidation” and the theory related to the role played by the microelements that are component part of coal corroborated with the theory of oxidation have led to the conclusion that the substances that are part of the zinc chloride may have inhibiting effects over the coal self-oxidation.

Several authors consider that the use of the zinc chloride has led to:

a) The decreasing of the temperature of the treated sample, compared to the untreated sample, during the determination of the coal tendency to self-ignition by the method of thermal oxidation with gaseous oxygen.

b) The decreasing of the temperature gradient with a 20 minutes period from the start of the oxidation, at the end of the self-oxidation, as well all through the oxidation process, compared to the untreated sample. It is important the decreasing of the temperature gradient that occurs at the beginning of the process when heat accumulation occurs and may trigger later the self-ignition.

For all the above-mentioned situations, the output is expressed by the following equations:

$$\eta_1 = 100 - \frac{\Delta T_f \text{ treated sample}}{\Delta T_f \text{ untreated sample}} \cdot 100 \quad (1)$$

$$\eta_2 = 100 - \frac{\Delta T / 20 \text{ minutes heating of the untreated sample}}{\Delta T / 20 \text{ minutes heating of the treated sample}} \cdot 100 \quad (2)$$

These measurements have led to the conclusion that the inhibitor that is part of the zinc chloride shall diminish coal tendency to self-ignition with around 87%.

2 ACTUALLY SITUATION OF EXPERIMENTAL WORKING FROM EM LONEA

The experimental working choused for the monitoring of the auto-ignition risk of coal thickness is localized into geological block VII on the bed 3, having the identification code in coal face no. 74.

The coal is exploited with exploitation method of undermined bar behind the front line which replaces the old exploitation method SCRI (coal crumbling and of the environmental rocks). In Figure 4 is presented the localization of monitored experimental working to assure an informational volume concerning auto-ignition risk of the coal bed 3.

Coal spontaneous combustion is directly connected, alongside the exploitation method, by the used aeration. For this experimental working the workplace aeration is made through aeration under general depression of the mine. In this way is assured fresh air to directional pre -face no. 74, roof cote 325 m, but the evacuation of vitiated air through directional gallery 400 m which makes the connection to surface through material well Jieț.

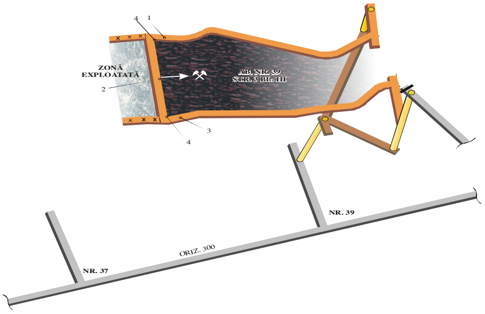

Figure 4. Location of the experimental front face.

3 ASPECTS CONCERNING THE AUTO-IGNITION RISK OF THE COAL

Spontaneous combustion is due to coal auto-ignition and is determined by a series of factors which contributes at the appearance of endogen fire. From these factors we remind objectives factors (physical – chemical property of the coal, petroleum-graphic structure and geological conditions of the place where is found the coal thickness) and subjective factors (those given by the exploitation mode and by aeration). To be able to implement the prevention solution of spontaneous combustion is necessary the monitoring of the phenomena of evolution – involution of these. Monitoring the evolution-involution of spontaneous combustion may be considered an indirect preventive method. This indirect method shows the moment when it has to take action with a direct method (i.e. mudding, inhibiting sprayed particles, injection into massif of water and inhibiting substances under pressure) so as to stop the evolution and direct towards an involution of the spontaneous combustion. The monitoring is made with the help of fire parameters. In practice is known many parameters from which in our country is used the followings:

1) Graham parameter (R_1), given by the relation:

$$R_1 = \frac{+\Delta CO}{-\Delta O_2} \cdot 100$$

where:

+ ΔCO – the growth of the carbon monoxide in rapport with reference level,

- ΔO_2 – the diminution of oxygen concentration in rapport with normal value.

The Graham parameter is determined only in the case the diminution of the oxygen content is more than ≥ 0.2 %.

This parameter is the most used and indicates with precision the beginning of spontaneous combustion process.

2) Respiration parameter (R_2), given by the relation:

$$R_2 = \frac{+\Delta CO_2}{-\Delta O_2} \cdot 100$$

where:

+ ΔCO_2 – the growth of carbon dioxide in rapport with the reference level,

- ΔO_2 – the diminution of oxygen concentration in rapport with normal value.

The respiration parameter is determined only in the cases in which the oxygen content diminution is more than 0.5 %.

The parameter R_2 is a valuable parameter, but because the underground activities (personal presence, shooting works, functioning of the engines with internal combustion) decrease the forecast exactitude of spontaneous combustions phases.

3) The Ethylene (C_2H_4) is an unsaturated hydrocarbon which appears only in the coal oxidation process. This appears at the temperature of 100^0C (on the focus).

4) Acetylene (C_2H_2) is also an unsaturated hydrocarbon which is natural produced only the coal oxidation process. This hydrocarbon appears in the temperature domain between 200^0C and 250^0C.

The introduction of these parameters leaded to the growth of forecast exactitude of spontaneous combustion phase, but necessitates qualified personal and laboratory dotted with performance equipment (gas chromatograph). For the reducing of the determination number of C_2H_2 and C_2H_4 is recommended like these to be sampled and determined only at direct determination of carbon monoxide at higher concentrations.

At national level for the current determination was generalized the application of all 4 fire parameters through us obtain high exactitude in the spontaneous combustion forecast.

The connection between Graham fire index and coal temperature in the self-oxidation process is shown in Figure 5.

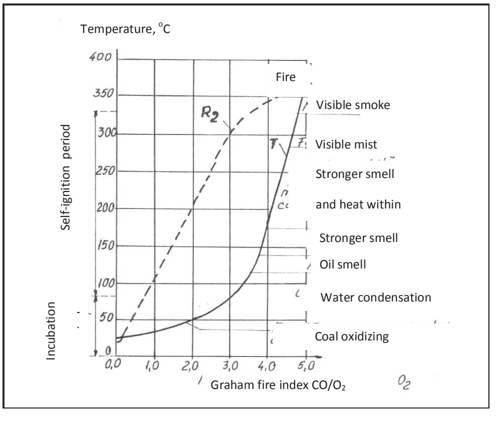

Figure 5.

The model developed by the Prof. Bystron is based by the correlation between the variation of monoxide carbon concentration in time and the temperature developed during the process.

4 PREVENTIVE ENGINEERING METHOD WITH INHIBITING TYPE SPRAYED PARTICLES

a) Description of the installation used to produce the inhibiting sprayed substance

The prevention of endogenous fires during hard coal mining relies on very fine (micron – sized particles) of the inhibiting solution spread all through the goaf and at the working face, depending on the location of the installation and in compliance with the air flow.

The installation used to produce sprayed particles (fig. 6) is made of a 200 ml tank (1), an air-water spraying device – (2) and the connection hoses (3) to the compressed air mains at the working place and to the tank with the inhibiting substance (5). A tap (4) is mounted on the compressed air hose of the spraying device.

The spraying device for spraying the inhibiting substance has got the following parameters:
- the working pressure: 0,3 – 0,6 MPa;
- consumption of compressed air: 0,8 – 1,05 m^3/min;
- consumption of inhibiting solution: 0,8 – 1 l/min;
- sprayed solution ratio: 90 – 100 %.

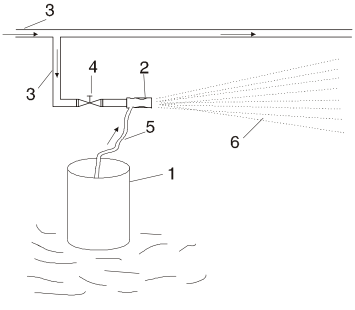

Figure 6.

b) Treatment method with sprayed particles of inhibiting substance

Due to the inhibiting characteristics displayed by the zinc chloride during the coal oxidation and self-ignition process, coal oxidation and self-ignition process, a suitable treatment with sprayed particles from the „zinc chloride" group shall diminish the risk of spontaneous combustions.

To treat adequately a goaf, this spraying device shall be located in the cross drift of the blasted pre-crushing raise. The sprayed device shall also be mounted in front of the holes in goafs where high concentrations of CO were previously detected (over 0,1 % vol.).

The spraying device shall be placed at a height of 1 -1,5 m from the mine floor, being orientated towards the area that is to be treated with the help of sprayed particles and shall operate in every point for approximately one hour.

5 MONITORING THE EVOLUTION-INVOLUTION FO SPONTANEOUS COMBUSTION AT THE LONGWALL WITH UNDERMINED BED NO. 74 LAYER 3 BLOCK VII AT LONEA MINING UNIT AS A RESULT OF USING THE PREVENTIVE METHOD WITH INHIBITING SPRAYED PARTICLES OF ZINC CHLORIDE SOLUTION

The measurement places for gas concentrations which are into the parameter fire components were:
- roof pre-face at appreciatively 2 ÷ 5 m from the intersection with the coal face;
- on the working plane at the appreciatively middle and in different points where was signalized the presence of the carbon monoxide;
- shelter working at appreciatively 2 ÷ 5 m from intersection;
- In other places from neighbored mining works (dams at the superior sub floors).

The surveillance duration of evolution – involution of spontaneous combustion at the undermined coal face 74 from Lonea Mining Unit was 8 months. Considering the great number of determinations made during monitoring in the Table 1 is presented only the values obtained on the working plane.

Graphic transposition of Graham parameter in function of time indicates a graphic with a shape close to Bystron model.

Table 1. Monitoring parameters.

Time	Place	Gas concentration						Temp ^{0}C	Graham Index	Resp. Index	Obs.
		O_2 (%)	CO_2 (%)	CH_4 (%)	CO (%)	C_2H_4 (%)	C_2H_2 (%)				
0	1	2	3	4	5	6	7	8	9	10	11
02. 2008	Coal face	20,4	0,25	0,2	0,0004	-	-	21	0,07	46,3	Into the exploited space aprox. 0.5 m after net
		19,9	0,25	0,01	0,004	0,00045	0,000062	22	0,386	24,0	Face
		19,9	0,25	0,3	0,0007	-	-	22	0,08	24,0	At the working middle 0.5m after net, in exploited space
		20,1	0,2	0,3	0,0008	-	-	22	0,10	23,8	Profile and face
03. 2008		17,4	1,2	0,9	0,0082	0,0002	0,00002	22	0,23	33,9	Face
		20,4	0,15	0,3	0,0006	-	-	23	0,11	27,7	
		19,6	0,25	0,5	0,0022	-	-	23	0,16	18,6	
		20,2	0,2	0,35	0,0008	-	-	23	0,11	27,0	
04. 2008		16,6	2,7	1,1	0,0660	0,000064	0,000010	22	1,53	62,8	
		19,4	0,25	0,25	0,255	-	-	20	1,700	16,6	Face
		19,7	0,3	0,15	0,0130	-	-	20	1,000	23,0	Face
		20,0	0,35	0,2	0,0066	-	-	20	0,702	37,2	Profile
		19,7	0,25	0,2	0,0087	0,00013	0,000010	20	0,725	20,8	Face
		19,9	0,25	0,3	0,0080	-	-	20	0,800	25,0	Profile
		19,9	0,25	0,3	0,0084	-	-	20	0,840	25,0	Face
05. 2008	Coal face	17,6	1,7	1,1	0,0335	0,00012	0,000018	22	1,015	51,5	face; specific smell of hydrocarbons
		19,8	0,35	0,25	0,0089	-	-	23	0,809	31,8	Profile
		19,8	0,35	0,35	0,0092	0,000008	0,0000042	23	0,836	31,8	Face
		19,6	0,3	0,45	0,0093	-	-	23	0,715	23,0	Profile
		19,6	0,33	0,5	0,0093	-	-	23	0,715	23,4	Face
06. 2008	Middle of the coal face	18,6	1,1	0,7	0,0104	-	-	21	0,452	47,8	Profile
		18,0	1,3	0,8	0,0390	0,00014	0,00002	21	1,344	44,8	Face; specific smell of hydrocarbons
		20,3	0,12	0,2	0,0100	-	-	20	1,563	18,7	Profile
		19,8	0,3	0,2	0,0100	-	-	20	0,909	27,3	Face
		19,9	0,25	0,3	0,0080	-	-	20	0,800	25,0	Profile
		19,9	0,25	0,3	0,0084	-	-	20	0,840	25,0	Face; specific smell of hydrocarbons
07. 2008		19,3	0,24	0,35	0,0110	-	-	19	0,688	18,5	In excavated space at 0,5 m depth
		14,9	0,045	0,75	0,0960	0,0001	0,000014	24	1,6	75	In excavated space at 0,5 m depth
		18,8	1,00	0,4	0,0300	0,0025	0,000023	25	1,429	47,6	In excavated space at 0,5 m depth
		20,2	0,15	0,15	0,0038	-	-	21	0,513	20,3	Profile / 50
		20,0	0,15	0,15	0,0040	-	-	21	0,425	16	Face / 50
08. 2008		20,0	0,1	0,1	0,0008	-	-	19	0,08	10,6	Profile
		19,7	0,25	0,85	0,0020	-	-	19	0,16	20,2	Face
		20,2	0,1	0,15	0,0019	-	-	19	0,26	13,5	Profile
		19,6	0,3	0,2	0,0025	-	-	19	0,19	22,4	Face
		20,1	0,25	0,15	0,0020	-	-	19	0,24	29,8	Profile
		19,3	0,3	0,2	0,0020	-	-	19	0,12	18,3	Face
09. 2008		20,5	0,05	0,1	0,0004	-	-	18	0,09	-	Profile and Face / 5
		20,2	0,15	0,15	0,0019	-	-	20	0,26	20,3	Profile / 35
		20,4	0,1	0,1	0,0010	-	-	20	0,18	18,5	Face / 35
		20,4	0,2	0,15	0,0015	-	-	23	0,28	37	Profile / 50
		20,4	0,2	0,15	0,0010	-	-	23	0,18	37	Face / 50

6 CONCLUSIONS

– Spontaneous combustion displays a high risk of occurrence during hard coal mining;
– The evaluation of evolution-involution of spontaneous combustion is made based on fire indices;
– INCD INSEMEX has developed several engineering methods for preventing spontaneous combustion, among which the highest efficiency is displayed by the preventive technology that uses sprayed particles with zinc chloride-type inhibitor;
– This method was used at Lonea Mine, on the stope with undermined coal bed no. 74;
– The in situ tests confirm the high efficiency related to the diminished tendency to self-ignition of hard-coal; there was used a watery solution of 0.5 % zinc chloride that led to a diminution of spontaneous combustion;
– The experts at INCD INSEMEX Petroşani propose a wide use of this method to prevent spontaneous combustions, at first in one mine.

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Optimizing the Location of Structures Used for Regulating and Driving Air

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ABSTRACT Lonea's current ventilation network stretches vertically all along seven active levels and is quite complex; its intricacy is increased by the high number of structures used for regulating and driving of air in underground.

The measurements that have been carried out in situ have underlined a wrong location of the ventilation structures and the use of three main ventilation stations necessary for providing health and safety of persons working in underground.

The first optimizing stage consisted in the balancing of the ventilation network; there followed the settlement of the network by removing certain ventilation structures. There has been used an expert software during this last stage; consequently, we have got a steady distribution of the air outputs.

The paper includes a detailed presentation of the stages, the results that have been gained, together with the measures to be implemented for their materialization.

1 INTRODUCTION

The current general ventilation of Lonea mine is upwardly, under the influence of the general depression created by-the three main ventilation stations:

- The main ventilation station at Level 840 equipped with two BCMM fans;
- Valea Arsului main ventilation station equipped with two BCMM fans;
- Jiet ventilation Shaft main ventilation station equipped with two VOD-2,1 fans;

At present the following hardcoal beds are being mined at Lonea Mine:

- bed no. 3 at the level 300 in block III;
- bed no. 3 at the level 300 in block II;
- bed no. 3 at the level 400 in block VII;

The coal deposits in the above-said blocks are being mined with the help of framework methods approved for the Jiu Valley coalfield, i.e. front stope with undermined coal bed.

Simultaneously with the mining operations, there have been stipulated 17 preparatory works and 4 opening works.

In accordance with the structure of the ventilation network, the operation of the three main ventilation stations produces three main ventilation circuits and two secondary ventilation circuits (see the 3D (x, y, z) scheme – Fig.1).

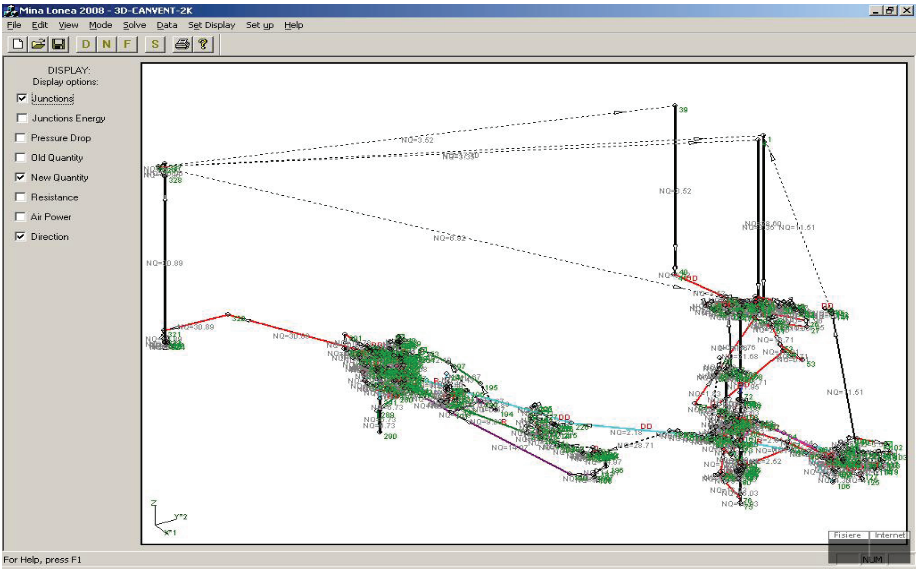

Figure 1. Ventilation network.

2 OPTIMIZING THE VENTILATION NETWORK OF LONEA MINE

An analysis of the ventilation network shows that it is quite complex; this intricacy is increased by the large number of structures used for air driving and regulation, more exactly 120, gathered into 57 groups of ventilation structures.

The main ventilation stations are located as it follows, if the main intakes are taken into consideration (New Skip Shaft, Old skip Shaft, Auxiliary Shaft):

-Centrally spaced for the case of Valea Arsului main ventilation station;

-Diagonally, for Jieț Shaft and Level 840 main ventilation station.

The ventilation network extends vertically along seven active main levels as it follows: level 615; 500; 460; 400; 350; 300 and 250 and level 200 which is in the opening process.

The number of the ventilation stations, the vertical and horizontal extension, the multiple connection among them have identified 339 functions and 424 branches. Subsequently, this is the most complex ventilation network that has ever been analysed up to now.

The air short circuiting with surface displays very high values at the main ventilation stations.

As a result of the document analysis and of in situ measurement, there have been discovered certain unsuitable locations of the ventilation structures.

The first stage during the optimization of the ventilation network of the mine comprised the settlement of the network on the currently existing structure.

The virtual settlement of the ventilation network was made with the help of an expert software 3D Canvent 2K. There followed an implementation of the results gained as a result of the virtual settlement, i.e. there were removed 41 groups of ventilation structures which comprised almost 90 individual structures. This removal gave a stable distribution of air flows, both from the point af view of magnitude and of flowing direction.

Of a total of 16 ventilation structures located within the ventilation network of the mine, 11 maintained both their location and the specific aerodynamic parameters.

In order to preserve the necessary air flow rates inside the stopes, four ventilation structures maintained their location but suffered changes in the aerodynamic parameters. There was also necessary the location of a new ventilation structure an the branch 256-257.

Figure 2 shows the alterations made for this simulation.

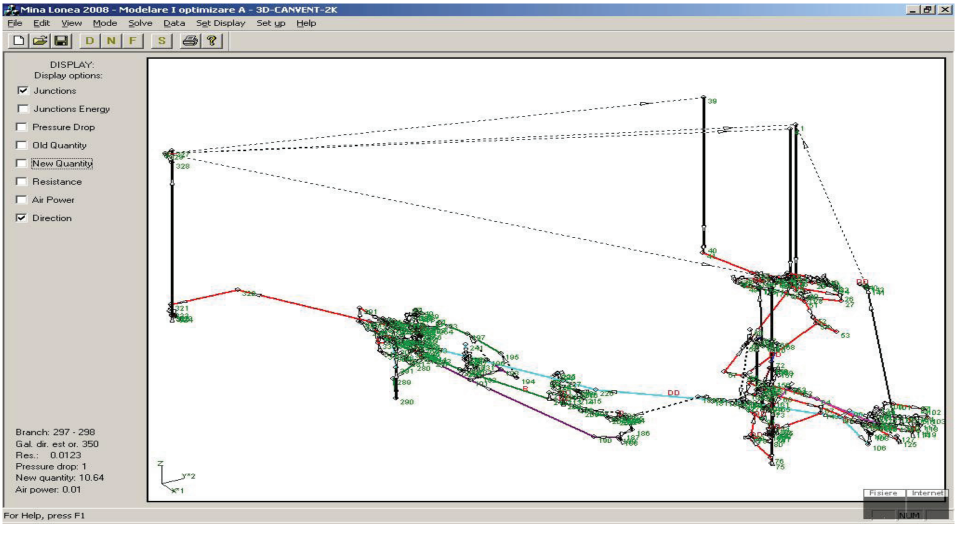

Figure 2. Altered ventilation network.

Subsequently, the following results have been gained:

- The intake flow rate reached 42.92 m^3/s (2575 $m^3/min.$) along the level 400 (New Skip Shaft, Old skip Shaft, Auxiliary Shaft), i.e. an increase of the flow rate with 21.04% compared to the present values (35.46 m^3/s – 2128 $m^3/min.$);
- The total air intake reaches 18,68 m^3/s (1121 m^3/min) in Valea Arsului Shaft which is lower than the current total air intake (21.67 m^3/s – 1300 $m^3/min.$) with 13.8%;
- The air flow rate increased with up to 10% in the active face stopes;
- The air flowe increased with approximately 4.8% (from 28.66 m^3/s to 30.05 m^3/s) along the incline that connects the level 350 and the level 300;
- The air flow rate increased from 4,71 m^3/s to 5.04 m^3/s along the main air return way of the face with undermined coal bed no. 39, bed no. 3, block III;
- The air flow rate diminished from 5.62 m^3/s to 4.18 m^3/s (i.e. approximately 25.62%) along the main air-return way of the face with undermined coal bed no. 75-73, bed no. 3, block VII;
- The air flow rate diminished from 5.87 m^3/s to 5.54 m^3/s (i.e. approximately 5.62%) along the main air-return way of the faces no. 36-34 A and B, bed no. 3, block III;
- An air flow rate of 61.6 m^3/s is being discharged from the whole mine, that means an increase with 7.8% compared to the present situation (57.13 m^3/s) , divided as it follows:
- Level 400 – Jiet Shaft – 32.89 m^3/s;
- Valea Arsului ventilation raise – 15.79 m^3/s;
- Cota 840 ventilation raise – 12.92 m^3/s;
- The air flow increased from 95.34 m^3/s up to 98.57 m^3/s (i.e. approximately 3.23 m^3/s).

3 MEASURES PROPOSED TO IMPROVE THE VENTILATION INSIDE THE MINE

One of the most important measures necessary for the optimization of the general ventilation of Lonea Mine is the removal of unsuitably located ventilation structures and the erection of new structures in the above-said locations, together with the change of the aerodynamic parameters on certain ventilation structures currently in place.

In order to provide a judicious operation of the main ventilation installations, the following operations should be carried out:

- The sealing of the house of the Ventilation Shaft and of the access way to the shaft from the surface up to the level up to the point where the air flow shorcircuited with the surface reaches 10% maximum of the air flow of the mine (for the case of Jiet main ventilation station);
- The sealing of connection to surface with the help of back-up fans (Valea Arsului ventilation raise and Cota 840 ventilation raise main ventilation stations);
- The sealing of connection to surface of the ventilation raises so as the flow shorcircuited with the surface should not exceed 5% of the mine air flow rate;
- Providing an accurate maintenance in order to provide a reliable operation of the fan and of the motor. This measure is very useful, especially for Jiet main ventilation station, as it shall become the only main ventilation network of Lonea mine.

With the view to providing a dynamic stability of the ventilation network, it is recommended that Jiet ventilation Shaft should be used exclusive by for the discharge of the return air from the underground.

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Numerical Prediction of Fan Parameters in Mine Air Controlled Recirculation Systems

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ABSTRACT Providing both a safe and economic solution for certain environmental problems, controlled recirculation of mine air was investigated, on theoretical basis which are clearly established. This paper is a synthesis of the results of analytical studies and numerical simulation carried out in view of assessing recirculation fraction variations effect on booster recirculation fans parameters. Several conditions were generated on computer, for inline and crosscut fan sitting, by varying the recirculation factor and aerodynamic resistance of airways. On comparative basis, feasibility conditions which allow the use of a recirculatory ventilation system were established.

1 INTRODUCTION

As mine venti1ation systems become more complex, it is increasingly difficult to supply fresh air to the working face. High percentages of the fresh air entering a mine do not reach the working face because of short circuits through stoppings and doors, the exiting mine without being utilized. When ventilation limits are reached, the only option presently available to most Romanian mines is to sink a new shaft - an extremely expensive undertaking. Reusing or recirculating a percentage of ventilating airflow may make it possible for some mines to remain operational while maintaining a safe environment for workers.

The possibility of recirculating mine air was considered until recently as an unsound ventilation practice. It had only become a practical proposition because of the development of continuous monitors which safeguard the quality of fresh, recirculated and mixed airstreams that are circulating in the working areas.

Controlled recirculation is still not allowed legally in Romania for primary ventilation systems, but based on the well documented existing theory, at the University of Petrosani (Moraru, 1999), research was carried out in view of establishing the feasibility criteria of controlled recirculation of mine air.

The recirculation booster fan position and the size of the recirculation circuit are two of the most important parameters in characterizing the effects of a recirculation system. There exist two main possible sittings for a recirculation fan. Inline, located in the recirculation intake or return, or positioned in the recirculation crosscut. The inline sittings of the same fan produce the same airflow but different pressure distributions around the district (Longson et al, 1985).

The physical length of the recirculation circuit, including its projected length as development takes place, will be important in determining the type of recirculation system to employ (Longson et al, 1987) and the operating parameters of recirculation booster fans.

This paper summarize some results achieved by computer simulations conducted for emphasizing the effects of various recirculation fractions and increases of aerodynamic resistance of recirculation circuit schemes. The rate of gas emission in the district was maintained constant.

2 INLINE RECIRCULATION SYSTEM

Shown in figure 1, such a system involves the use of a booster recirculation fan which handles the total amount of air required in the face and a regulator located in branch 4 for controlling the airflow in the circuit.

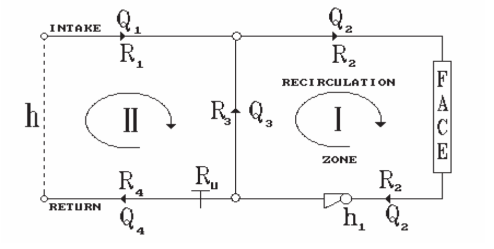

Figure 1. Inline recirculation system.

In this configuration, the operation of the booster fan tends to increase the total airflow, as a consequence of the higher airflow handled by the main fain and diminished air losses through existing leakage paths (Calizaya et al, 1992).

The workings out bye the recirculation crosscut were replaced by equivalent resistance branches and the pressure drop between intake and return airways were considered as constant.

Assessment of recirculation fan's and the regulating device's size providing the airflows required consists in applying Kirchhoff's laws, as it follows:

$$Q_1 + Q_3 = Q_2 \quad (1)$$

$$Q_3 = F \cdot Q_2 \quad (2)$$

$$Q_2 = \frac{1}{1-F} \cdot Q_1 \quad (3)$$

$$Q_3 = \frac{F}{1-F} \cdot Q_1 \quad (4)$$

Applying Kirchhoff's second law, we obtain:

Mesh I: $h_1 = R_2 \cdot Q_2^2 + R_3 \cdot Q_3^2$ (5)

Mesh II: $R_u = \dfrac{h + R_3 \cdot Q_3^2 - R_1 \cdot Q_1^2}{Q_4^2} - R_4$ (6)

The power consumption of the fan is:

$$N_1 = h_1 \cdot \frac{Q_2}{\eta_1 \cdot 1000} \quad (7)$$

where: R_i-airway resistance coefficient, kg/m^7 (i=1, 2, 3, 4); Q_1-fresh air quantity, m^3/s; Q_2-mixed air quantity, m^3/s; Q_3- recirculated air quantity, m^3/s; Q_4-main return air quantity, m^3/s; F-recirculation factor; η_1-total efficiency of booster recirculation fan; and N_1-air power consumption, kW.

By replacing expressions (1), (2), (3) and (4) in equations (4), (5) and (6), it follows:

$$h_1 = \left(\frac{1}{1-F} \cdot Q_1\right)^2 \cdot \left(R_2 + F^2 \cdot R_3\right) \quad (8)$$

$$R_u = \frac{h}{Q_1^2} + R_3 \cdot \left(\frac{F}{1-F}\right)^2 - R_1 - R_4 \quad (9)$$

$$N_1 = \left(\frac{1}{1-F} \cdot Q_1\right)^3 \cdot \left(R_2 + F^2 \cdot R_3\right) \cdot \frac{1}{\eta_1 \cdot 1000} \quad (10)$$

Based on the previous relationships, dependence between the booster recirculation fan parameters and circuit resistance coefficient was simulated on computer for different recirculation factors. For the numerical study the following values were attributed: R_1=0,2 kg/m^7, R_2=0,2 to 1 kg/m^7, R_3=0,2 kg/m^7, R_4=0,25 kg/m^7, h=1350 Pa, η_1=75 %, F=0 to 0,7 step 0,1.

The resistance of the regulator for increasing resistance coefficient of the recirculation circuit was also established.

In Figures 2, 3 and 4 are, respectively, these dependencies are graphically plotted for the theoretical case study.

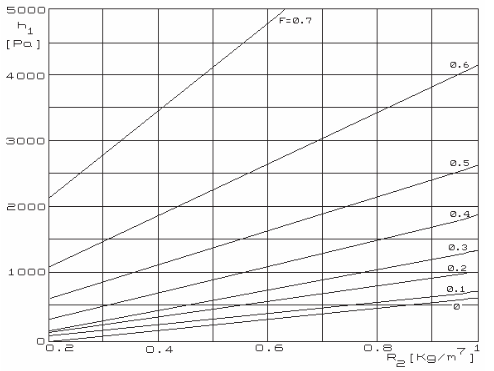

Figure 2.

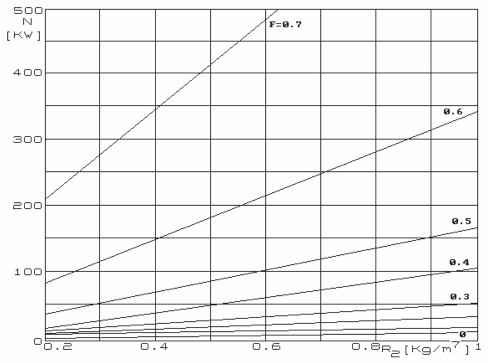

Figure 3.

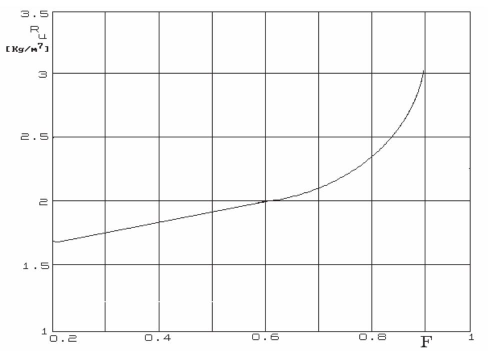

Figure 4.

3 CROSSCUT RECIRCULATION SYSTEM

This kind of controlled recirculation system involves the use of two booster fans: the first one sited in the recirculation crosscut, handling only the recirculated amount of air, and the second one in the main return, compensating the pressure loss in the main return airway and, consequently, maintaining a constant fresh airflow entering the district (Q_1).

Being settled these main requirements for this type of recirculation system, his efficiency highly depends on providing a correct interconnection of the two booster fans in view of achieving a preset recirculation factor without decreasing the through-flow in the district.

This kind of controlled recirculation system is diagrammatically represented in Figure 5.

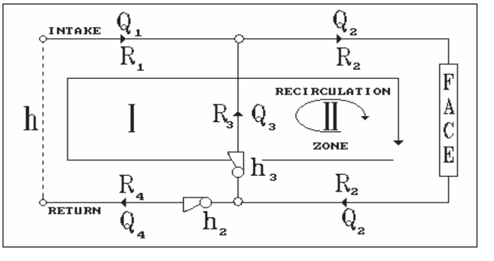

Figure 5. Crosscut recirculation system.

Maintaining - for comparison reasons - identical values of the basic parameters and based on a similar approach as for the inline system, the following equations were obtained:

$$h_2 = (R_1 + R_4) \cdot Q_1^2 + + R_2 \cdot \left(\frac{1}{1-F} \cdot Q_1 \right)^2 - h \tag{11}$$

$$N_2 = h_2 \cdot \frac{Q_1}{\eta_2 \cdot 1000} \tag{12}$$

$$h_3 = \left(\frac{1}{1-F} \cdot Q_1 \right)^2 \cdot (R_2 + F^2 \cdot R_3) \tag{13}$$

$$N_3 = h_3 \cdot \frac{F}{1-F} \cdot \frac{Q_1}{\eta_3 \cdot 1000}$$

(14)

where: index 2 and 3 relates to pressures „h" and air power consumptions „N" indicates the fan which parameters are computed.

In Figures 6, 7 and 8 are plotted the dependences expressed by equations (11), (12) and (14).

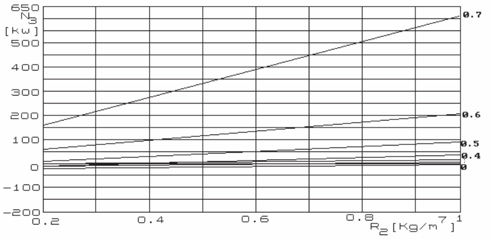

Figure 6.

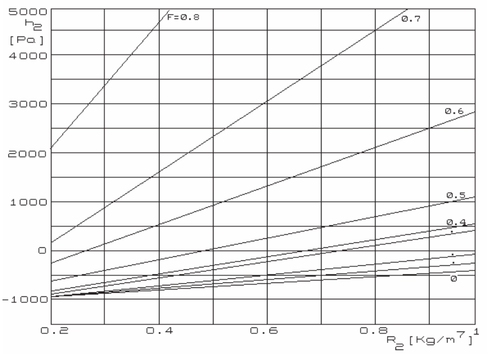

Figure 7.

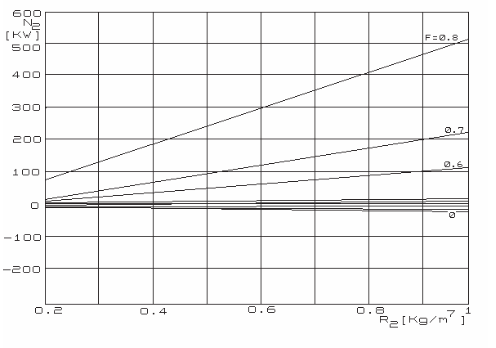

Figure 8.

4 CONCLUSIONS

Establishing the optimal sitting of fans and the recirculation circuit's size involves measurements to be done in view of determining the fresh air quantity available, gas concentrations in the existing conventional ventilation system, pressure losses and computation of aerodynamic resistance coefficients and gas (CH_4 or CO_2) emission rates.

Based on these data and using simple relationships and diagrams the feasibility of controlled recirculation can be assessed.

Whatever the type of recirculation system, the assessment of recirculation fans parameters represents a major issue. Taking into account the development in time of the mine workings configuration, the variation of these parameters were analyzed with respect of grown aerodynamic resistance coefficients of the circuit, for several recirculation factors.

The use of such diagrams, easily obtained by numerical simulation, allows not only to establish the feasibility of controlled recirculation, but also to carry out previsional studies regarding the system's behavior when the exploitation zone expands. It can be, in this manner, established the appropriate moment for displacing the recirculation fan into another point of the system or for regulating his operational parameters.

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Management of the Ventilation Network of Paroşeni Mine During Accidental Stoppage of the Main Ventilation Station

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ABSTRACT The settlement of the ventilation network in a mine allows to establish accurately its ventilation limits, as well to simulate possible technical situations, including a damage that can occur during normal operation.

The accidental stoppage of the main ventilation network represents one of the least favorable situation that can occur during operations in underground, representing a risk for the health and safety of workers.

The use of modern IT together with the Canadian expert software CANVENT allows a rapid identification of the best solution for mine ventilation in the newly created conditions.

1 INTRODUCTION

Paroşeni mine is located in the western part of the Jiu Valley coalfield.

The mine has an ascending ventilation, under the influence of depression generated by the main ventilation station no. 18 - VOD 3.0. The ventilation is accomplished along five levels: 250, 300, 360, 425, 575. The fresh air penetrates through 4 open shafts (Auxiliary Shaft no. 1, Auxiliary Shaft no. 2, Shaft with Skip and the Eastern Shaft) and is discharged through the blind shaft towards the main ventilation station no. 18, equipped with two VOD 3.0 fans.

At present, Paroşeni mine uses longwall mining and shortwall mining. These mining methods are used based on framework methods, specific to thick bed of low and average incline. At present, coal is mined at four panels at Paroşeni mine.

2 VENTILATION NETWORK

In compliance with the annual project for general ventilation, the ventilation network of the mine comprises three main ventilation circuits:

a) the ventilation circuit of level 250, with three subcircuits;

b) the ventilation circuit of level 300, with one subcircuit;

c) the ventilation circuit of level 360, with one subcircuit.

3 SETTLING THE VENTILATION NETWORK OF THE MINE

The Canadian expert software has been used to settle the ventilation network of the mine.

According to the analysis of information gained from topo maps, there have been identified 171 junctions and 216 branches.

After measuring depressions in situ on each branch of the ventilation network, the values were processed in the laboratory and then they were introduced in the database of CANVENT.

The settlement of the ventilation network made possible to get the best possible division of air flows along each branch – Figure 1.

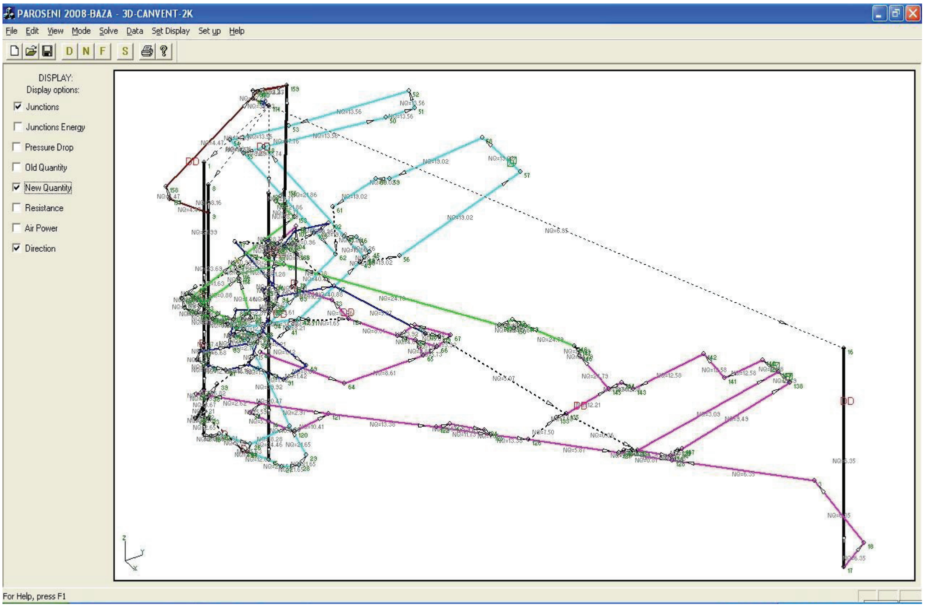

Figure 1.Ventilation network.

If we take into consideration this division, the following air flow rates have been obtained for each level:

-level 250: 39.92 m^3/s (2,395 m^3/min);

-level 300: 36.55 m^3/s (2,193 m^3/min);

-level 360: 4.11 m^3/s (247 m^3/min);

-level 425: 7.19 m^3/s (431 m^3/min);

-level 575: 4.46 m^3/s (268 m^3/min).

The depression created by the main ventilation station discharges an air flow at a rate of 92.23 m^3/s (5,534 m^3/min).

The newly modeled and settled ventilation network of Paroşeni mine saw 4 simulations:

1.The settlement of the ventilation network when modifying the discharge route of poisonous air along the head incline at panel no. 3, bed 3, block VI, level 300, including the related ventilation structures;

2.The settlement of the ventilation network in case of opening a group made of 3 ventilation doors on level 250, at the bottom side of the ventilation raise 250 - 360;

3.The settlement of the ventilation network in case of opening a group made of 3 ventilation doors located at the bottom side of the blind shaft 360 - 575;

4.The settlement of the ventilation network in case of opening a group made of 2 ventilation doors from the gallery no. 4 that contacts the western directional gallery, the level 250 and the conjugated directional ventilation gallery, level 250.

4 MANAGEMENT OF INCIDENTS

The ventilation network of Paroşeni mine is ventilated with the help of one main ventilation station equipped with two VOD 3.0 fans. The fans operate alternatively: one month the fan no. 1, followed by the fan no. 2 the following month, a.s.o.

At the beginning of September 2008, fan no. 1 broke suddenly. At high speed, a blade of the rotor broke the housing of the fan and destroyed the ceiling of the housing. Consequently, the ventilation network of the mine was left unventilated and there appeared the risk of explosive atmospheres in underground.

The first measure taken by the accident committee was to take out the soonest possible the working personnel from underground and to start the fan no. 2.

By using only one fan, the main ventilation station couldn't meet the legal provisions that state the need of two fully operational fans at one main ventilation station: one in operation and one in stand-by.

Consequently, it was necessary to identify another solution: this solution involved the activation of another ventilation station in conservation and located at the Eastern Ventilation Shaft. This station had two VOKD 2.4 fans.

The main problem that arose was that nobody knew whether Eastern Shaft VOKD 2.4 ventilation station could deliver the air flow necessary inside the mine and the actual rearrangement of the mine ventilation network with the discharge of the poisonous air in the Eastern Ventilation Shaft.

To settle these problems, the managing board of CNH Petroşani and of Paroşeni mine asked for the support of INSEMEX Petroşani.

The expert team of the laboratory for industrial ventilation, equipped with all the necessary tools (technical support and CANVENT software) arrived the soonest possible at Paroşeni mine, where we performed a technical simulation for the case of accidents.

The simulation of this situation included several stages:

-The stoppage of the main ventilation station VOD 3.0.

-The start of the main ventilation station at the Eastern Shaft VOKD 2.4. After the start of the new ventilation station, the allocation of air flow rates in underground became chaotic - Figure 2.

-The re-arrangement of the underground ventilation network with the view to producing an optimal allocation of air flows in each branch.

The ventilation network was re-arranged and balanced, operations that led to its stabilization - Figure 3.

In accordance with the new allocation, we got the following air flow rates along each level:

-level 250: 12.84 m^3/s (770.4 m^3/min);

-level 300: 8.89 m^3/s (533.4 m^3/min);

-level 360: 5.49 m^3/s (329.4 m^3/min);

-level 425: 10.39 m^3/s (622.8 m^3/min).

Under the direct action of the depression created by the main ventilation network, it is possible to discharge an air flow at a rate of 37.6 m^3/s (2,256 m^3/min).

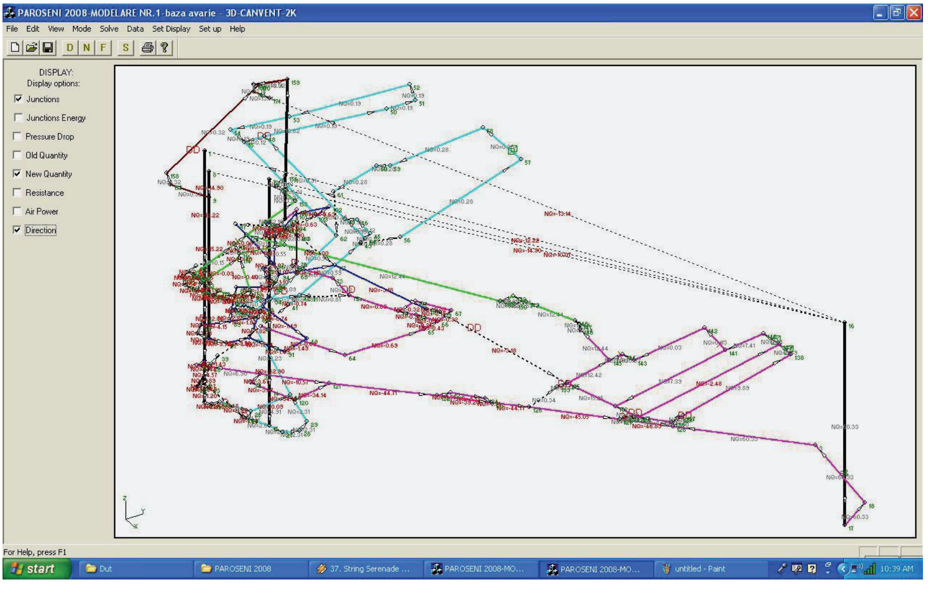

Figure 2. New ventilation station.

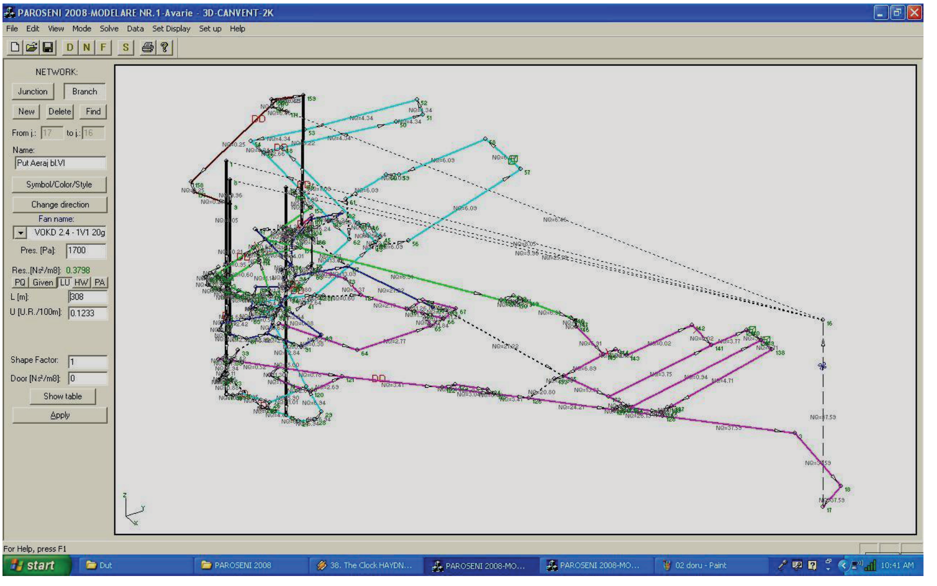

Figure 3. Re-arranged and balanced network of the ventilation system.

The simulation said that whether only the main ventilation station at the Eastern Shaft VOKD 2.4 was in operation, it was possible only to provide a dilution of gases, without the possibility to work normally at coal faces due to the low air flow rates along branches.

5. CONCLUSIONS

1. The ventilation network related to Paroşeni mine is ventilated with the help of one main ventilation station VOD 3.0.
2. The Canadian expert software helped to settle the ventilation network of the mine. Consequently, there have been identified 171 junctions and 216 branches.
3. The newly modeled ventilation network of the mine was used for carrying out 4 simulations of incidents possible to occur in underground.
4. Fan no. 1 of the main ventilation network was destroyed.
5. Consequently, it was necessary to start fan no. 2 and to identify a stand-by solution in case the second fan would damage as well.
6. The solution was to re-start the main ventilation station located at the Eastern Shaft VOKD 2.4.
7. The experts at INSEMEX Petroşani used a Canadian software to simulate the new conditions at Paroşeni mine: the use of the ventilation station located at the Eastern Shaft VOKD 2.4 couldn't support a normal activity in underground and it would operate only incase of damage, providing only the dilution of gases, with no current work at coal faces.

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Metan Drenajının İşletme Performansı Üzerindeki Etkilerinin Araştırılması

Investigating the Effects of Methane Drainage on the Operating Performances of Mines

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ÖZET Kömür işletmelerinde genelde kömürün her bir tonu için 5-20 ton havanın yeraltına gönderilmesi metanla mücadelede yeterli olmaktadır. Bu yöntem ile hava çıkış kuyularında metan konsantrasyonu % 1'in altında tutulabilmektedir. Ancak, yüksek gaz içeriğine sahip kömür damarlarında havalandırma sistemlerinin yalnız kullanılması çeşitli problemlere yol açabilmektedir. Bu problemlerin oluşmasını engellemek ve kömür üretiminin planlı bir şekilde gerçekleşmesini sağlamak için bünyelerinde metan bulunduran damarlarda metan drenajının uygulanması gerekmektedir. Bu çalışmada metan drenajının işletme performansı üzerindeki etkileri incelenmiştir. Bu amaçla öncelikle kömür madenlerinde metanın yol açtığı problemler belirlenmiş ve belli başlıklar altında gruplandırılmıştır. Daha sonra ise belirtilen problemlerin önlenmesinde metan drenajının önemi üzerinde durularak çeşitli yorumlar yapılmıştır. Çalışma sonucunda metan drenajının üretim aksamalarını azalttığı, işletme maliyetlerini düşürdüğü ve çalışma koşulları iyileştirdiği tespit edilmiştir.

ABSTRACT: Coals mines supply 5-20 tones of air per ton of coal to cope with methane. This method allows keeping methane concentration below 1 % in mine exhaust air. However, only using ventilation system may cause various problems at coal seams of higher methane concentrations. Drainage from coal seams of higher methane content is the most important endeavor to prevent these problems and to conduct coal excavation as planned. In this study the effects of methane drainage on the operating performance of coal mines investigated. Having first defined the problems caused by methane; then the importance of methane drainage was discussed. As a result of this study the possible benefits from the methane drainage were identified as decreases in operation breaks, operating costs and enhancements in operating/working condition.

1 GİRİŞ

Kömürleşme sırasında başlıca metan, karbondioksit, azot ve su oluşmaktadır. Metan oluşumunda biri biyojenik diğeri termojenik olmak üzere iki mekanizma söz konusudur (Dallegge ve Barker, 1999). Bitkisel kökenli organik maddelerin kömürleşme sürecinin ilk aşamalarında mikrobiyolojik ayrışma sonucunda biyojenik metan oluşumu gözlenir. Gerek miktarının azlığı, gerekse oluşan metanın birikebileceği bir rezervuar kayanın ortamda bulunmayışı nedeniyle biyojenik metan birikimleri çok ender olup, ancak çok hızlı çöken az sayıda havzada görülebilmektedir. Gömülmeye bağlı olarak artan derinlikle birlikte ulaşılan yüksek sıcaklık değerleri kömürleşme derecesinin artmasına ve termojenik gaz oluşumunun başlamasına neden olmaktadır. Gaz oluşumunun kinetiğine bağlı olarak yaklaşık 55 °C den itibaren karbondioksit, 100°C den itibaren de metan ve azot gazları oluşmaya başlar. Artan kömürleşmeyle birlikte oluşan metan miktarı da artar (Yalçın ve Durucan, 1983). Kömürün kolloidal yapısı, kömür hacminin 1-40 misli kadar metan gazını içinde tutmasına imkan sağlamaktadır (Dallegge ve Barker, 1999; Aydın ve Karakurt, 2009b).

Yeraltında, katı kömürle beraberindeki metan gazı bir basınç altında dengededir. Kömür damarları içerisinde depolanmış olan metan;

a. Çatlaklarda, kırıklarda ve gözenek içinde serbest gaz olarak,
b. Çatlaklarda ve gözeneklerde kömür yüzeyine tutunmuş olarak ve
c. Su içerisinde çözünmüş olarak bulunur.

Kömürde oluşan gaz önce adsorpsiyon yoluyla tutulmaktadır. Adsorplama kapasitesinin üzerine çıkıldığı durumlarda, gaz, formasyon suyu içerisinde çözünmekte ve/veya serbest gaz olarak gözenek ve çatlaklarda birikmektedir. Yukarıda belirtilen maddelerden yalnızca ilk ikisi üretim esnasında açığa çıkan metan gazı miktarı açısından önemlidir. (Gürdal ve Yalçın, 1992).

Metan gazı renksiz, kokusuz, havadan daha hafif yanıcı, patlayıcı ve boğucu bir gazdır. Hava içerindeki konsantrasyonu % 5-15 olduğu durumda patlayıcı özellik göstermektedir. Çizelge 1'de metan gazının bazı fiziksel özellikleri verilmektedir.

Çizelge 1. Metan gazının bazı fiziksel özellikleri (Arslan, 2006)

Molekül Ağırlığı	16.042 kg/kmol
Molekül Hacmi	22.36 m^3/kmol
Yoğunluk	0.7168 kg/ m^3
Spesifik Ağırlık	7.0294 N/ m^3
Kaynama Noktası	111.3 °K
Erime Noktası	90.5 °K
Kritik Sıcaklık	190.5 °K
Kritik Basınç	463.03 N/cm^2
Kritik Yoğunluk	162 kg/m^3
Yayılma Katsayısı	0.196 cm^2/s
Kalorifik Değeri:	
Net alt	35.994 MJ/m^3
Brüt Üst	39.942 MJ/m^3

Mevcut yasalara göre metan konsantrasyonu aynada ve ocak içerisindeki diğer bölgelerde kontrol edilmelidir (EPA-1, 1999). Kömür işletmelerinde, metan kontrolü iyi bir şekilde planlanmış havalandırma sistemleri kullanılarak gerçekleştirilebilmektedir. Büyük kömür işletmelerinde kömürün her bir tonu için 5-20 ton hava yeraltına gönderilmektedir. Hava çıkış kuyularında metan konsantrasyonunu %1'in altında tutmak için bu yöntem çoğu zaman yeterli olmaktadır. Ancak, yüksek gaz içeriğine sahip kömür damarlarında havalandırma sistemlerinin yalnız kullanılması çeşitli problemlere yol açabilmektedir (Hartman vd., 1997). Bu tip kömür madenlerinde metan drenajının havalandırmayla birlikte uygulanması işletmeye birçok açıdan fayda sağlayacaktır.

Metan drenajı ile birlikte damarın metan içeriği azaltılmakta ve çalışma bölgelerine erişmeden önce ocak dışına gönderilmektedir. Böylelikle çalışmalar için güvenli koşullar sağlanırken havalandırma sistemlerinin verimi de hissedilir derecede artırılmaktadır (Aydın, 2008). Metan gazının kaynağından emilerek saf dışı bırakılmasıyla, nispeten gazlı sınıfa dahil edilen kömür ocaklarında kartiye, pano ve uzun ayak havalandırma sistemlerinin verimi hissedilir derecede artmaktadır (Güney, 1971). Ek olarak havalandırma sistemleriyle eş zamanlı olarak yürütülen metan drenajı çoğu kömür madeninde metan konsantrasyonunu düşük tutmanın en ekonomik yöntemi olabilmektedir.

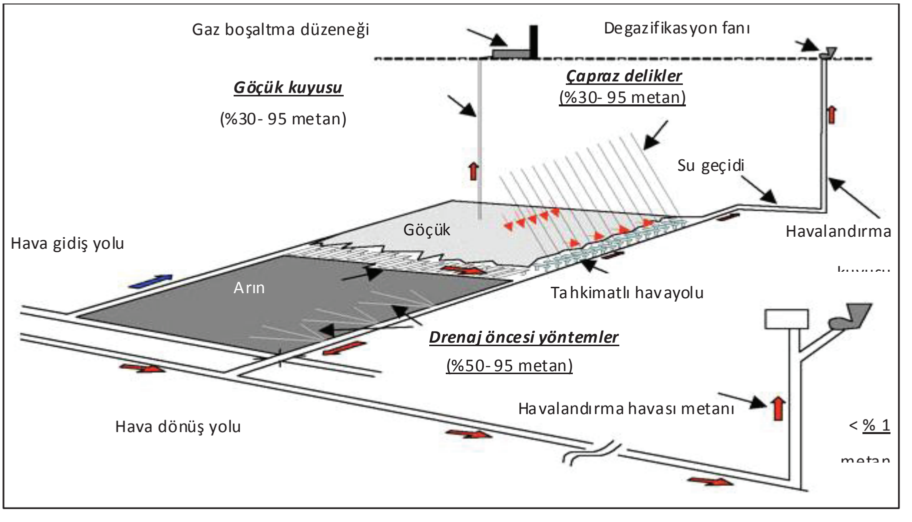
Madencilikte güvenlik koşullarını geliştirmek ve maden açıklıklarındaki metanın bir sonucu olarak oluşabilecek aksama sürelerini azaltmak için, dünyadaki birçok madencilik şirketi madencilik öncesi veya madenin ömrü boyunca damarın bünyesinde bulunan metanı kazanmak için bir drenaj sistemi kullanmaktadır (EPA-1, 1999; Jakubow ve Tor, 2006; Pilcher vd., 2004).

Bu çalışmada metan drenajının işletme performansı üzerindeki etkileri incelenmiştir. Bu amaçla öncelikle kömür madenlerinde metanın yol açtığı problemler belirlenmiş ve belli başlıklar altında gruplandırılmıştır. Daha sonra ise belirtilen problemlerin önlenmesinde metan drenajının önemi.

2 METAN ÜRETİM YÖNTEMLERİ

Genel olarak, gaz kaynağıyla temasa geçilmesi ve gazın emilmesi metan drenajı olarak adlandırılmaktadır. Drenaj işlemi, metanın bir boru sistemine dahil edilmesi, yüzeye ulaştırılması ve atmosfere bırakılması veya kullanılmak üzere endüstriyel tesislere ulaştırılması ile tamamlanmaktadır. Metan drenajı ile ortalama % 20-95 oranında metan (saf yada kirleticiler içeren) yeraltı işyerlerine erişmeden önce ocak dışına gönderilmektedir (Kruger ve Franklin, 2006; Gatnar ve Tor, 2003).

Drenaj sistemleri madencilik öncesinde, madencilik döneminde ve madencilik sonrasında kömür damarı içerisinde bulunan metan gazını üretirler. Yaygın olarak kullanılan drenaj yöntemleri (Şekil 1), yüzeyden damara delinen düşey kuyular, yüzeyden göçük bölgesine delinen düşey kuyular, hazırlık galerilerinden arın önüne delinen yatay delikler ve çevreleyen tabakaya doğru delinen çapraz deliklerden oluşmaktadır (EPA-1, 1999; Jakubow ve Tor, 2006).



Şekil 1. Yaygın olarak kullanılan metan drenaj yöntemleri ve üretilebilecek gaz miktarları (EPA-1, 1999; Aydın, 2008; Aydın ve Kesimal, 2007; Aydın vd., 2009)

3 DRENAJ UYGULAMALARININ İŞLETME PERFORMANSI ÜZERİNDEKİ ETKİLERİ

Gazlı yeraltı kömür damarlarında, drenajın uygulanmadığı durumlarda metan çeşitli problemlere yol açmakta ve çalışma koşulları kötüleşmektedir. Bu ise üretimin aksamalarına yol açmaktadır. Bu bölümde bahsedilen koşulların oluşumuna neden olan metan kaynaklı olaylar belirlenmiş ve üç ana başlıkta toplanmıştır (Çizelge 2). Bunlar üretim aksamaları, maliyet ve çalışma koşullarıdır.

Kömür madenlerinde meydana gelen üretim aksamalarına yüksek metan içeriği, ani metan püskürmeleri, grizu patlaması/yanması ve kömür tozu patlamaları sebep olmaktadır. Maliyetle ilgili problemler ise rezervin bir bölümünün işletilememesi, gazın pazarlanmasının mümkün olmaması, havalandırma maliyetlerinin artması ve suyun yol açtığı ek maliyetler olarak sıralanabilmektedir. Ek olarak yüksek hava hızı ve etkisiyle oluşan toz da çalışma koşullarını etkilemektedir.

Çizelge 2. Gazlı yer altı kömür ocaklarında metan kaynaklı problemler.

Değerlendirme Kriterleri	Kaynak
Üretim aksamaları	Ani metan püskürmesi Yüksek metan içeriği Grizu patlaması/yanması Kömür tozu patlaması
Maliyet	İşletilemeyen rezerv Gazın pazarlanamaması/kullanılamaması Havalandırma ve hazırlık Su
Çalışma koşulları	Toz, yüksek hava hızı

Takip eden kısımlarda metan drenajının yukarıda belirtilen kaynaklar üzerindeki etkisi ele alınacaktır.

3.1 Üretim aksamaları

Yüksek metan birikimi, metan patlaması, patlama sonucu oluşan kömür tozu patlamaları ve ani metan püskürmesinin oluşmasıyla kömür işletmelerinde üretim aksamakta/durmakta ve can kayıpları meydana gelmektedir (Flores, 1997).

3.1.1 Yüksek metan içeriği

Drenajın yapılmadığı ve havalandırmanın klasik havalandırma sistemleriyle sağlandığı gazlı kömür madenlerinde belirli bölgelerde metan birikimi olduğunda güvenli çalışma koşullarının sağlanması için üretim yavaşlatılmakta ya da durdurulmaktadır. Bu süreçte gazın bölgeden uzaklaştırılması için çalışmalar yapılmaktadır. Bu işlemler çok zaman almaktadır ve işletme ekonomisini etkilediğinden ötürü önlenmesi gerekmektedir.

Drenajın uygulanması halinde damardan açığa çıkacak gaz potansiyelinin azalmasının bir sonucu olarak metan birikimlerinde azalma gözlenecek ve üretim kesintisiz bir şekilde devam edecektir.

3.1.2 Metan Patlaması

Metan hava karışımın tutuşmasında etkin parametreler yeterli miktarda (% 5-15) bir karışım, sıcaklık ve zamandır. Bunlardan herhangi birinin eksik olması durumunda patlama gerçekleşmemektedir (Ergin, 1977; Flores, 1997; Yerebasmaz, 1987).

Metan drenajı bu miktarlarda gazın ocak içerisine sızmasını engelleyeceğinden patlama riskini ortadan kaldıracaktır.

3.1.3 Ani metan ve kömür püskürmesi

Ani gaz ve kayaç püskürmesi, kayaç içinde bulunan gaz basıncı sonucu arının bu basınca dayanamayarak parçalanması ve büyük ölçüde gazla birlikte çalışma sahasını doldurması olarak tanımlanmaktadır. Ani gaz ve kayaç püskürmeleri Zonguldak kömür havzasında ölümle sonuçlanan kazalarda önemli rol oynamaktadır. Olay sonucu bölgede birçok madenci yaşamını yitirmiştir (Kocal, 1985).

Drenaj uygulandığı takdirde damar içerisindeki boşluklarda yüksek basınç altında bulunan gaz ortamdan uzaklaştırılacağından bu ve benzeri olayların meydana gelme olasılığı dikkate değer bir şekilde azalacaktır.

3.1.4 Kömür tozu patlaması

Daha önceden metanın minimum %5 oranında olduğu durumlarda patlamanın gerçekleşeceği belirtilmişti. Ancak hava-metan karışımına kömür tozunun eklenmesi durumunda patlamanın alt sınır değeri düşebilmektedir. Bu olay kömür tozunu üretildiği ve metanın serbest kaldığı aynalarda oluşabilmektedir (Cashdollar ve Sapko, 2006; Didari, 1985; Üstünkol, 1975)

Bünyelerinde metan içeren damarlarda metan drenajının uygulanması, metanın kömür tozu patlaması üzerine olan etkisini azaltacaktır.

3.2 Maliyetler

Yukarıda belirtilen aksaklıkların işletme ekonomisi üzerindeki olumlu etkilerinin yanı sıra metan drenajıyla işletilebilecek olan rezerv artırılabilmekte, elde edilen gaz çeşitli uygulamalarda kullanılabilmekte ve havalandırma ve hazırlık maliyetleri düşürülebilmektedir. Dolayısıyla işletme maliyetlerinde gözle görülür bir düşüş elde edilmektedir.

3.2.1 İşletilebilecek olan rezerv

Metan gelirinin yüksek olduğu ve metan drenajının yapılmadığı damarlar, yüksek gaz içeriği yüzünden işletilememektedirler. Drenajın yapıldığı durumlarda ise bu durum ortadan kalkacak ve rezervden mümkün olduğunca faydalanılacaktır.

3.2.2 Gazın pazarlanması/ kullanılması

Kömürde metan drenajının uygulanması söz konusu olduğu zaman, konu hakkında işletmenin vermesi gereken kararı etkileyen en önemli faktörlerden birisi gazın kullanımı ya da pazarlanmasıdır (Aydın ve Kesimal, 2007). Gazın pazarlanabilmesi/kullanılması sera gazı etkisinin azaltılmasının yanı sıra gaz için ekonomik bir kullanım alanı yaratmaktadır (Aydın ve Karakurt, 2009b; EPA-1, 1999).

Batı Avrupa ocaklarında drene edilen metan çeşitli yollarla demir çelik endüstrisi, kok fırınları, tuğla fırınları, cam fabrikaları, plastik üreten kimya endüstrisi gibi yerlerde yakıt olarak kullanılmaktadır (Flores, 1997). Çeşitli firmalar yüksek ısıl değere sahip kömür kökenli metanı değerlendirmek için çeşitli araştırmalar yapmaktadır.

Kömür damarlarından elde edilen gaz kalorifik değerine göre sınıflandırılabilir. Bunlar;

a. Yüksek ısıl değerli gaz (>8.425 kCal/m^3)
b. Orta ısıl değerli gaz (2.660-8.425 kCal/m^3)
c. Düşük ısıl değerli gaz (<2.660 kCal/m^3) (Karakurt vd., 2009)

3.2.2.1 Yüksek ısıl değerli gaz

Bu gazlar bir doğal gaz hattında kullanılabilecek olan yüksek kalorifik değerli gaz olarak tanımlanırlar. Yüksek kalorifik değere sahip gazlar için çeşitli kullanım alanları mevcuttur. Eğer drenaj sistemleri ile yüksek kalorifik değere sahip gaz üretilebilir ise gaz bir doğal gaz şirketine pazarlanabilir. Bu bağlamda kömür madenine yakın bölgelerde doğal gaz hattı sisteminin mevcut olması gazın bu amaçla değerlendirilebilmesi açısından en önemli kriterdir. Çevrede doğal gaz hattının ve gazı pazarlayan bir şirketin olmaması durumunda gazın kullanımı için mevcut olan diğer seçenekler değerlendirilmelidir (Flores, 1997).

Bu seçeneklerden bir tanesi amonyak, asetik asit ve etanol üretmek için bir besin deposu olarak gazın kullanılmasıdır.

Kömür kökenli doğal gazı kullanmanın diğer bir yolu da gazın otomobillerde yakıt olarak kullanılması için sıvılaştırılması ya da sıkıştırılmasıdır. Uygulama Ukrayna'da başarılı bir şekilde uygulanmaktadır (Kruger ve Franklin, 2006; EPA-1, 1997). Madencilikte yüksek ısıl içeriğe sahip metan üreten şirketlerin sayısı nispeten düşüktür.

3.2.2.2 Orta ısıl değerli gaz

Bu kategorideki gazlar kalorifik değerlerindeki geniş yayılımından dolayı birçok kullanım alanına sahiptir. Gaz, kalorifik değerinin 8.425 kCal/m^3 değerine yakın bir değerde olması durumunda doğal gaz olarak kullanılması için zenginleştirilmesi gerekmektedir. Zenginleştirme iki şekilde yapılabilmektedir. Bunlar;

a. Gaza kalorifik değeri yüksek bir gazın ilavesiyle gazın zenginleştirilmesi
b. Ortamda bulunan nitrojen, oksijen ve karbondioksit gibi gazların metandan arındırılmasıyla gazın zenginleştirilmesi (Jakubow ve Tor, 2006).

Bu kategorideki gazların önemli ve gelişen bir kullanım alanı da doğal gaz, petrol ve kömürün kullanıldığı uygulamalarda gazın kullanılmasıdır. Bu uygulamalarda kömür kökenli metan maden tesislerini ve maden havasını ısıtmak, termal kurutucu kaynağı, seraları ısıtmak ve ağır metaller içeren suyun işlenmesinde bir ısı kaynağı olarak kullanılabilmektedir. Bu uygulamalar çoğu durumlarda birincil yakıt olarak kömür kökenli doğal gazı kullanırlar (Pilcher vd., 2004).

Kömür kökenli metanın ikincil bir kaynak olarak kullanıldığı uygulamalar da mevcuttur. Bu kullanımlar kömürün yakılması, endüstriyel kaynatıcılarda ve maden eritme ocaklarında bir doğal gaz desteği olarak kömür kökenli metanın kullanılmasıdır (Gatnar ve Tor, 2003).

Elektrik üretiminde gazın kullanılması belirtilen kalorifik değere sahip kömür kökenli metan için diğer bir kullanım alanı olabilmektedir. Bu amaçla içten yanmalı motorlar, tirbünler ya da yakıt hücreleri kullanılmaktadır. Üretilen elektrik madende veya başka yerlerde uygun amaçlar için kullanılabilmektedir. (Yerebasmaz, 1987; Sööt vd., 2006).

Orta kalorifik değere sahip gazlardan elde edilen gelir, yüksek kalorifik değere sahip gazların değerlendirilmesi sonucu elde edilen gelir kadar yüksek değildir. Ancak gazın kullanım alanları büyümekte ve ilgili teknolojiler gelişmektedir (EPA-2, 1998; EPA-3, 1998).

3.2.2.3 Düşük ısıl değerli gaz

Maden işletmeleri, genellikle düşük içerikli ısıl değere sahip gazı klasik havalandırma sistemleri ile atmosfere yayarak kullanmazlar. Ancak bu kategorideki gazlar için de çeşitli kullanım alanları mevcuttur ve gazın kullanılmasına imkan sağlayacak yeni kullanım alanları için araştırmalar yapılmaktadır. Bu kategorideki gazların en önemli kullanım alanlarından biri ısı üretmek için termal bir oksidanda gazın kullanılmasıdır. Bu olayda metanın yaklaşık olarak % 75'i ısıya dönüştürülür ve enerji tasarrufu sağlanmış olur (EPA-4, 1998; Mattus, 2006; Su ve Agnew, 2005).

3.2.3 Havalandırma ve hazırlık maliyetleri

Çoğu kömür madeninde kesintisiz üretimin sağlanması için kullanılan geleneksel havalandırma sistemleri oldukça pahalıdır.

Drenajın uygulanmadığı kömür madenlerinde ortamdaki metan gazı miktarı arttıkça, gazı seyreltmek için gerekli hava miktarı da artmaktadır. Artan hava gereksinimlerini karşılayabilmek için daha büyük boyutlu fanların seçilmesine ve tali havalandırmanın uygulanmasına gerek duyulacağından enerji sarfiyatı da artacaktır.

Drenaj maliyetlerinin havalandırma maliyetlerinden düşük olduğu bilinmektedir. Bu sebepten metan drenajını artan havalandırma gereksinimleri yerine kullanmak daha karlı olacaktır (Hartman, 1997; McPherson, 2004).

Drenajla birlikte ayak içine sızması muhtemel gaz potansiyeli azalacağın pano boyutlarının seçiminde daha serbest hareket edilebilecektir. Dolayısıyla daha büyük pano boyutları seçilebilecektir. Pano boyutlarının yüksek seçilmesi kömür damarlarının üretilmesi için gereken hazırlık galerilerinin sayısını azaltacaktır. Yapılan hazırlıklar minimuma ineceğinden dolayı hazırlık maliyetleri azalacaktır.

Kim ve Mutmansky (1990) bir madende 20 yıllık maden ömrü boyunca, birkaç durum için havalandırma maliyetlerin analizlerini yapmışlardır. Sonuçlardan biri 11 m^3/ton' luk gaz içeriğine sahip bir kömür damarında yüzeyden delinen düşey kuyularla drenajda, enerji maliyetlerinin yirmi yıllık zaman diliminde 11.000.000 $ civarında azalma meydana geleceğidir. Yüzeyden delinen kuyulara ek olarak tavan ve taban galerilerinden arın önüne delinen delikler metan üretimi için uygulanırsa 3.000.000 $ lık bir tasarruf sağlanmaktadır (EPA-1, 1999). Çizelge 3'ten da anlaşılacağı gibi drenajın uygulandığı durumda havalandırma maliyetleri yarı yarıya azalmıştır. Drenaja ek olarak metandan kaynaklanan üretim kesilmelerinde azalmanın söz konusu olduğu durumda ise maden ömrü iki üç yıl azalmaktadır.

Çizelge 3. Metan drenajının havalandırma maliyetine etkisi (EPA-1, 1999).

Durum	İşçi üretimi (ton)	Uzun ayak üretimi (ton)	Havalandırma Maliyetleri ($)	Hazırlık Maliyetleri ($)	Drenaj maliyetleri ($)
Drenaj yok	12 463 200	36 333 500	26 863 400	12 071 200	-
%60 Drenaj	7 321 900	51 729 800	14 561 100	7 076 200	32 528 000
%60 Drenaj ve üretim aksamalarında azalma	7 321 900	51 729 800	15 688 000	7 076 200	32 528 000

3.2.4 Suyun yol açtığı problemler azalır

Madencilik çalışmalarında ortamda bulunan su çalışmaların etkili bir şekilde sürdürülebilmesi için ortamdan uzaklaştırılmalıdır. Bu işlemler tavan ve taban galerilerinden arın önüne deliklerin delinip mevcut olan suyun ortamdan alınması ve özel pompalar vasıtasıyla suyun dışarı gönderilmesi aşamalarından oluşmaktadır (Saltoğlu, 1976). Bu işlemler çalışmaları yavaşlatmakta ve ek işgücü ve ekipman gerektirmesinden dolayı işletme maliyetlerine ek maliyetler getirmektedir. (EPA-1, 1999). Ancak, drenajın uygulandığı durumlarda gazla birlikte suda ortamdan uzaklaştırılacağından, çalışmalar için elverişli bir ortam hazırlanmış olacaktır.

3.3 Çalışma koşulları

Gazlı kömür madeninin drenaj sistemleri kullanılmadan geleneksel havalandırma sistemleriyle havalandırılması durumunda ortamda bulunan gazı seyreltmek için yüksek hızlar gerekmektedir. Yüksek hızda havanın yeraltına gönderilmesi fazla miktarda toz oluşumuna neden olacaktır.

Toz görüş mesafesini azaltmakta, gözü tahriş etmekte ve çalışma verimini düşürmektedir. Bunlara ilaveten en önemlisi kömür tozu çok miktarlarda ve uzun süreli teneffüs edildiği pnömokonyoz hastalığına neden olmaktadırlar. Havalandırma ile drenaj eş zamanlı olarak uygulanırsa gazın büyük bir çoğunluğu drene edileceğinden gazı seyreltmek için ortama gönderilecek hava miktarı düşecektir. Bu olay toz oluşumunu büyük bir oranda azaltacaktır (Saltoğlu, 1970).

Drenaj sistemlerinin uygulandığı işletmelerde hava hızının azalmasının bir sonucu olarak toz oluşumu azalacak ve işçilerin tozdan zarar görmesi engellenmiş olacaktır.

Çizelgeler 4 ve 5 sırasıyla Zonguldak Taşkömür Havzası'nda 1970-2005 yılları arasında meydana gelen iş kazalarında hayatını kaybeden ve yaralananların sayısını göstermektedir. Metan kaynaklı patlamalar genellikle ölümlerle sonuçlanmaktadır. Belirtilen yıllar arasında 1422 kişi hayatını kaybetmiştir. Grizu ve gazlara bağlı olarak yaşamlarını kaybedenlerin sayısı ise bu oranın yaklaşık olarak % 38'ini oluşturmaktadır. Bu yıllar arasında 181419 kişinin yaralandığı görülmektedir. Grizu ve gazlara bağlı olarak gerçekleşen aralanmalar bu miktarın yaklaşık % 1' ini oluşturmaktadır.

Bu olaylar işçilerin psikolojisini olumsuz yönde etkilemekte ve çalışma verimini düşürmektedir.

Drenajın uygulandığı kömür damarlarında grizu kaynaklı problemler dikkate değer derecede azalacağından işçiler kendilerini daha güvenli bir ortamda hissedecekler ve daha verimli bir şekilde çalışmalarını sürdüreceklerdir.

Çizelge 4. Zonguldak taşkömür havzasında 1970-2005 yılları arasında grizu patlaması sonucu hayatının kaybedenlerin sayısı (üretim isçisi) (Buzkan ve Ofluoğlu, 2007)

Yıl	Ö.S	Yıl	Ö.S	Yıl	Ö.S	Yıl	Ö.S	Yıl	Ö.S
1970	19	1978	25	1986	-	1994	1	2002	-
1971	4	1979	7	1987	-	1995	-	2003	1
1972	32	1980	5	1988	-	1996	-	2004	-
1973	6	1981	-	1989	2	1997	-	2005	9
1974	8	1982	1	1990	5	1998	-	-	-
1975	19	1983	116	1991	-	1999	-	-	-
1976	10	1984	-	1992	264	2000	1	-	-
1977	7	1985	-	1993	-	2001	-	TOP.	542

Çizelge 5. Zonguldak taşkömür havzası kazalarında 1970-2005 yılları arasında grizu patlaması sonucu meydana gelen yaralanma sayısı (üretim isçisi) (Buzkan ve Ofluoğlu, 2007)

Yıl	Y.S	Yıl	Y.S	Y.S	Y.S	Yıl	Y.S	Yıl	Y.S
1970	14	1978	2	1986	218	1994	-	2002	3
1971	6	1979	6	1987	243	1995	-	2003	7
1972	102	1980	7	1988	270	1996	-	2004	2
1973	9	1981	-	1989	1	1997	-	2005	4
1974	2	1982	1	1990	5	1998	-	-	-
1975	7	1983	110	1991	-	1999	-	-	-
1976	11	1984	-	1992	78	2000	2	-	-
1977	1	1985	1	1993	-	2001	3	TOP.	1114

3 SONUÇ

Gazlı kömür damarlarında uygulanan metan drenajıyla üretim aksamalarının önüne geçilmekte, işletme maliyetleri düşmekte ve çalışmalar için güvenli koşullar sağlanmaktadır. Ek olarak damarlardan elde edilen gaz pazarlanabilmekte ya da işletmede çeşitli uygulamalarda kullanılabilmektedir. Bu işletme maliyetlerini düşürdüğü gibi işletmenin enerjide dışa bağımlılığını da minimuma indirecektir.

Bütün bu avantajlar göz önünde bulundurulduğunda bünyelerinde yüksek oranlarda metan gazı bulunduran kömür damarlarında metan drenajının uygulanmaması büyük bir yanlışlık olacaktır. Metan drenajına karar verme aşamasında yöntemin avantajları ve maliyeti açık bir şekilde ortaya konularak bir karlılık analizi yapılmalı ve işletme için en faydalı olabilecek yönteme karar verilmelidir.

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Maden Havasındaki Metanın Oksidasyonu ile Enerji Üretimi
Energy Production by Oxidation of Mine Ventilation Air

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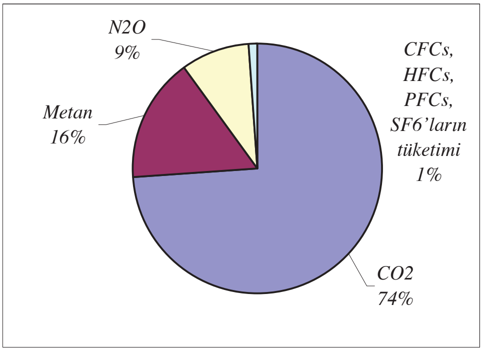
ÖZET Metan, etkili bir sera gazı ve doğal gazın en önemli bileşenidir. Kömür damarları önemli miktarlarda metan içerirler ve kömür üretimi esnasında bu gaz, ocak havasına karışır. Ocak havasına karışan metan, düşük konsantrasyonlarda (%5-15) patlama özelliği olduğundan işletmeler için tehlike oluşturmaktadır. Kömür işletmelerinin çoğu, bu tehlikenin önüne geçmek için iyi bir havalandırma sistemi ile ocak havasına karışan metanın havalandırma çıkış kuyusundan atmosfere yayılmasına izin verirler. Gazın bu şekilde atmosfere salınması küresel ısınmaya katkıda bulunduğu gibi ekonomik olarak değerlendirilebilecek bir kaynağın israfı anlamına da gelmektedir. Son teknolojik gelişmeler, maden çıkış kuyusundan düşük konsantrasyonlarda atmosfere salınan metanın değerlendirilmesine olanak sağlamıştır. Bu teknolojiler sayesinde metan, karbondioksite dönüştürülebilmekte ve bu dönüşüm sonucunda enerji elde edilmektedir. Böylelikle, metan gazının küresel ısınma üzerindeki etkisi 20 kata kadar azaltılabilmektedir. Bu çalışmada, ocak havasındaki metanın oksidasyon teknolojileri ile enerji üretimi anlatılmıştır. Ayrıca, metan oksidasyon teknolojilerinin uygulanabilirliklerine yönelik bir değerlendirme de sunulmuştur.

ABSTRACT Methane is one of the major greenhouse gases and the principal component of natural gas. Coal seams contain significant quantities of methane and it is released into the mine ventilation air during extraction of coal. Because the methane released into the mine ventilation air is explosive at relatively low concentrations (5-15%), it creates hazard for mines. Majority of coal mines allow to discharge the methane in the ventilation air through uptake shaft to atmosphere. Releasing the methane to atmosphere both contributes to global warming and means wasting of an economical source of energy. Recently, technological developments have enabled to evaluate the methane gas discharged at low concentrations through uptake shaft to atmosphere. The methane can be transformed into carbon dioxide by these technologies and energy is obtained using the transformed methane. Therefore, the effect of methane on global warming can be reduced as much as 20 times. In this study, the energy production by oxidation technologies of methane found in the mine ventilation air is explained. In addition, an evaluation for the applicability of methane oxidation technologies is presented.

1 GİRİŞ

Her geçen gün etkisini daha da arttırarak hissettiren Küresel ısınma, giderek daha büyük bir problem oluşturmaktadır. Küresel ısınma ve iklim değişikliği konusunda mücadele eden en büyük yapılanma olan Kyoto Protokolü'nü şimdiye kadar birçok ülke imzalamıştır. Bu protokolü imzalayan ülkeler karbon dioksit ve metan gazı gibi sera etkisine neden olan gazların emisyonlarının minimumuma indirileceğini beyan etmişlerdir (Ruixiang vd., 2008). Metan, karbondioksit gazından sonra en önemli ikinci sera gazıdır (Şekil 1) ve başlıca oluşum kaynakları arasında, tarımsal faaliyetler, bataklıklar, petrol ve gaz üretimi ile kömür madenleri gelmektedir (Marin vd., 2009).

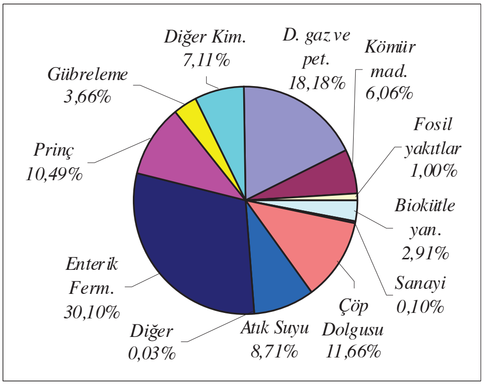


Şekil 1. Sera gazlarının küresel ısınmaya katkısı (Kruger ve Franklin, 2006).

Küresel ısınma bazında düşünüldüğünde, metan karbondioksite oranla 21 kat daha fazla etkiye sahiptir. Buna karşın, karbondioksitin atmosferdeki ömrü yaklaşık 100 yıl iken metan gazının ömrü 12–15 yıldır (Singh ve Singh, 2009). Küresel ısınmaya katkıda bulunan başlıca sera gazının metan olduğu düşünülmektedir.

Metan gazı emisyonları değişik kaynaklardan açığa çıkmakla beraber gaz ve petrol sektörü, katı yakıtların yakılması, tarımsal faaliyetler, bataklıklar ve kömür madenleri metan gazını oluşturan başlıca kaynaklar arasındadır. Metan gazı emisyon miktarları oluşum yeri ve açığa çıkma şekline göre değişiklik gösterebilir (Ruixiang vd., 2008).

Yeraltı kömür madenleri antropojenik metan gazı emisyonlarının (2005 itibarı ile) yaklaşık % 6-8'ini oluşturur (Şekil 2). Ayrıca, yeraltı kömür madenlerinden özellikle ocak çıkış kuyusundan atmosfere salınan metan düşük konsantrasyonlu olmasına rağmen küresel anlamda bakıldığında toplam kömür kaynaklı metan gazı miktarının yaklaşık % 70'ini oluşturmaktadır. Bundan dolayı, son zamanlarda bu konuda yapılan çalışmalara yönelik araştırma ve geliştirmeler yeraltı kömür damarlarından açığa çıkan metan gazının azaltılması ve kullanımına odaklanmıştır (Su vd., 2008).



Şekil 2. Çeşitli sektörlerin metan emisyon miktarları (Kruger ve Franklin, 2006).

Metan yeraltı kömür damarlarından genellikle üç farklı şekilde açığa çıkmakta ya da ele geçirilebilmektedir (Reddick, 2005):

a) Çalışma süresince maden havasına karışarak ve havalandırma yoluyla atmosfere salınarak (% 0,1–0,7)
b) Madencilik faaliyetleri öncesinde drenajla (% 60–90)
c) Üretim süresince damarlardan drenajla (%30–95)

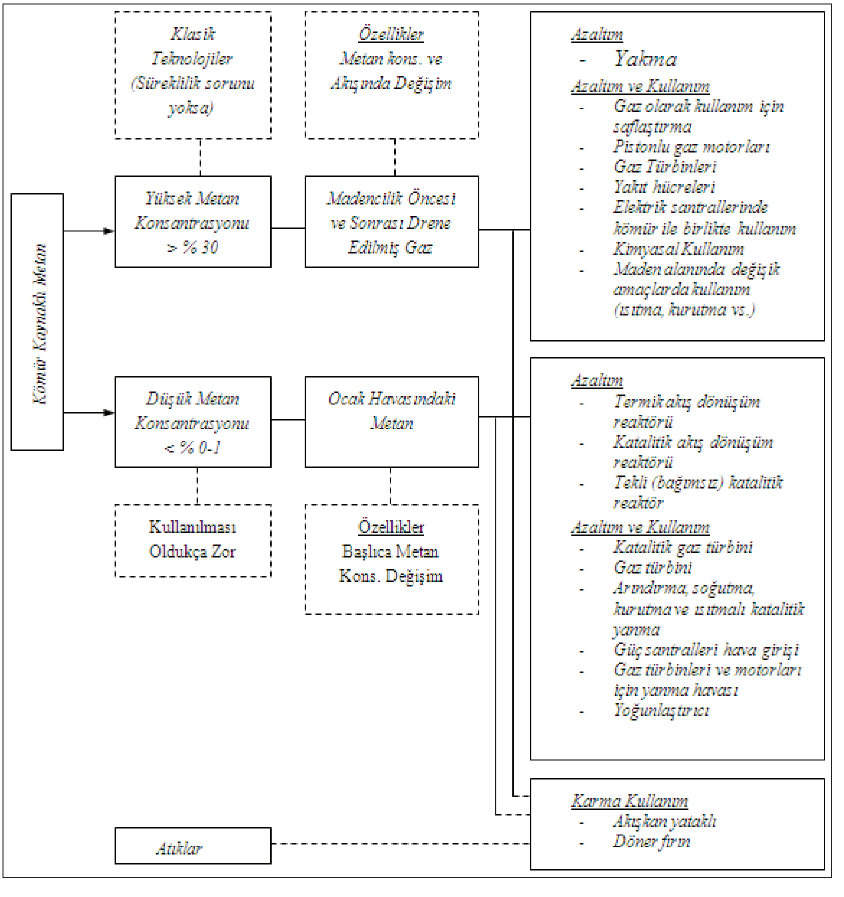
Ocak havasında bulunan metanın, hava ile birlikte hacminin büyük, metan konsantrasyonunun düşük ve hava akışının da değişken olması nedeniyle bir enerji kaynağı olarak kullanılması oldukça zordur. Özellikle, metanın düşük konsantrasyonlarda bulunması, gazın azaltım ya da kullanım seçeneklerini kısıtlamaktadır. Ocak havasındaki metan ya bu şekliyle azaltılma yoluna gidilecektir ya da belirli bir seviyeye kadar zenginleştirilerek enerji kaynağı olarak kullanılabilecektir. Ocak havasındaki düşük konsantrasyonlu metanın konsantrasyonunu arttırmaya yönelik etkin bir teknoloji olmamasına rağmen çalışmalar devam etmektedir (Mallet ve Su, 2003).

Ocak havasındaki düşük konsantrasyonlu metanın enerji kaynağı olarak değerlendirilmesi yönünde yapılan çalışmaların önemli bir kısmı düşük konsantrasyonlu metanın oksidasyonu üzerinde yoğunlaşmıştır (Su ve Agnew, 2006). Bu çalışmada, ocak havasındaki düşük konsantrasyonlu metanın ocak çıkış kuyusundan atmosfere salınması yerine oksidasyon teknolojileri ile nasıl karbondioksite dönüştürüldüğü ve enerji kaynağı olarak kullanıldığı hakkında bilgiler verilmiştir. Ayrıca, yöntemin dünyadaki uygulamalarına yönelik örneklerde sunulmuştur.

2 METANIN GAZININ OKSİDASYON TEKNOLOJİSİ VE MEKANİZMASI

2.1 Teknoloji Sınıflaması

Kömür kaynaklı metanın, azaltım ve kullanım teknolojilerine yönelik genel bir sınıflama Şekil 3'de gösterilmektedir. Temel mekanizmalar, teknik esaslar ve uygulanabilirlik durumu bakımından ocak havasındaki metanın azaltım ve kullanım teknolojileri ise Çizelge 1'de gösterilmiştir. Çizelgeden de anlaşılacağı üzere, ocak havasındaki metanın kullanım/azaltım teknolojisi iki şekilde sınıflandırılmaktadır. Bu sınıflama ocak havasının kullanım şekline göre yardımcı ve ana kullanım olarak yapılmıştır. (Su ve Agnew, 2006., Mallet ve Su, 2003).



Şekil 3. Kömür kaynaklı metanın azaltım ve kullanım teknolojileri (Su vd., 2005).

Çizelge 1. Ocak havasındaki metanın kullanım ve azaltım seçenekleri (Su ve Agnew, 2006).

Teknoloji	*Oksidasyon Mekanizması*	*Temel Kullanım*	*Uygulama*
Yardımcı Kullanım			
Elektrik santrali için yanma havası	*Termal*	*Elektrik santrali fırınlarında yanma*	*Azaltım/Kullanım, henüz pilot çapta. Büyük ölçeli uygulaması da düşünülmektedir.*
Gaz türbin/motoru için yanma havası	*Termal*	*Klasik gaz türbin/motorlarında yanma*	*Azaltım/Kullanım. Henüz uygulama yok*
Fırınlarda kömürün hibritleştirilmesi	*Termal*	*Döner yanma odasında yanma*	*Azaltım/Kullanım. Pilot çapta uygulama mevcut*
Akışkan bir yatakta kömürün hibritleştirilmesi	*Termal*	*Akışkan bir yatak içerisinde yanma*	*Azaltım/kullanım. Sadece bir kavram olarak var*
Ana Kullanım			
Termal akış dönüşüm reaktörü	*Termal*	*Isı alıp veren akış dönüşüm reaktörü*	*Azaltım seçeneği var, kullanım seçeneği henüz mevcut değil*
Katalitik akış dönüşüm reaktörü	*Katalitik*	*Isı alıp veren akış dönüşüm reaktörü*	*Azaltım seçeneği var, kullanım seçeneği henüz mevcut değil*
Bağımsız katalitik yakma odası	*Katalitik*	*Geri kazanımlı bağımsız reaktör*	*Azaltım seçeneği var, kullanım seçeneği henüz mevcut değil*
Gaz türbini katalitik yanma	*Katalitik*	*Katalitik yakma odalı ve geri kazanımlı gaz türbini*	*Azaltım, yanma olarak uygulama mevcut ama kullanım olarak henüz laboratuar ölçekte uygulama var*
Gaz türbini	*Termal*	*Yakma odalı ve geri kazanımlı gaz türbini*	*Azaltım, yanma olarak uygulama mevcut ama kullanım olarak henüz laboratuar ölçekte uygulama var (Geliştirilmeye ihtiyacı var)*
Yoğunlaştırıcı	*N/A Adsorplama*	*Adsorblayıcı ve uzaklaştırıcı kullanılarak akışkan/hareketli yataklı çoklu safha*	*Azaltım/Kullanım seçenekleri hala geliştirme aşamasında*

2.2 Metan Gazı Oksidasyon Mekanizması

Metan gazının yanma mekanizması, genel haliyle aşağıdaki bağıntı ile ifade edilebilir.

$$CH_4 + 2O_2 = CO_2 + 2H_2O$$

Esas yanma reaksiyonu, çok sayıda reaksiyon denkleminden meydana geldiğinden yukarıda ifade genelleştirilmiş basit bir yanma reaksiyonu ifadesidir. Metanın yanması, hava/metan oranına bağlı olarak CO ve CO_2 üretebilir. Metan gazı yanmasından meydana gelebilecek diğer reaksiyonlardan bazıları da şu şekildedir.

$$CH_4 + 2H_2O = CO + 2H_2$$

$$2H_2 + O_2 = 2H_2O$$

$$CO + H_2O = CO_2 + H_2$$

Metanın katalitik yanma mekanizmasında, çok sayıda farklı reaksiyonların meydana gelmesi söz konusudur. Metanın muhtemel bir katalitik yanma mekanizması, özellikle heterojen reaksiyonlarında meydana geldiği düşünüldüğünde oldukça karışıktır.

3 OCAK HAVASINDAKİ METAN GAZININ AZALTIM/KULLANIM SEÇENEKLERİ

Ocak havasındaki metanın azaltım ve kullanım teknolojileri Çizelge 1'de de gösterildiği gibi temel olarak iki sınıfa ayrılmaktadır. Birincisi, ocak havasının ikincil bileşen olarak kullanıldığı yardımcı kullanım, ikincisi; ocak havasının temel bileşen olarak kullanıldığı ana kullanım.

3.1 Ocak Havasının Yardımcı Bileşen Olarak Kullanımı

Ocak havası bu aşamada, mevcut yanma işleminin performansını arttırmak için ortam havası olarak kullanılır. Ocak havasındaki metan, ikincil bir yakıt işlevini görür. Bu işlemin uygulandığı alanlar;

i Enerji santrallerinde kömür ile birlikte ocak havasının yanma işlemlerinde kullanılması
ii Atık kömürlerin hibritleştirilmesini sağlayan yanma ünitelerinde (akışkan yataklı ortam ve fırınlar)
iiiKlasik gaz türbinlerinde kullanılması
iv İçten yanmalı motorlar

Çizelge 2'de ocak havasındaki metan gazının yukarıda bahsedilen alanlardaki kullanım teknolojilerinin, teknik ve mühendislik açıdan uygulanabilirliği, ana işletim parametreleri bakımından bir değerlendirmesi sunulmuştur. Burada en önemli sorun, enerji kazanımı için gerekli ünitelerin ocak çıkış kuyularına güvenli bir şekilde yerleştirilebilmesidir. Ancak bu durum her maden için farklılık göstereceğinden detaylı bir araştırma gerektirmektedir.

3.2 Ocak Havasının Ana Bileşen Olarak Kullanımı

Ocak havasının ana bileşen olarak kullanıldığı bu safhada ocak havasındaki metan yanma işlemlerinde birincil yakıt olarak kullanılır. Bu daha çok ikinci derecede yakıt olarak kullanılacak kaynağın fazla olduğu durumlar için geçerlidir. Bu nedenle, ocak havasındaki metanın yanma işlemlerinde birincil yakıt olarak kullanımı.

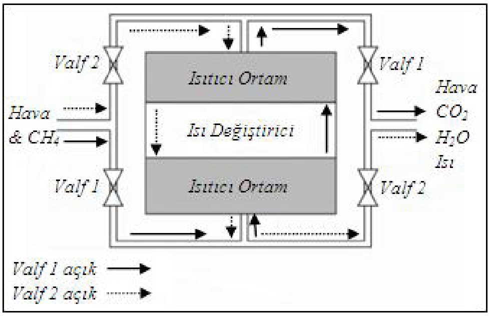
İkinci derecede kullanılacak yakıta, ocak havası içindeki metan içeriğine ve konsantrasyonuna bağlı olmak üzere hemen her azatlım/kullanım teknolojisi için geçerli değildir.

Çizelge 2. Ocak Havasındaki Metanın Yardımcı Bileşen Olarak Kullanım Teknolojilerinin Karşılaştırılması (Su vd., 2005).

Teknoloji	*Özellik*	*Yanma Derecesi (°C)*	*Teknik ve Mühendislik Olarak Uygulanabilirlik*	*Potansiyel Sorunlar*
Enerji santrallerinde kömür ile birlikte ocak havasının yanma işlemlerinde kullanılması	*İnce kömürle birlikte fırınlarda*	*1400–1650*	*Teknik olarak uygulanıyor*	*Sınırlı uygulama alanı*
			Mühendislik olarak herhangi bir madende henüz uygulama yok	*Mevcut fırın/kazanların Potansiyel işletim problemleri*
Atık kömürlerin hibritleştirilmesini sağlayan yanma ünitelerinde (akışkan yataklı ortam)	*Döner fırın*	*1200–1550*	*Teknik olarak uygulanabilir*	*Kendiliğinden yanma*
			Mühendislik olarak herhangi bir madende henüz uygulama yok	*Kömür atık kalitesi için minimum gereksinim*
Atık kömürlerin hibritleştirilmesini sağlayan yanma ünitelerinde (fırınlar)	*Akışkan yataklı*	*850–950*	*Teknik olarak uygulanabilir*	*Kömür atık kalitesi için minimum gereksinim*
			Mühendislik olarak herhangi bir madende henüz uygulama yok	*CH_4 oksidasyonu için gerekli deneme testleri*
Klasik Gaz türbinlerinde kullanılması şeklindedir.	*Gaz Türbini*	*1400–1650*	*Teknik olarak uygulanabilir*	*Türbin yakıtlarının az olması*
			Mühendislik olarak herhangi bir madende henüz uygulama yok	*Tek bir kompresör kullanımında fazla miktarda CH_4 yayılmaktadır. İki kompresör kullanımında da ekipman fazlalığı söz konusudur. Bu da ocak havası kapasitesini düşürmektedir.*
İçten yanmalı motorlar	*Motor*	*1800–2000*	*Teknik olarak uygulanabilir*	*Motor yakıtlarının az olması*
			Mühendislik olarak bir madende uygulaması var	*Düşük miktarlarda ocak havasının kullanılması*

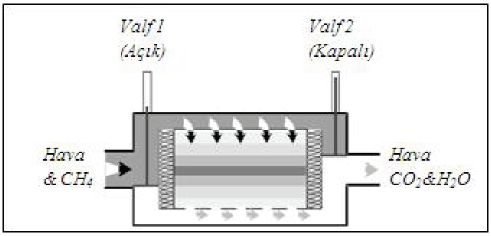
3.2.1 *Termal akış dönüşüm reaktör teknolojisi*

Termal akış dönüşüm reaktör teknolojisinin uygulanma prensipleri, birçok araştırmacı tarafından açıklanmıştır (King ve Traves, 2000., Danell vd., 2002., Su vd., 2005). Bu teknoloji; ocak havasındaki metanın, katı sıcak bir ortamda yanarak ısıya dönüşmesini sağlar. Bu ortam sıcaklığı, ocak havasındaki metanın tutuşma sıcaklığı için gereklidir. Katalitik akış dönüşüm reaktöründen farkı katalitik teknolojisinde kullanılan katalizördür (Mallet ve Su, 2003). Şekil 4'de tipik bir termal akış dönüşüm reaktörünün şematik görüntüsü gösterilmiştir.

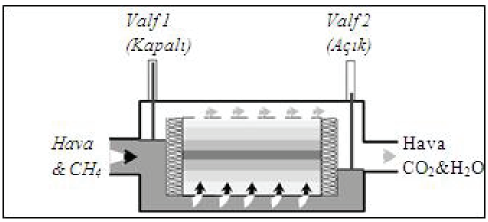


Şekil 4. Termal akış dönüşüm reaktörü şematik gösterimi (US EPA, 2009).

Termal yanma teknolojisinde, işlemin başlaması için ortamdaki elektrikli ısıtıcı bileşenler metanın otomatik tutuşmasını sağlamak amacıyla reaktörün orta tabanının önceden ısınmasını sağlarlar. İşlemin ilk safhasının ilk yarısında, ocak havası ortam sıcaklığında reaktöre girer ve reaktörün bir kenarı boyunca dolaşır. Metanın oksidasyonu reaktör tabakasının ortalarına yakın bir yerde, reaktör içindeki karışımın metanın kendiliğinden tutuşma sıcaklığını aştığında meydana gelir. Reaktör tabakasının yanma işlemine uzak kısımları yeterince ısındığında ya da yanma işlemine yakın kenarları ortama giren ocak havası nedeniyle soğuduğunda, reaktör otomatik olarak akışın yönünü değiştir. Böylelikle yeni ocak havasının ortama girmesiyle sıcak olan kısımlar soğumaya, soğuk olan kısımlar da tekrar ısınmaya başlar. Ayrıca, reaktör içindeki hava akışının dolaşım şekline ve valf durumuna göre iki şekilde dizayn edilmiş reaktörler de bulunmaktadır (Şekil 5-6). Reaktörün merkezine yakın ve ya merkezde metan, kendiliğinden yanma sıcaklığına ulaşır, oksidasyona uğrar ve CO_2'de dönüşecek ısı üretir. Merkezde sıcaklık 1000 ^{o} C'ye çıkar ve bu sıcaklığa ek olarak adyabatik sıcaklık artışı söz konusudur (US EPA, 2009). Termal akış dönüşüm reaktörleri ocak havasındaki metanın % 95'den fazlasının oksidasyonunu sağlayarak karbondioksite dönüşmesine imkan tanır (Kosmack vd., 2003).



Şekil 5. Termal akış dönüşüm reaktöründe aşağı doğru hava akışı (Kosmack vd., 2003).

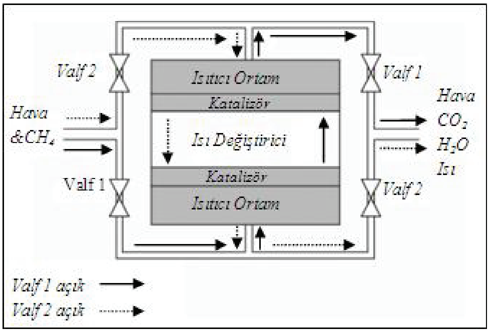


Şekil 6. Termal akış dönüşüm reaktöründe yukarı doğru hava akışı (Kosmack vd., 2003).

3.2.2 *Katalitik akış dönüşüm reaktör teknolojisi*

Çalışma prensibi olarak termal akış dönüşüm reaktöründen en önemli farkı, kullanılan katalizör olan bu reaktörler, ocak havasındaki düşük konsantrasyonlarda bulunan metanın yanma esnasında kendiliğinden tutuşma sıcaklığını düşürür ve metanın oksidasyonu ile açığa çıkan enerji süresince de sistem reaksiyonun sürekliliğini korur. Sistemdeki aşırı ısınmalar ısı değiştiricisine ya hava ilavesi ya da hava-su ilavesi ile önlenebilir. Bu tür reaktörlerin düşük sıcaklıklarda çalışması, hem NO_x gazının açığa çıkmamasını sağlar hem de mühendislik ve üretim maliyetini düşmesine sebep olur (US EPA, 2009). Şekil 7'de, tipik bir katalitik akış dönüşüm reaktörünün şematik görüntüsü sunulmuştur.

Çizelge 3'de, ocak havasındaki metanın oksidasyon teknolojilerinin (Termal, Katalitik ve Katalitik-Monolitik) bir karşılaştırması sunulmuştur.



Şekil 7. Katalitik Akış Dönüşüm Reaktörü Şematik Gösterimi (US EPA, 2009)

Çizelge 3. Ocak havasındaki metanın oksidasyon teknolojilerinin (Termal ve Katalitik Akış Dönüşüm ve Katalitik-Monolitik Reaktör) karşılaştırılması (Su vd., 2005).

Özellik	*Termal Akış Dönüşüm Reaktör Teknolojisi*	*Katalitik Akış Dönüşüm Reaktör Teknolojisi*	*Katalitik-Monolitik Reaktör Teknolojisi*
Çalışma prensipleri	*Akış dönüşüm*	*Akış dönüşüm*	*Monolitik reaktör*
Katalizör	*Yok*	*Var*	*Var*
Kendiliğinden tutuşma sıcaklığı	*1000 °C*	*350–800 °C*	*500 °C*
İşlem süresi	*Kısa*	*Uzun*	*Sürekli*
Minimum CH_4 konsantrasyonu	*% 0,2*	*% 0,1*	*% 0,4*
Uygulanabilirlik	*CH_4 Azaltılması*	*CH_4 Azaltılması*	*CH_4 Azaltılması*
Açığa çıkan ısının elektrik üretiminde kullanma potansiyeli	*CH_4 kons. arttırmak ve sabit tutmak için ek olarak yakıta ihtiyaç olabilir*	*CH_4 kons. arttırmak ve sabit tutmak için ek olarak yakıta ihtiyaç olabilir*	*CH_4 kons. arttırmak ve sabit tutmak için ek olarak yakıta ihtiyaç olabilir*
CH_4 konsantrasyonu değişebilirliği	*Değişken*	*Değişken*	*Değişken*
İşletmede yer kaplama	*Büyük yer kaplar*	*Geniş yer kaplar*	*Az yer kaplar*
İşletim	*Karışık*	*Karışık*	*Basit*
İşletim ömrü	*N/A*	*N/A*	*> 8000 saat (katalizör için)*
NO_x emisyonu	*N/A*	*Düşük*	*Düşük (< 1 ppm)*
CO emisyonu	*Düşük*	*Düşük*	*Düşük (~ 0 ppm)*

3.2.3 Katalitik - Monolitik Reaktör Teknolojisi

Bu teknolojide yüksek mekanik dayanım, büyük geometrik alan ve yüksek akış kütlesinde çok düşük basınç düşmesi gibi göze çarpan özellikleri olan ve bal peteğine benzer delikli bir Monolitik reaktör kullanılır. Burada kullanılan monolitikler, katalitik olarak aktif parçacıklar içeren gözenekli bir yapı ile kaplanmış paralel duvar yapılarından oluşurlar. Bu nedenle, diğer oksidasyon teknolojileri ile karşılaştırıldığında, aynı miktarda ocak havasının oksidasyonunda daha etkin olabilirler. Ancak, havanın ön ısıtması için diğer teknolojilerde yenileyici tabaka olmasına rağmen burada ek donanım donanım gerekmektedir (Su vd., 2005).

4 SONUÇLAR VE ÖNERİLER

Günümüzde, kömür işletmelerinin çok azı, kömür damarlarından elde ettikleri metanı kullanmaktadır. Kömür işletmelerinde metan üretim işlemleri, çalışma koşullarının iyileştirilmesi amacıyla yapılmakta ve elde edilen gaz ocak dışına çıkarılarak atmosfere salınmaktadır. Isıl değere sahip bu gazın, bu şekilde atmosfere gönderilmesi küresel ısınmaya katkıda bulunduğu gibi ekonomik olarak değerlendirilebilecek bir kaynağında israfı anlamına da gelmektedir. Bu nedenle, gazın üretim öncesi, üretim esnasında ve sonrasında drene edilerek çeşitli amaçlarda kullanımının yanı sıra, ocak havasındaki düşük konsantrasyonlu metanın ocak çıkış kuyusundan atmosfere salınması yerine çalışmada bahsedilen oksidasyon teknolojileri ile ısıya dönüştürerek enerji üretiminde kullanılması hem küresel ısınmaya faydalı yönde katkıda bulunacak hem de ekonomik kazanç sağlayacaktır.

Ocak havasındaki metan konsantrasyonunun her bir oksidasyon teknolojisi için gerekli olan minimum konsantrasyonu aştığı durumda, bu oksidasyon teknolojileri metanın bertaraf edilmesi için teknik olarak kullanılabilmekte ve ekonomik olarak da fayda sağlamaktadır. Eğer ocak havasındaki metan konsantrasyonu hemen hemen sabit ise, metan oksidasyonu sonucu açığa çıkan ısı, enerji üretimi için kullanılabilir. Metanın değişken konsantrasyonlarda bulunması, elde edilen ısının ya da CO_2'in enerji için kullanılması zorlaşır.

Ocak havasının yanma işlemlerinde ikincil bileşen olarak kullanıldığı oksidasyon teknolojilerinde özellikle toz kömür ile birlikte yanma işlemlerinde kullanılmasında bu tür santrallerin az olması, bu teknolojinin uygulamasını kısıtlamaktadır.

Oksidasyon teknolojilerinin mühendislik uygulamasında, özellikle termal akış dönüşüm reaktör teknolojisi için büyük alanlar gerekmesi sorun oluşturabilmektedir.

Ülkemizde, metan gazı ile ilgili çalışmalar genellikle iş yeri emniyeti açısından yapılmakta ve belli bir plan-programa sahip olmamaktadır. Metan drenajı aralıklı olarak gaz oranının yükseldiği durumlarda uygulanmakta ve ayağa sızması muhtemel olan gaz ortamdan uzaklaştırılmaktadır. Metan gazının sadece bir patlayıcı değil aynı zamanda da bir enerji kaynağı olduğu bilinmelidir. Henüz faaliyete geçmemiş kömür işletmelerine kömür içerisindeki metan miktarının belirlenmesine ve değerlendirilmesine yönelik çalışmalar yapmaları önerilebilir. Üretim faaliyetlerini devam ettiren kömür işletmelerinde ise gazın içerdiği metan miktarına bağlı olarak, gazı üretmeleri yada ocak çıkış kuyusundan atmosfere salmak yerine çalışmada bahsedilen oksidasyon teknolojileri ile gazın kullanılması önerilmektedir.

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Tailings Dam “Benkovski 2” – Perspectives for Development of the Equipment

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ABSTRACT “ELLATZITE-MED” is the biggest mine company in Bulgaria and its main object of activity is extraction and processing of copper-porphyr ores with gold content from deposit “ELLATZITE”. Annually the enterprises processes 12 500 000 t ore.

The deposit from filtration in the enrichment company was first disposed in the tailings dam “Benkovski 1” and its capacity was runned out in 2001.

In 1994 the company “GEOTEHMIN” designed a new tailings dam with capacity 140 000 000 tones. It is situated near the tailings dam that was active then. The construction of “Benkovski 2” began in April 1996. One of its section – “Ai dere” was ready and it started to be exploited in June 1998, the second section – “Suludja dere” was finished in 1998.

1 DESCRIPION AND SPECIFIC DATA FOR THE TAILINGS DAM

1.1 General Information

Tailings dam “Benkovski 2” is situated 75 km far from Sofia. Its southern limit is river Topolnitza and its northern limit is tailings dam “Benkovski 1” (photos.1).

As a result of the research that are made before the construction of this tailings dam it was found that till depth passed through drillings 30 – 50 m from the terrain ,the surface that encompasses the territory of the tailings dam consists of area of proteozoic rocks and quarter rocks.

Proteoizoic rocks are presented by muscovite, biotit and gneiss with two mica. There are also granate and amphybol as auxiliary materials.

Gneiss are light green to light grey and they are medium to big grain size with an injection of quarz-pegmatit. Dyke rocks are also found and they are genetically related with old mountain Ka and alkali formation. They are presented by diorite and quarz diorite. The rocks are with uniform grain size – grey-green.

The thickness of the corps is 4 – 5 m. While incorporating they reveal the gneiss and in some places they take some material from them.

The gneiss especially muscovite are such that are faded everywhere in surface and in depth. The limit between the greatly faded and medium faded is between 10 and 25 m.

The upper bead till 1 – 2 m depth the faded processes are greatly developed and the main rocks are degraded till gruss and sands with clay.

The area that encompass tailings dam “Benkovski 2” the main rocks are covered everywhere with quarter beds. They are genetically related to alluvial – deluvial materials and they are presented by claysands cumulations with edge rocks slices. The thickness of the quater accumulations vary from 0,5 to 2,0 m. Deluvial pockets are found in some places and the thickness of the sediments is up to 10m.

The collection of water of the tailings dam is characterized as medium mountained and it is completely occupied by metamorphic rocks and dykes with limited encompass and with low capacity for retaining water.

These circumstances provoque predominance of the surface flow off of comparing the underground flow off.

1.2 Hydrotransportation System

The transportation of the waste from the flotation of the enrichment complex is done completely through gravitation.

The quantity of the pulp is 1,3 m^3/s and the ratio of solid phase and the water is 1:3. The approximately weight of the pulp is 1,17 t/m^3 and the average diameter is 0,10 mm.

The hydrotransportation system includes:

– Double conduit with groove for the waste and it goes consequently in double steel tube

– A ditch for division and two steel tubes with diameter 800 mm that direct the pulp to “Ai dere” and “Suludja dere” correspondingly.

– Two steel tubes for washing the waste with diameter 700 mm

– Hydrocyclones with diameter 500 mm on stands with washing and pressurized hoses, including stopped tights.

1 – Digue for division
2 – Toe dams
3 – Pump station for move of drainage waters
4 – Pump bages

Photo 1.

1.3 Tailings Dam

Tailings dam “Benkovski 2” is divided in two separate sections: “Ai dere” and “Suludja dere”, they are predetermined by the natural relief forms – the main plies of the terrain, the right influx of the river Topolnitza.

Through a division digue (position.1 on photos.1) that is embedded in height with the construction itself of the tailings dam the two lakes with sedimentation are totally divided one from another .In practice the exploitation of the two sections is done independently one from another and it can be said that the tailings dam itself is composed from these sections.

The designed characteristics of tailings dam “Benkovski 2” are as follows:

– Beneficiary volume 100 000 000 m^3 for disposal of 140 000 000 t waste
– Occupied area 3000 dka
– Height of the tailings dam – 150 m
– Incline of the air slope – 1:4,0
– Coefficient of seismic – 0,15

1.3.1 Starter dams

The hydrocyclone begins from 4 starter dams (position.1 on figure.1) on level 620 with height from 15 to 54 m depending on the topography of the terrain. The starter dams are with stones and the screen is clay. The total volume is 1 500 000 m^3 from which 1 250 000 m^3 is stone and 250 000m^3 is clay.

1.3.2 Toe dams

They consist of 3 walls with stones (position.2 on figure.1) on level 560 with height from 25 to 30 m. The same have a drainage bed under them and it facilitates the big drainage of the accumulated waste and reduce the depression curve to the maximum. The total volume of the walls is 170 000 m^3.

All the materials for construction of the starter and the toe dams are extracted from the revealed stone and clay quarries of terrains in tailings dam “Benkovski 2” and in the process of exploitation of the equipment they are covered from the accumulated waste. By this way a new technological and economical efficiency is achieved.

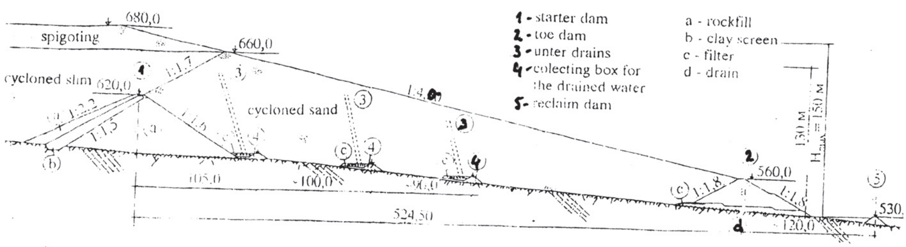


Figure 1.

1.4 Technology for waste disposal

The initial technology for waste disposal is done through hydrocyclone of the pulp and its division to sands from which is built the supporting prism with dip 1:4 of the tailings dam and the remaining cofluent is directed to become a sediment in the cup of the lake. The separation of the waste is done through hydrocyclones, situated on 10 m one from another and they divise fraction through turbulence of the pulp that goes in them, these fractions are with $D > 0{,}18$ mm on the air slope of the tailings dam and it rises this way in height with 5 m from the beach. The fine particles $D < 0{,}04$ mm from the separation are directed in the lake where they remain as sediment. The division of the waste compared to the dry weight is 45 % sands and 55 % slime.

When the beach reaches the existing working couronne the disposal of the waste will be transferred to the next section of the tailings dam. A new working couronne is built and it is 5 m higher than the old one thanks to the advanced rise of the supporting prism after that the assembling and the disassembling works related to the move of the whole conduit for washing ,the hydrocyclones and their equipment from the old to the new couronne of the corresponding section.

After assimilating the volumes between the starter and the toe dams from level 660 will begin the next stage of waste disposal through direct wash and the tailings dam will begin to rise in direction of the beach and the lake.

For every section “Ai dere” and “Suludja dere” the new stage of exploitation and disposal begins after the lake of the other section is filled.Then there begins a new working couronne and then a transfer of the waste conduit for washing from the old working couronne to the new one. This way the disposal waste in both sections of tailings dam “Benkovski 2” is done consequently in time and the same sections work independently one from another.

1.5 Drainage System

Under the sands of the walls “Ai dere” and “Suludja dere” are built drainage systems (position.4 on figure.1) that directs the drainage waters out of them and in the same time reduce the curve of depression in the corps of the walls and rise the static and the dynamic stability of the tailings dam. Through dips for collection and conduits all drainage waters are directed in the front chambers of pump station for drainage waters (position.3 on photos.1).

The aim of this is to return the waters that are in the front chambers in the lakes with sedimentation “Ai dere” and “Suludja dere” of tailings dam “Benkovski 2”. This way including these waters in the system for turn-over supply of the enterprise with water can be prevent their drop in the environment.

1.6 System for Supply of Returned Waters

After sedimentation of the waste in both cups of the tailings dam through pump bages (position.4 on photos.1) one for “Ai dere” and one for “Suludja dere” situated in the end of the corresponding lake the cleared waters are directed back to enrichment complex “Ellatzite”. They work independent one from the other on maximum distance from the front of washing. This way is guarantee the clear of the waters and their normal return to the turn over cycle of the enterprise.

Through the corresponding steel conduit which is pressurized 1020 mm the cleared water goes to collective water tower from conduit 1020 mm with length 8000 m it goes in the front chamber of an intermediate pump station. The same repump the water till reservoirs of the territory of the enrichment complex and from them is done the feeding of all technological processes in enrichment complex “ELLATZITE”, that use turn-over water.

With the closing of the cycle of supply of water to the enterprise we can achieve:

– Closed cycle of supply of water with 90 % guarantee of turn-over water
– Minimum energy consumption
– High profit
– Ecological expediency

2 ENLARGMENT OF BENKOVSKI 2

2.1 Premises and Alternatives

After the researches that were made supplementary and the prove of ore beds more than these mentioned in the contract for concession the contract was prolonged until 2022. Since the designed capacity of the tailings dam “Benkovski 2” would not encompass the supplementary 120 000 000 tones researches were done on the base of the technical project that was elaborated for enlargement of the tailings dam. The aim of this project is ”Maximum assimilation of the volume through displace of the walls of “Ai dere” and “Suludja dere”” in the direction of the air incline and the same have to encompass the valleys in the vicinity (position.7 on photos.2). The second stage of the enlargement of the tailings dam will include the change of the final level of the couronne from 680 to 700.

The natural particularities and the relief near the existing equipment predetermined the decision for the development of the object toward the two valleys under the existing tailings dam.

As an alternative to this technical project was the elaboration of investment intention and it means the choice of other place to fulfill this investment intention and construction of new tailings dam, that will be related to bigger areas and it will be necessary to built a completely new infrastructure. We have to mention also the economical and the ecological expediency of such new decision.

To pass to an enlargement of the existing hydrotecnical equipment will not be done any changes in the technology for disposal of the pulp that enters from the enrichment factory and in the hydrotransportation system itself.

Because of the fact that with rise in height of the tailings dam is disoolve the cup of the lake and in the approved technology for disposal of the waste in the project through hydrocyclon or fraction of the waste – separation of the particles with big grain size toward the air incline and in this way to construct the wall in height and the little fractions with fine particles are directed in the lake where they remain as sediment while doing the normal fulfill of the separation itself and there is an excess of hydrocycloned sands in the side of the air slope of the tailings dam in front of the hydrocyclones is accumulated the quantity that is necessary for the normal construction in height of the tailings dam hidrocycloned waste. It gives opportunity through move of the hydrocycloned to construct a transversal platforms and they are toward the air slope of the walls. This way rising in height will be assimilated bigger volumes for supplementary disposal of waste from flotation.

The second stage of disposal of the waste through direct washing will not begin from level 660, since the displace, the construction of new toe dams and the premises for new free volumes in front of the air slope of the tailings dam will be embedded in height through hydrocyclon to level 690. For direct washing will remain only the volumes in the side of the two lakes between level 690 and 700.

The big advantage of the technology related to displace of the walls "Ai dere" and "Suludja dere" consists in the fact that the move of the walls in the direction of air slope and the rise of the volume of the supporting prisms of the tailings dam, made from hydrocyclon sands will rise their static and dynamic stability and will reduce the curve of depression and will rise the opportunity to take a surface flow off of rains and it will minimize its negative influence.

The chosen decision for assimilation of the air slopes to the existing equipment is optimum from point of view of the infrastructure already built. This way the assimilation of new terrains area is minimized.

New designed characteristics of the tailings dam "Benkovski 2"

– Beneficiary volume 185 700 000 m^3 for disposal of 260 000 000 tones waste

– Occupied area 4 000 dka

– Height of the wall "Ai dere" – 190 m

– Height of the wall "Suludja dere" – 200 m

– Incline of the air slope – 1:4,0

– Coefficient of seismic – 0,15

2.2 New Equipment

2.2.1 New toe dams

While doing the displace of the two walls "Ai dere" and "Suludja dere" the waste will begin to overflow through the existing three bottom walls. This imposes the construction of new bottom walls.

The new toe dam of "Ai dere" (position.6 on photos.2) will be on 120 m from the existing toe dam and will be on level 560 with height 50 m and its volume will be 180 000 m^3.

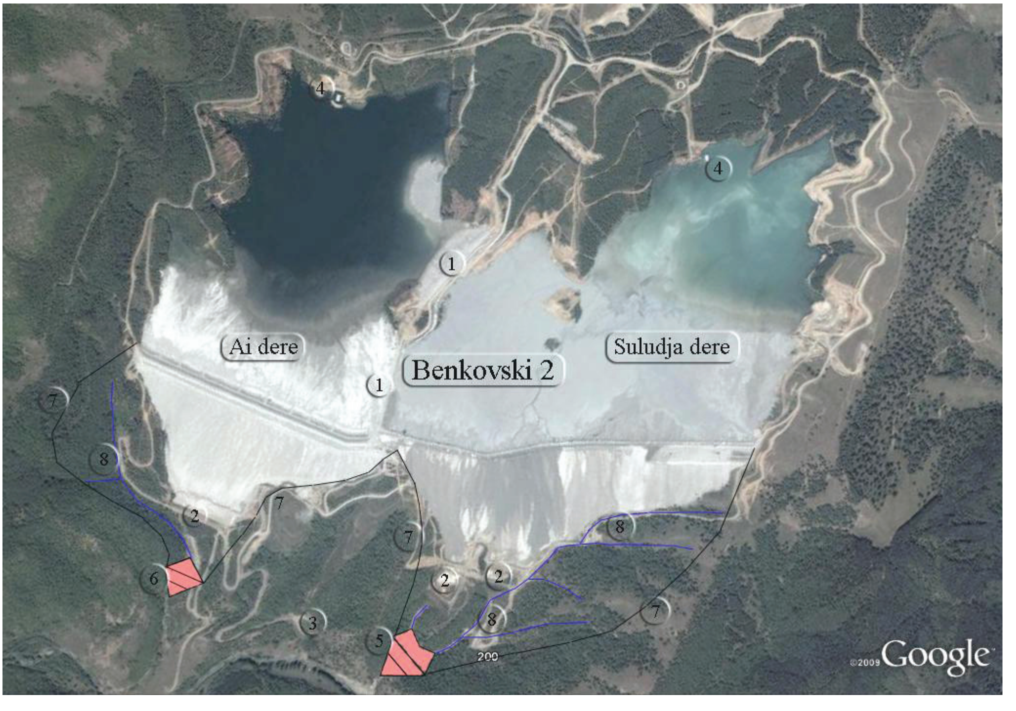
Due to the favorable topography in "Suludja dere" in spite of the existing two toe dams only one new will be constructed (position.5 on photos.2) on the place where the two valleys collect. The toe dam will stay on 300 from the existing toe dams and will be on level 560 with height 60 m and its volume will be 380 000 m^3.

The new toe dams will be filled with stones and central clay nucleus and due to the vicinity of river Topolnitza in the foot of the air slopes are foreseen to be constructed steel-concrete walls to keep the equipment from the future high waves of the river.

All the materials that are used for digues are extracted from terrains that may be splashed in the future from disposed waste. This way the influences from the environment over the surrounding terrains will be minimized.

2.2.2 New drainage systems

To reduce the negative influence on the subsoil waters and to guarantee the static and the dynamic stability of the tailings dam "Benkovski 2" on the existing plies of the terrains of the future terrains that will be splashed will be constructed linear drainages (position.8 on photos.2) and in the main valleys "Ai dere" and "Suludja dere" these linear drainages are doubled with surface drainage in the upper part and its thickness is 2m and it encompasses the valleys in transversal an lengthwise direction. All drainage waters are directed to the existing drainage pump station and from it they return back in the two lakes of tailings dam. With the whole execution of the drainage system described above will be done a normal drainage in depth of the new drainage waters from the enlargement over the auxiliary terrains outside the terrains that are occupied at present of the tailings dam.



1 – Digue for division	4 – Pump bages	7 – Profile of the tailings dam after the enlargement
2 – Toe dams	5 – New toe dam of "Suludja dere"	8 – New drainage systems
3 – Pump for drain water	6 – New toe dam of "Ai dere"	

Photo 2.

2.2.3 New piesometrical systems

While doing the displace of both walls with "Ai dere" and "Suludja dere" a big part of the existing now piesometers for monitoring the level of the curve for depression in the corpus of the walls will penetrate consequently in both lakes and it will not be more exploited .To be able to do normal and true observation of the level of drainage waters in the future exploitation of the hydrotechnical equipment will be constructed new piezometers in different stages.

3 ANALYSE AND CONCLUSIONS

The method Downstream for construction tailings dam "Benkovski 2" through hydrocyclones is highly reliable and economical. It is the most suitable for this case. The technical decision that was accepted for enlargement of the hydrotechnical equipment is with maximum technical-economical parameters at the expense of the minimum ecological influence.

The construction of new drainage systems for "Ai dere" and "Suludja dere" correspondingly as linear drainage, copy the plies of the terrains will lead to a depth and sure drainage of the tailings dam. The assimilation of the return of the new drainage waters as old drainage waters in the cycle of turn-over supply with water of the enterprise will be done through the existing pump station and it will lead to economical and ecological efficiency.

The practical division of the tailings dam of two independent sections through the digue for division between "Ai dere" and 'Suludja dere" will lead to a relative independence in the exploitation of the hydrotechnical equipment. Having in mind that both walls are feed through separate conduits with tubes we can make the conclusion that while exploit the tailings dam there exists a good guarantee to its reliable and foreseen future development.

The Pump bages assure economy of water and energy. The fact that the same work independently one from another gives sure about the feeding of the enrichment factory with industrial water.

The constructed systems for monitoring and control of the cleared waters, the drainage waters, the division, made from hydrocyclones and the analyses that are done periodically during the exploitation of tailings dam "Benkovski 2" for the static and the dynamic stability of the hydrotechnical equipment provide good guarantee for its reliable and sure future development.

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Anthropogenic Influence of the Open Pit Mining and Land Reclamation in the National Park Environment

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ABSTRACT Trachyte stone exploitation at "Kišnjeva glava" locality, which begun in 1937, does not endanger the environment in ecological sense, does not pose a threat for plants and animals, and apart from the terrain configuration changes, it did not left behind permanent consequences to the environment. The paper gives a short review of the condition at the open pit mine; furthermore, the key problems and basic demands were presented, together with a concept of technical and biological land reclamation and spatial arrangement of the mining complex.

1 INTRODUCTION

Open pit mine "Kišnjeva glava" is the only active open pit of the constructive – technical stone in Vojvodina. Trachyte exploitation at the Quarry "Rakovac" started in 1937. After 1945, due to poor condition of the equipment and lack of financial resources, works were neglected, with production decreasing each year, despite increasing needs for the technical stone.

The enterprise "Quarry – Rakovac" was founded in 1962. In 1965, it is integrated with five other road construction companies from the Vojvodina Autonomous Region into the company "Vojvodina road" – Novi Sad. Ever since, open pit mines "Kišnjeva glava" and "Srebro" near Stari Ledinci are working within the "Non-metallic mines Rakovac" Company.

During the last decade of the former Century, alongside with the dissolution of Yugoslavia, and imposing the sanctions, hard times fell on the Company, climaxing in 1999, by NATO bombing. That year, a record breaking negative production was reached, with only 23,300(m^3ROM) of excavated material. During the last couple of years production and sale were settled to approximately 200,000(m^3ROM/year).

The exploitation was chaotic, benches were not formed in a regular manner, and it is almost impossible to define the elevation of the bench planes. To the south of the pit, several smaller and larger debris are formed.

The open pit "Kišnjeva glava" is situated within the boundaries of the Fruška gora National Park, in the area of natural wealth of the Category I, i.e. the area of high importance for the Republic of Serbia. In the wider region of the open pit mine, forest communities of sprout origin (linden and beech, beech and oak, and beech, hornbeam and linden) are present, proving the significantly changed natural composition of these communities due to anthropogenic influences (Vujić *et.al.,* 1997).

Scientific research reservation "Zmajevac" is located near the open pit. The reservation was protected in 1962, as an institution for biocenological research. This area is one of the last remnants of spontaneously developed and anthropologically least changed basic forest compositions in Fruška gora. In the wider region of the mine, several other natural monuments are situated: Beočin meadows, Volcanic tuff near the Rakovac village.

2 EXPLOITATION FIELD CONDITION

The open pit was open in the zone of the point (x=5 003 411; y=7 405 839; z=366.40), with front developing from west to east. The mine is of irregular elliptic form, with wider axe 720(m) long, and the shorter axe alongside north-south direction. The width of the north-south grasp is variable, approximately 360(m) in the west, and 200(m) in the east. Vertically, the highest elevation at the outer rim lies in the southwest (debris) at 471.04(m), and the lowest at the bottom of the pit (308.11 m).

Exploitation is completed to the south, with trachyte excavated all the way to the flysch contact. The same is true for the east and the west, since the boundaries were limited by the surrounding objects. Alongside the southern final slope, due to surface decaying of the flysch building the slope, several debris, rockslides and sliding of the slope masses. This process is particularly notable in the eastern area of the southern slope. The slopes of the northern, western and the southern wing are not formed regularly. However, since they are largely formed in trachyte, there are no problems with rockslides and sliding of the material. In the south-east area of the pit, a working trench was built, enabling the gravitational discharge of atmospheric water from the open pit contour. Overall open pit mine area amounts to 154,780(m^2).

The waste was disposed by depth, at the outer deposition site, to the north and north-west. The site has an irregular shape, with several plateaus, ranging from 422(m) to 324(m). Most of the surfaces and slopes are being grasped by the natural succession. Overall surface of the deposition site is 142,978(m^2) (Vujić *et. al.,* 2002 and 2005).

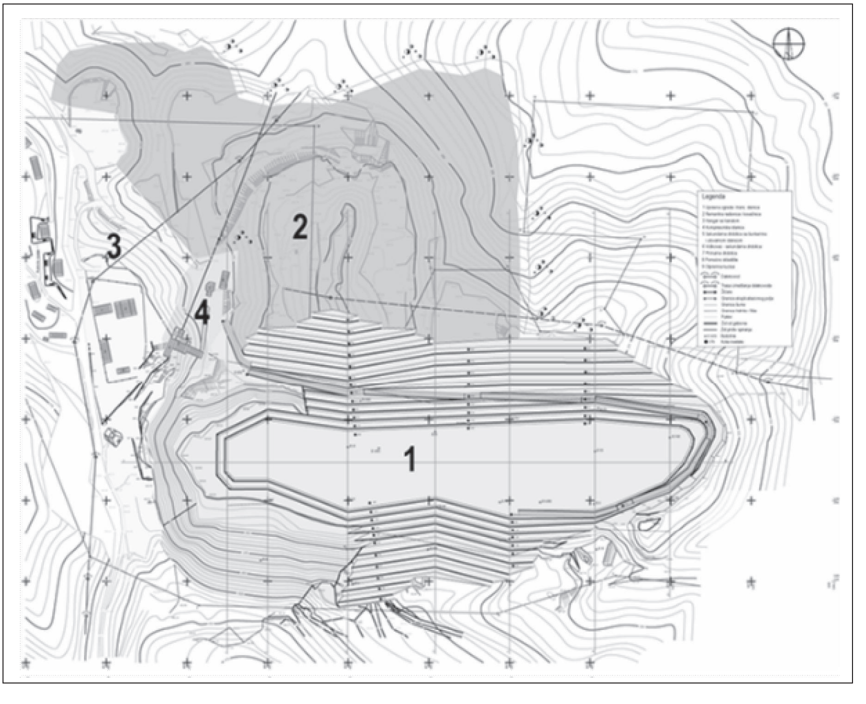

Figure 1. Functional entities.

At the plateau, to the west and north from the opening point, the objects of the primary and secondary crushing and the loading station of the cable railway. The plateau is connected to the public road Zmajevac-Rakovac, with the macadam road 240(m) long, with width ranging from 7 to 14(m) and ascending by the rate of 8(%).

At the plateau, approximately to 35(m) above and to the west of the crushing plant plateau, logistics objects are located: management building, transformer, maintenance workshop, blacksmith workshop, garages, hangers, storage rooms, carpenter workshop and the compressor station. This group of objects is directly connected with the road Zmajevac-Rakovac.

From the land reclamation and spatial arrangement point of view, there are four zones (Figure 1.) (Vujić *et.al.,* 2007):

1. Open pit Mine;
2. Deposition site;
3. Mine management area;
4. Crushing plant with aggregate deposition site.

3 ANTHROPOGENIC INFLUENCES

Trachyte opencast exploitation does not disturb the environment in ecological sense, nor it poses a threat to the plants or the animals and apart from the changes in terrain configuration, it did not left permanent consequences to the environment. The Trachyte is chemically inert and non-toxic material which does not pollute the environment. The influences of the opencast mining of trachyte on the environment in the ecological sense mostly relates to the temporary spatial and temporal degradation of the surface occupied by exploitation operations. As a consequence of trachyte open pit mining on this locality, the following environmental influences with various intensities occur:

- Slight air pollution by dust, due to drilling, blasting, dropping of trachyte during blasting, load and transport to the mobile crushing plant and its operation. The influence of dust on air pollution is assessed as relatively low.
- It is assessed that the air pollution by exhaust gases from the internal combustion engines used for drilling machinery, load, transport and auxiliary works is low. The assessment rely on the fact that the machinery work takes place inside the open pit, that the number of machines is low, that the power of the engines is low, and that the intensity of machinery operation is relatively low;
- The sources of noise are drilling machines, load, transport, and auxiliary works, mobile crushing plant and blasting. Taking the terrain configuration, geometry of the open pit mine, the vegetation and the forest in the environment into the account, and also the fact that the technological operations are taking place within the mine complex, it is assessed that the intensity of noise influence is low;
- During the mining, the negative influences are manifested also through air blasts and seismic effects. It can be concluded that the intensity of the influence on the environment is low;
- The open pit mine changes, to a certain degree, the temporary water streams of the waters of atmospheric origin. It is assessed that the influence is low;
- The erosion influence of atmospheric waters can be manifested to a certain effect along the slopes of the open pit mine. The influence is low;
- With the development of the open pit mine, a degradation of the forest land occurs. Since the open pit mine belongs to the category of smaller open pit mines, the influence is low;
- The destruction of flora and fauna is assessed as the medium to low, taking into account that the extent of open pit is not large, and that forest ecosystems are already poor due to the anthropogenic effects such as cutting down the forest, hunting etc.;
- Within the open pit depression, a formation of freezing points can be expected. This influence is assessed as medium;

- The limitation of the visual complexity holds significance, having in mind that this area is located within the Fruška Gora National Park. The influence is assessed as significant;
- The influence of taking up the environmentally protected surfaces is particularly important. By land reclamation, these negative influences can be minimized, or completely removed. It is assessed that all the measures named by the Land reclamation and spatial arrangement Project will be accomplished. In this case, the influence is assessed as low.

4 LAND RECLAMATION AND SPATIAL ARRANGEMENT

Land reclamation, in general, in opencast mining of hard mineral resources, means re-establishing of herbal associations (vegetation) on the surfaces left after the exploitation of mineral resources. It is accomplished periodically, or continually by synchronisation with the mining activities. In the conditions of the "Srebro" open pit mine, land reclamation was not performed until now.

Most often, it is not possible, nor it is necessary, to restore the area into condition identical to the original one (before the exploitation) by land reclamation. The selected variety of purpose of utilization of the degraded land must meet the needs of local community, in this particular case – especially the needs of natural habitat, stratigraphy after exploitation, expenses, etc. In order to understand this concept, a conservation concept must be apparent, main targets clearly defined (preservation of natural processes or species preservation – biodiversities), and solutions on coexistence of reclaimed land and its preservation must be found.

The purpose and organization of surfaces after the completion of mining operations consists of:

1. Change of purpose of the existing mine objects;
2. Land reclamation of the open pit depression;
3. Land reclamation of the deposition site.

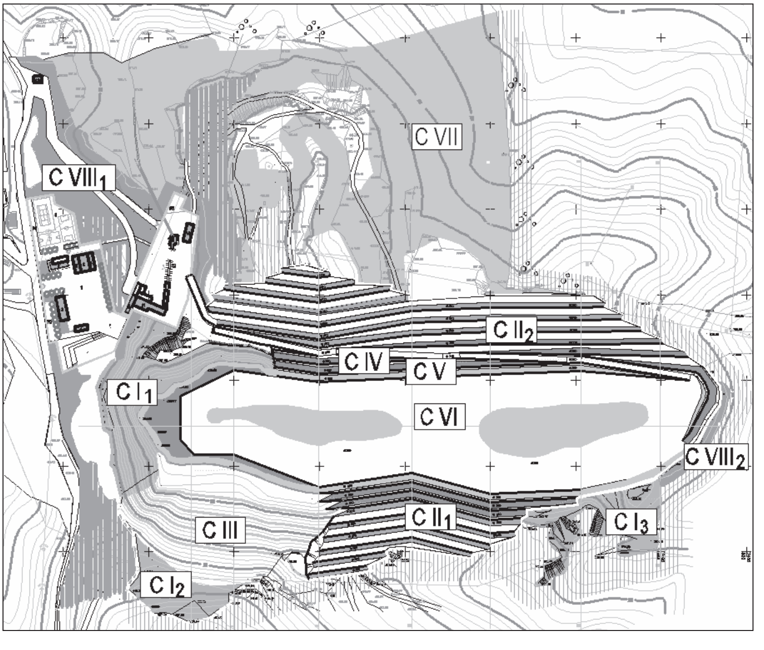

Figure 2. Functional arrangement entities inside the area.

Table 1. A concept of arrangement of functional entities with areas.

Entity	Area (m^2)	Arrangement concept
C I_1; C I_2; C I_3	28,141.40	Natural succession (trachyte) + formation of the protective greenery at the rim of the pit
C II	15,077.86	Afforestation of the newly formed benches without soil covering with protective greenery formation at the edge of the pit
C III	28,998.51	Rebuilding of debris by technical and biological measures
C IV_1; C IV_2	41,036.70	Covering the newly formed benches with the waste mixed with the source substratum and the afforestation with the formation of protective greenery at the edge of a rim.
C V	7,966.88	Rebuilding of rockslides and embankments at the rims of the open pit mine bottom by afforestation of these slopes
C VI	15,243.61	Formation of swamp ecosystems and wetlands at the bottom of the pit
C VII	87,007.14 +45,637.74	Natural succession (dumps) and meadows formation
C $VIII_1$; C $VIII_2$	21,586.90	Change of purpose of existing objects with arrangement and turning into the green the attached areas and afforestation.

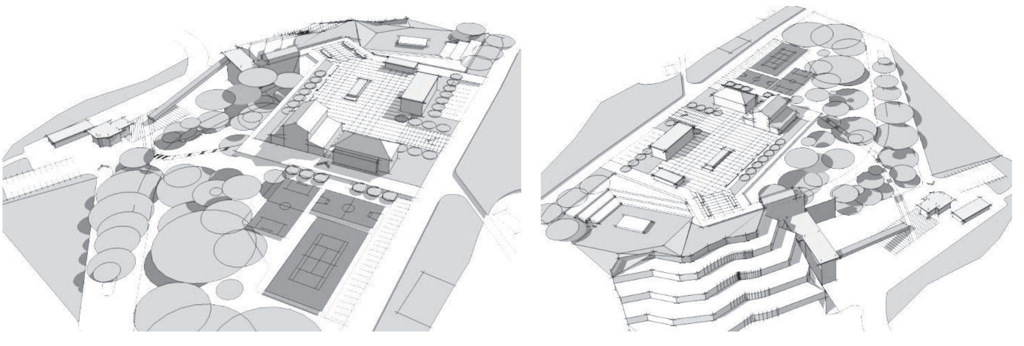

Figure 3. 3D view of the mining site arrangement.

Technical land reclamation of the open pit mine “Kišnjeva glava” as a predecessor to the biological reclamation consists of:

- Shaping the pit contour;
- Levelling and shaping the depression of the area by depth;
- Mitigation of slopes and their shaping;
- Removing the part of material at the bottom of landslide, and stopping the sliding process in the western area of the southern slope;
- Covering the trachyte benches with the solum and waste;
- Excavation, load, crushing and transport of the trachyte;
- Excavation, load, transport, deposition and levelling of the waste material within the area of the open pit mine.

Spatial arrangement of the mine complex with all the objects and infrastructure is planned as a part of the land reclamation process of the open pit mine “Kišnjeva glava” (Figures 2 and 3, Table 1). The transformation and activation of the disregarded, devastated and abandoned spaces within the mine is planned, with minimal intervention. Every spatial limitation, condition, adaptation and reconstruction costs were considered, and the change of purpose of all the objects suggested, having in mind the spatial and functional location (Vujić *et.al.,* 2007).

Since the land reclamation and turning to green the area of the Open pit Mine means creating a new, high quality surroundings, new ambience and natural entities, it was suggested that two existing groups of objects in the mine management complex (with exception of the worker's settlement) transform into two functional entities:

1. Research-ecological camp;
   The establishment of the interdiciplinary-educational camp is suggested by completing the reconstruction and adaptation of the management building complex with hangers, storage rooms and workshops.
2. Industrial park;
   Establishment of the industrial park by completing the reconstruction and adaptation of the crushing complex.

5 LESSONS LEARNED

For several years now, there is permanent pressure of ecological movements and some institutions to stop the exploitation of trachyte at the "Kišnjeva glava" and "Srebro" open pit mines. There are claims that the cessation is necessary, that exploitation is harmful and undesirable at the Fruška gora National park. At the same time, professional combat of opinions and arguments pro et contra is avoided.

The legislation is explicit in ordering the land reclamation as a procedure to return the natural functions and production capabilities to the demoted land. We believe that there are more substantial reasons to conduct the land reclamation:

1. Moral and civilization, meaning the obligation and concerns to ourselves and our descendants;
2. Ethical, since the land is renewable natural wealth, and must not be permanently demoted or destroyed;
3. Economic and social, since the reclaimed and arranged areas, after they have been restored to their natural function and the production capability, are presenting a new source of existence.

According to the Rio de Janeiro Charter, development sustainability must be directed not only to ecological concerns, but also to the social and economic development. To this end, the exploitation of mineral resources, Trachyte in this case, apart from the ecological has a particularly significant business-economy dimension, since Autonomous region of Vojvodina does not own other deposits of technical stone of such quality, necessary for civil engineering purposes. A question is asked: should the stone be transported from distant locations? Is the technical stone a resource that can bear the transport cost?

If the idea on banning the Trachyte exploitation is realized, it would be a strategic business mistake. If the problem is approached in a professionally grounded manner, suggested by the Rio de Janeiro Charter, and experiences from countries that have already gone through circumstances like this one are accepted, a compromise solution can be achieved in opposition of ecological and business-economy goals.

The compromise should be strived for, because without mineral resources there is no life, and empty tales on struggle against mining and exploitation of mineral resources are falling down with first shortages of mining products. Understandably, there is an ecological tolerance threshold that must not be violated.

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Technical and Biological Reclamation of the Topolnica Tailing Dump

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ABSTRACT Flotation waste is disposed of into a canyon type of dump. Discharge has been done in layers for more than twenty-five years that has led to the present state of the tailing dump. Flotation of copper ore is one of the production process in the Buchim Copper Mine which seriously disturbs the surrounding land.

The drastic disturbance has severely changed the chemical and physical properties of the dump which entails the need for reclamation of the newly created surrounding. Technical and biological reclamation have been carried out for several years through several phases and projects.

The experiences obtained during activities undertaken so far and the new activities and ideas envisaged to be carried out in the future will contribute to efficient actions for biological reclamation or general revitalization of the areas affected by negative impact of the tailing dump.

1 SOME CHARACTERISTICS OF THE TAILING DUMP

The tailing dump is located in the canyon of the River Topolnica extending from east to west. North of the dump is the reservoir at 609 meters above sea level. The south slope of the dump and the crest of the dam cause the biggest environmental hazards.

The entire dump surface amounts to 38.72 hectares. The part north of the crest amounts to 10.5 hectares. The crest occupies some 1.5 hectares. Some 16.72 hectares are in the south portion extending downstream the slope.

The area between the crest and peak 610 meters above sea level is an unstable part in which, now and in the coming years, waste will be deposed of as far as the planned dump height to peak 630 meters above sea level (Fig. 1).

Tailing is not deposed of any longer in the area between the lowest point in the tailing dump and peak 610 meters above sea level.

The formation of the tailing dump altered the earlier relief of the terrain. The most striking is the slope consisting of sterile dust and fine-grained sand. The fractions in settled and piled up material are easily movable and can be transported into the environment with heavy rains and wind due to their small size and volume. The most affected seem to be the Village of Topolnica and the vicinity.

The tailing dump is a waste pile consisting of two inclined parts: one towards the reservoir (the upstream side) and a slope towards the village (downstream the south slope). The inclines of the tailing slopes, particularly the one of the downstream slope are relatively large for this kind of substrate and make them rather unstable.

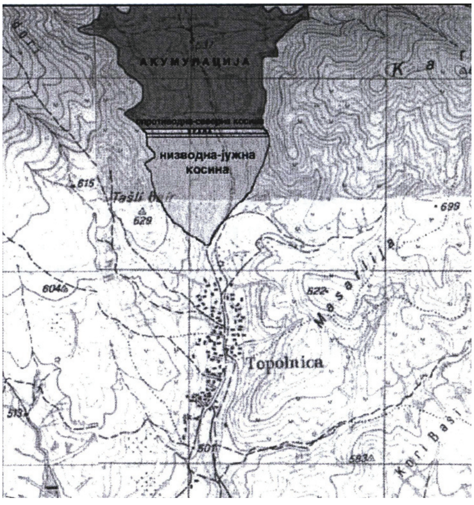

Figure 1. Location and topographic characteristics of the tailing dump.

1.1 Chemical and Technical Properties of the Tailings Dump Substrate

Determination of the properties of the disposed of substrate is necessary for successful implementation of any biological and technical activities in the tailing dump. Several samples were collected from different sites downstream the tailing dump envisaged for biological reclamation. The properties of the waste were studied applying standard methods. The results obtained are given in Table 1.

The table shows that the waste is poorly carbonate with neutral reaction (pH amounting from 6.3 to 7.08). This value means that the soil is favorable for growing of some kinds of plants.

Table 1. Agrochemical properties of the waste substrate.

Location	C_aCO_3	$CaCO_3$ (active) %	PH		Humus %	Total N %	Mg/100 gr soil	
			NKCL	H_2O			P_2O_5	K_2O
1	1.26	1.75	7.08	8.77	0.52	0.03	0.2	10
2	2.52	1.00	6.91	0.63	0.39	0.02	0.4	10
3	2.52	0.75	6.30	3.54	0.78	0.04	0.2	5
4	2.52	0.50	6.40	7.28	0.34	0.02	0.2	5

The table shows that humus and phosphorus percentages are low and according to classification this is a very poor medium regarding availability of phosphorus and humus. As regards potassium, it can be said that the waste contains average amounts of potassium for plant growth.

Considering the properties, the chemical composition of the flotation waste as well as the defined mechanical composition based on sieve analysis of tailing samples (the average sand grain size from 0.98 to 21,12 mm), it can be inferred that the substrate of the flotation tailing is favourable for biological and technical reclamation.

It is necessary to fertilise it (upon completion of agrotechnical or biological reclamation activities) with moderate amounts of nitric, phosphoric and potassic fertilisers in order to improve the chemical properties of the substrate and to check washing away processes.

2 GOALS AND METHODS FOR BIOLOGICAL RECLAMATION

The primary task in biological reclamation is taking measures and implementing activities for physical and biological stability of the tailing dump. The project for biological reclamation of the tailing dump in the Buchim mine envisages exceptional measures and activities. Experiences gained during earlier phases and the introduction of new ideas in the coming phases may contribute to more successful and cheaper reclamation work.

The implementation of projects for biological reclamation would require permanent observation and study of the results (the positive ones) particularly the kind of plants, technique and manner of implementing.

Reclamation process, beginning with the planning until the phase of final implementation and future sustainability, is a complex process that involves various kinds of activities. Actions are directed towards renewal of the reproductive properties of the damaged land, creation of new green areas that will protect and improve the human environment for the needs of the people.

The reclamation stage is divided into two phases:

- Technical - mining and

- Biological reclamation.

These activities may achieve many direct or indirect goals: to meet the legal requirements, protect the settlements and the surrounding from harmful effects of the dust and the fine sand from the tailing dump, sustainable development and improvement of the human environment, land protection from erosion, create improved and new green areas, create new landscape values, create new herb and forest varieties etc..

The concept for biological reclamation of the Buchim tailing dump will achieve creation of grass cover, afforestation of the plateau and slope terraces and their total improvement. The physical stabilisation of the dump slope and formation of terraces create conditions for biological treatment of every part of the terraces.

Basically, the success and long life of biological reclamation depend on the manner and efficiency of the implementation of physical reclamation, micro natural, climatic and pedologic conditions and factors, the selection of trees and bushes for afforestation as well as herb or greenish kinds for planting or sowing. This also includes timely and right implementation of the measures and activities necessary for growing, care, treatment and protection.

2.1 Engineering and Technical Operations and Activities

Because of the unfavourable natural and ecological conditions for biological reclamation (first of soil) the following technologies - variations of biological reclamation were used:

1. preparation and planting of contour furrows or holes, dug and prepared in the existing substrate 30 to 40 cm to depth. In this manner, the nursery plants are planted either in a fertile or a mixture of fertile and organic fertiliser the ratio being 3:1.

2. ploughing of furrows in soil - substrate 30 to 40 cm to depth and depositing a 20 cm layer of fertile soil (alluvium) in which contour furrows or holes will be prepared for planting of nursery plants. Furrowing should enable better connection between the base and the fertile soil - alluvium and create better conditions for the development of root system of plants. Each nursery plant, either in a contour furrow or hole, will be planted in a fertile soil or in a mixture of fertile soil and organic fertiliser the ratio being 3:1.

First, terracing or forming of horizontal and slope surfaces are done in the downstream slope in which substrate has been deposited.

This is done consistent with the geomorphologic characteristics of the tailing dump in order to provide successful implementation of physical and biological reclamation.

The technical reclamation or terracing of the terrain and depositing a layer of fertile soil change soil thickness, its quality and fertility. These are also important precondition for successful reclamation. (Fig. 2).

Terracing starts at peat 610 meters and ends at peak 632.5 meters. Over the past period, biological reclamation was carried out in the lower and middle parts of the slope from peak 511 meters, in the lowest peak in the slope (the foot) to peak 610 meters. New activities should start there to finalise reclamation. Biological reclamation is carried out on the whole terrace or the earlier formed slopes and on the plateau - horizontal lines.

Figure 2. Beginning phase of technical reclamation - slope terracing with a machine.

The height difference between the plateau or the flat areas of the terraces amounts to 2.5 meters. The average width of terrace plateau amounts to 9,0 meters, whereas the slope to 10.0 meters (horizontal projection α≈9.70 meters).

Technical reclamation is an important phase and is implemented prior to biological reclamation. It is important for the soil composition and the new terrain orography. It is an important condition and factor in implementing the biological reclamation. This implies that there is need for close connection between mining and biological measures - activities for the right implementation of engineering - biological operations.

Biological reclamation includes the following important phases: soil preparation for planting or sowing, selection of kinds for afforestation, method and technology of planting, the use of protective means and additives, care and protection.

Soil preparation includes depositing a 20 cm deep fertile top layer. Contour furrows and terrace slopes will be prepared on the stabilised plateau for planting (Fig. 3).

First, the soil should be furrowed in order to provide better connection between the soil and the deposited fertile layer. This will also provide better conditions for the development of the root system of plants.

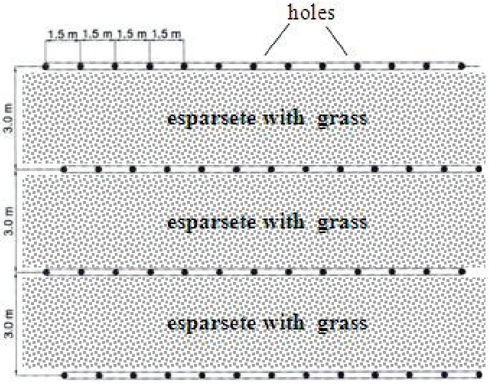

Figure 3. View of terrace with row, holes and plateau.

This will avoid failure and create better conditions for good nursery plant development and better water conservation. It will provide rational use of rainfall water and eliminate the necessity for watering.

Afforestation, is carried out in contour furrows or holes (most of all in contour furrows) filled with mixture of fertile soils - alluvium and organic fertiliser the ratio being 3:1 (Fig. 4). Nursery plants were planted in two ways: in undercut and in hole.

Figure 4. Nursery plants planted in holes.

Undercut planting is used only for container type of nursery plants possessing root system covered in peat and small in size. In earlier prepared furrows or holes fertile soil and organic fertiliser are deposited. The nursery plant is fixed by pressing or stepping over the soil round the root. When doing so care should be taken not to damage the root.

One of the major factors in the successful biological reclamation is the selection of kind of trees and bushes for afforestation. For soils of this kind, the trees selected should be such that can grow using moderate amounts of water and nourishment. This also means that the trees planted can carry out the assimilation process and develop rich leaf mass in conditions of short supply. The leaf mass will protect the soil from erosion and later, as organic waste (leaf mass - wood cover), will have a positive effect on the pedogenesis and soil evolution.

Important factors here are the substrate composition, the deposited soil (the alluvium), the relief of the newly created surfaces, climatic and microclimatic conditions as well as the plants present in the natural surrounding.

Due to the unfavourable natural conditions and the lesser number of autochthonous variety of trees and bushes, allochthonous varieties were used in the biological reclamation process. Significant effects can be achieved using numerous adopted allochthonous varieties (black locust trees, cypress etc.) which were the dominant varieties in earlier implemented biological reclamation.

Different kinds of wood varieties have been planted in the prepared area. Leaf kinds of trees such as black locust (robinia pseudoacacia), the wild olive tree (eleanus angustufolia), linden (tilia tomentosa) and some other are the predominant types.

All needle-shaped varieties are present such as cypress (cypreccus arisonica), red juniper tree (juniperus oxsecedrus) etc.

The conditions for biological reclamation made it possible to plant 3500 trees per hectare.

Besides afforestation, covering with grass is also important in biological reclamation. Covering with grass is a biological land reclamation measure which is widely used in anti erosive protection and restoration. Apart from afforestation, it is the most commonly used biological measure. It is used in all kinds of soils and climatic conditions.

Grass cover protects the land quite well from immediate strikes of rainfall drops. In addition, it has a positive effect on the process of creating structural
soils that increases the infiltration process of soil and improves the water and air system. These increase soil resistance to erosion.

To achieve the desired effect, moves are necessary to till the entire soil (between the planted rows) into belts.

The most important thing when covering with grass is good preparation. First, the surface is ploughed (shallow). Then, chopping up and levelling of the soil are carried out. Fertilisation with organic and mineral fertilisers is done to improve soil fertility. Grass sowing is best if it is done in spring or autumn. If irrigation is available, it can be done all year round except during freezing cold winter months.

It should be done in both directions, cross sowing 1 to 2 cm in depth. After sowing, raking is done to cover the seed with soil. For better and more successful sprout out, rolling of surface is recommended with a garden roller.

The methods described and carried out are rather expensive solutions owing to the use of expensive machinery for soil preparation, making of furrows and holes, the need of transport and deposition of fertile layer - alluvium and the use of fertilisers.

3 CONCLUSION

The biological reclamation of waste dumps is a complex process owing to the geomorphology (slopes that should be levelled), material that can be transported by wind and the low percentage of nutrients that the tailing contains. However, this is the unique efficient and lasting method. In the case described, biological reclamation includes tree planting and grass sowing on the surface of the tailing dump. The results achieved include as follows:

- mechanical stabilisation of deep layers in the tailing dump with the strong, deep, net of branching roots as well as stabilisation of shallow parts with the dense net of herb roots.

- in addition to preventing from dust, ground waters under the trees and grass, the organic material from the trees and grass falling down every year create a stable cover consisting of humus layer that starts the continuos authogenous process in the area.

The complex reclamation process and the need for scientific and professional understanding of the methods and techniques mentioned entail systematic monitoring of the situation and carrying out experiments and investigations for deciding on the adequate techniques for land preparation.

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Emergency Risk Situation and Risk Reduction at Four Albanian Tailing Dams

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ABSTRACT During the period 1935-1996 32 copper mines have been in activity in Albania. The copper ore production has been about 24 million ton with an average cooper content of 1.5% Cu.

There have 9 dressing plants constructed, where around 13 000 000 tons of coper ore have been processed.

For the storage of tailings from dressing plants, 13 dams have been constructed and 11 500 000 tons of tailings with an average content of 0.20 % Cu.

Due to lack of maintenance and rain waters activity 4 000 000 tons of tailings have been discharged to rivers and the present stock of tailings is 7 500 000 tons.

Current situation of the dams in Fush-Arrez, Reps, Rreshen and Kurbnesh sites is not good, the dams are not covered, are not maintained, are under the impact of the atmospheric agents like wind and rain, etc, so all these dams are in the phase of continuous destruction.

1 INTRODUCTION

Many centuries ago, in Illyrian times the minerals were extracted and elaborated in Albania. The Piruste tribes, located in Mirdita region used to extract and elaborate the mineral of copper.

The initial period of extracting industry of minerals in Albania dates the years of 30's of the twentieth century. Actually the first mine extraction of copper minerals dates 1935 in Rubik of Mirdita region. Then other coal mines were opened in Krabe of Tirana and chrome mines in Kam of Tropoja.

After the years of 50's of the twentieth century the extraction industry of minerals was developed rapidity.

During 1950-1996 the extraction and elaboration industry of minerals has these data:

- Chrome mineral for the period 1948-2008: 27.2 million tons
- Ferrochrome, starting from 1976, about: 780,000 tons
- Copper mineral, 1938-2008: 24.4 million tons
- Coal, 1938 -2008: 38.7 million tons
- Mineral Fe-Ni, 1962-2008: 15.6 million tons
- Mineral Ni-Si up to 2008: 1.5 million tons
- Gold: 0.5 tons
- Silver: 2.4 tons
- Olivine: 0.4 million tons
- Dolomite: 0.1 million tons
- Gypse: 1.1 million tons
- Rock salt: 1.4 million tons
- Phosphorus: 1.2 million tons
- Volcanic glass: 0.3 million tons
- Pure Naturel Bitumen: 0.28 Million Tons

Some of these minerals are furthered elaborated in enrichment and metallurgic processes. The mineral of chrome is enriched and treated in metallurgy. The coal extracted in Valias and Memaliaj mines has been through the enrichment processes in particular factories and in some cases the coal has been selected by hand after being extracting by the underground.

2 THE DRESSING PLANTS AND ITS TAILING DAMS

2.1 The General Consideration about Dams

As a result of the selection and the enrichment processes of chrome and coal minerals, stocks of sterile or dams of wastes.

Dams of sterile of coal enrichment factories, by their own nature, create the chances to spring the plants (as they contain nutritive elements for the plants). Dams of enrichment factories of chrome minerals do not create the condition of vegetation.

These dams are being used by the private sector for recycling so at this stage they are not stocks but part of the mineral activity. Besides, they quantity is not considerable. In Bulqize there are 2.1 million tons while in Bater there are 300 000 ton.

The copper ores enrichment factories tailing dams taking in consideration the quantity of fine material deposed there and numerous problems that it's present are totally different.

Out of 32 mines opened at the source area of copper mineral, 24 million tons are extracted with an average content of 1.3%.

Out of 24 million tons of copper mineral, 13 million tons are elaborated with an average of 1.5% Cu. Nine enrichment factories were built and have been operational.

At present, only one factory is operational, the one in Fushe Arrezit under the Turkish company named Ber Alba.

To deposit the wastes of enrichment factories of copper mineral, 13 dams were built and have been operational. In these dams are deposited about 11.5 million tons with an average content of copper 0.20%.

Nowadays in these stocks there is about 7.5 million ton. As a result of miss respect of the technology of the years 1950-1996 and lack of maintenance during the years 1996-2009 approx. 4 million ton of dam material have been discharged in the river.

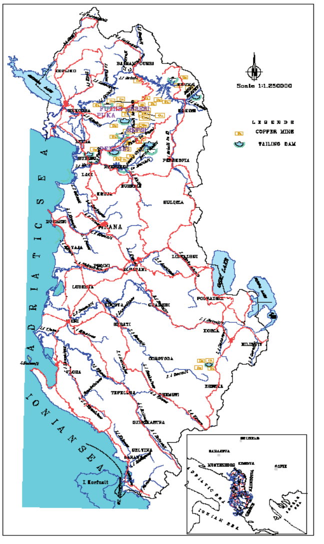

Figure 1. Location of mines dressing plants and tailing dams.

2.2 The Copper Dressing Plant Tailing Dams

2.2.1 Dams of Fush-Arrez site

The enrichment factory dams in Fushe-Arrez. This factory is located 145 km in northern part of Tirana, near by the town of Fushe Arrez. In the town of Fushe-Arrez there are some houses.

Fushe-Arrezi is a small town located at the center of the copper mineral area. In a distance of 10-20 km some copper mines used to be operational in the past, such as (Qaf-Bari, Rruga e Rinise, Tuc, Lak-Roshi, Kcire, Porave, etc). At present, there are two mines operational, the one of Munelles and Lak Roshi, both managed by the Turkish company Ber-Alba.

Depositing in this dam has started in 1976, the very same year the enrichment factory started its operation.

Depositing in the first dam has continued up to 1982. In this year it started the depositing in the dam no.2. In this dam the depositing process of enrichment factory of sterile under the management of the Turkish Ber-alba company continues.

The situation of the two dams differs. In the first dam, because of the lack of maintenance, financing and investments the depositing canal of waters is destroyed and all the raining waters of the hill around the dam go through the dam creating a canyon which is continuously getting deeper and wider.

Also, as a result of erosion power of waters the embankment of dam is almost destroyed.

The raining waters move the materials of dam and together they get discharged in the river "Fan i Madh", polluting its waters.

In these dams there are deposited 3.3 million tons, whereas the present situation is 2.1 million ton.

This shows that a quantity of 1.2 million ton sterile of dams have been discharged in the river Fan i Madh, which is located approx. 100 m away from the dams. This process of pollution increases its intensity during the raining.

Figure 2. A big part of the dam is damaged due to water erosion.

Figure 3. Canyon on the dam nr 1 Fush-Arrez created due to water erosion.

Figure 4. A part of water diversion structures and erosion canyon.

2.2.2 Dams of Reps site

Another enrichment factory of copper mineral is Repsi, a small town in the north of Tirana and 100 km away. In 1970 the first enrichment factory was built as a result of the need of elaborating the copper mineral extracted by some mines in the town of Reps (Spac, Gurth, Thirre, Maja e Madhe, Laje Kullaxhi etc.). This factory stopped operating in 1996.

In the four dams there are deposited approx. 2.8 million ton starting from 1996, while in nowadays there are 2.2 million ton. The same story has happened in the dams of the Reps, where have been cases when the wastes of the factory have been discharged in the river. Starting from 1992 and on, no maintenance or investment has taken place. This has led to a discharge of 600 000 ton sterile of dams in the river of Fan I Vogel.

Construction of Repsi dams near by the river Fan i Vogel have led to a discharge of these materials to the river because of the erosion power of waters. The worst dam is considered the one with Nr 4 which is constructed in the river, and as a result more than ½ of it is taken by the waters of the river, which will most probably lead to its total elimination.

Dam Nr 3 in Reps site: Dam 3 Reps, is a dam created by the deposition of the tails by pumping, because it lies over the level of the factory. There were deposited 790 000 tons of tails in this dam and at present there are deposited only 660 000 tons, because one part of the tails have been displaced.

The situation in this dam appears to be very problematic. The water diversion channel is completely ruined along some segments of its length and almost blocked at the entire remaining part. All the surface waters and rain waters pass over the surface of the dam, eroding continuously its material. At the front of the dam due to lack of maintenance and investments and due to the eroding activity of waters, there have been removed considerable quantities of tailings and they have discharged to the Small Fan River, which is running 70 meters far from the dam. This damp is located about 600 meters from the Reps town and close to it at the southern part there are some inhabited houses.

Figure 5. Missing part of the dam due to water erosion.

Figure 6. Dam No 4 above and No 2 below in the Small Fani River bed.

2.2.3 Dam of Rreshen site

Near the town of Rreshen, about 2 km far from it there have been constructed the dam for the depositing of the tailings coming from the processing factory of cooper ore of Rreshen. This town is located 76 km at north of Tirana. The situation in this dam is highly concerning, because, besides the destruction of the channel for the discharge of high waters, the main collector which collects the waters from the torrent is also out of function. The damage of the collector has caused also the destruction of the dam structure and there has been created two big funnels, which are becoming wider every day. This dam during the raining fills with water and creates a lake over its horizontal surface. The lack of maintenance and investments, the damage and going out of function of the channel for the collection of high waters has created the premises for the total destruction of the dam.

Figure 7. The surface of Resheni dam covered by water.

Figure 8. The view of 2 dangerous holes in Resheni dam.

Figure 9. Defect at water diversion channel.

2.2.4 Dam of Kurbnesh site

Dam of Kurbnesh, is the first constructed and oldest one, and the deposition of the tailings in this dam has started since firstly the factory was constructed at 1961. In these dams from the starting of works at 1996 there have been deposited 2.2 milion tons, meanwhile at our days there are found only 1.8 milion tons. Due to lack of maintenance, investments and due to the eroding activity of waters, the quantity of 400 000 tons has finished at the river of Uraka, which stream is near to the dam. In this dams the channel for the collection of high waters has been blocked as well, the collectors are damaged and does not function, some criss-cross canyons are created on the dam and the banks of the dam is ruined.

Figure 10. Canyon on the dam Kurbnesh created due to water erosion.

Figure 11. The hole on the Kurbneshi dam.

3 THE SITUATION IN ALL DAMS

All of them are open, not covered; there is no maintenance and investments since year 1996, excluding the dam nr 2 Fushë-Arrëz, which is maintained from the Turkish company Ber-Alb. All of them are under the impact of the atmospheric agents, especially under the impact of winds and heavy rains, and the impact increases due to their granular metric which is very fine granules from 0-200 micron (0-20 micron is 40 %).

Presence of heavy metals and other chemical elements have been proven in the dams' material content, such as : Cu 0.2%; Zn 0.046%; Fe 8.87%; S 15%; Pb 10 (mg/kg) As 6.2 (mg/kg), Ni 120 (mg/kg) etc.

Water diversion channels are out of their function, because they are destroyed.

The rain waters does not pass through the channels, but on the surface of the dam, eroding the material and removing it from the dam to the river streams, Fan and Uraka River. It is to be underlined that the dams are toward a complete destruction.

4 IMPACTS AND RISKS FROM THE DAMS

The construction of dams very close to inhabited centers and national roads, Fush Arrezi 30m, Reps 60 m, (near the high way Durres-Kukes) Rreshen 70m create the premises for massif incidents. Beside this

Figure 12. Risk of collapse by the dam of Reps site.

Their construction nears the rivers Fan I Madh, Fan I Vogël, Fan and Uraka and inside the river bed (dam nr 4 Reps) causes the pollution of the waters continuously in these rivers. As a consequence the polluted waters contains harmful elements such as acidic water (ph 2.5-5), heavy metals etc.
In the cases of strong winds, the fine material of the dams is spread way in the surrounding environment, impacting the respiratory health of the people.

5 THE MEASUREMENT TO BE TAKEN

As a conclusion we can say that these dams, being located near the rivers and close to the inhabited centers, national roads and being in a very bad technical condition, constitutes a permanent risk for causing environmental or human accidents.

In order to improve the situation and to prevent the unexpected accidents there is need of exercising serious controls on pollution level. Also urgent interventions are needed to the rehabilitation of the water diversion channels, filling of the dams in order to stop the continuation of erosion process of the dams from the winds and waters, especially at the dam nr 3 Reps and Rreshen. All these measures will be finalized with the coverage with gravel or vegetation of the dams' surfaces.

Figure 13. View of a iron nickel dam close to ohrid lake rehabilitated last years.

6 CONCLUSIONS

In these conditions, taking in consideration the risk situation at copper tailing dams in Albania, the UNDP is involved in a project preparation with the aim to improve the situation and, if it is possible the avoid totally the risk coming from these dams. The project, actually is in the preparation faze and the team which will be engaged in this project is composed by foreigner and Albanian specialist well known in this field.

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Improvement in the Explosives of ANFO Type with the Purpose of Decrease in the Toxic Emissions and Increase in the Energy Parameters

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ABSTRACT By the carried on new researches in the explosive chemical reaction of the explosives for civil purposes related to the execution of the requirements of EU and the world trends it has been found that considerable amounts of highly toxic nitrogen gasses (nitroso) are evolved at the balanced explosives of ANFO type. Apart from that, it has been found out that considerable amounts of summary hydrocarbons are evolved, which sharply changes the balance of the explosives.

A series of researches with changing in the oxygen balance form +3.5 to -7.0% have been made with the purpose of improvement in this type of explosives.

It has been found out that at work with oxygen deficient explosives, which balance is -7.0%, the summary toxic gasses $CO+NO_x$ decrease 2-2.5 time and simultaneously the detonation speed decreases over 40%.

As a result of the researches the oil-ammonium nitric explosive ANFO-L has been remade to an oxygen deficient with balance -7. It has been found out that with the remade explosive of ANFO type there are less toxic gas and dust emissions in industrial conditions.

1 INTRODUCTION

The origination and commissioning of oil-ammonium roughly-dispersed explosive mixtures increases the safety of their production and use and significantly reduces consumption of blasting operations, improves fragmentation of the rock mass and remarkably enhances the technical and economical efficiency of blasting operations.

Extending of the use of those simple roughly dispersed explosives the mechanization of blasting operations has also commenced and is fast developing now, namely mechanizing of the mixing of components, including mixing on the site of their use and mechanized charging of the blasting holes and boreholes under underground and opencast conditions.

In most of the countries all over the world, the oil-ammonium explosive mixtures are composed of 94% ammonium nitrate and 5.5–6.0% diesel oil. Under conditions of afore mentioned ratio the oxygen balance of the mixture varies from “0” to slightly negative – minus 1.66 %. Researches carried out in the USA revealed that when the content of diesel oil is decreased below 5.5%, higher quantities of orange nitrogen gasses are obtained, and the content of diesel fuel is more than 6–7%, the liberation of carbon monoxide is increased Baron & Cantor (1998). With regard to the above the recommended percentage is 6% of fuel oil.

The observations carried out at industrial sites, where large-scale blasting operations are performed, i.e. at the open pits and quarries, indicate the total release of significant quantities of orange nitrogen gases, which contaminate the environment. The analyses reveal that the liberation of higher quantities of nitrogen gases witnesses, first of all, the insufficient quantity of diesel fuel on the explosive mixture. In addition, negative effect also exerts incorrect and insufficient mixing and homogenizing of the components, violation of the technology of charging, low density of charging etc.

The quantity of released gasses depends on many factors, however the reduced quantity of diesel fuel, including the possibilities of its further reduction due to evaporation or dilution in watered boreholes and blast-holes definitely brings to significant release of nitrogen oxides.

Therefore, the topic concerning the composition of oil-ammonium explosive mixtures is of extreme importance.

2 RESEARCHES ON THE TOXIC GAS EMISSIONS ACCORDING TO THE NEW REQUIREMENTS OF THE EUROPEAN UNION

In order to study those topics we have performed some research on the released toxic gas emissions from a oil-ammonium explosive mixture, marked as ANFO, composed of porous ammonium nitrate with a very good absorbability of 7–8% of diesel fuel Kamburova (2005).

The explosive mixture is composed of 94.5% ammonium nitrate and 5.5% diesel fuel, and the oxygen balance is + 0.14 %, i.e. practically zero.

Researches have been performed according to the requirements of the new standard EN 13631– 16 in a 142 m^3 pressure chamber.

The obtained results are shown in Table 1.

Table 1.Toxic gas emissions from the explosive ANFO with 94.5% porous ammonium nitrate and 5.5% fuel oil (Oxygen balance +0.14%).

Sample no	Quantity of explosive, g	Gas emissions, l/kg explosive					
		CO_2	CO	NO_x	C_xH_y	∑ $CO+NO_x$	Усл. CO
1	1000	142	22.24	20.99	74	43.23	159
2	900	85	21.01	20.92	86	41.93	157
3	900	102	21.25	20.82	80	42.07	157
Average	-	110	21.50	20.91	80	42.41	157

The following has been established by the research:

1. When content of diesel fuel in the explosive oil-ammonium mixture is 5.5%, i.e. practically zero oxygen balance – significant quantities of the highly toxic orange total nitrogen oxides NO_x of approximately 21 l/kg are released. It should be considered that those gasses are nearly 6.5 times more toxic than the carbon monoxide and represent a powerful source for the formation of acidic rains.

2. The content of liberated carbon monoxide is nearly 22 l/kg, i.e. similarly to the quantity of nitrogen oxides.

3. The total toxic gases $CO+NO_x$ according to the new standard are nearly 42–43 l/kg.

4. When explosive chemical reaction occurs of the oil-ammonium explosive mixture with approximately zero oxygen balance, significant quantities of total hydrocarbons (CxHy) are emitted, nearly 80 l/kg. Mutual binding of the carbon and the hydrogen and their emitting in a free condition during the explosive chemical reaction of the oil-ammonium explosive mixture with approximately zero oxygen balance brings to the release of significant extra oxygen and with regard to the above the

explosive mixture becomes with a positive oxygen balance. With regard to the above an emission of significant quantities gasses occurs.

Note: In compliance with Bulgarian valid norms the concept "conditional CO" has been introduced, which is obtained according to the expression:

conditional CO =l CO+6.5 1 NO_x from 1 kg explosive.

5. In order to reduce the quantities of nitrogen gases, the oil-ammonium explosive mixtures should be composed of a higher quantity diesel fuel, of the order of 7–7.5% instead of 5.5–6%, and this will compensate the formation of hydrocarbons and reduce the quantity of toxic nitrogen gases.

The modification of the quantity of diesel fuel brings to the following modification of the oxygen balance of the oil-ammonium explosive mixtures.

Cont of AN in %	FO,%	Oxygen balance, %
94.5	5.5	+0.14
94.0	6.0	minus 1.66
93.5	6.5	minus 3.47
93.0	7.0	minus 5.27
92.5	7.5	minus 7.08

3 STUDYING THE TOXIC EMISSIONS AND INCREASING THE ENERGY PARAMETERS OF THE EXPLOSIVE, TYPE ANFO

In order to reduce the toxic emissions and increase the energy parameters a series of new researches on the explosive, type ANFO, have been performed.

The researches included modification of the oxygen balance from plus 3.5% to minus 7.0%.

A comminuted water-resistant ammonium nitrate of the type "JVK" Kamburova (2007) has been used for preparing the explosive.

The results from the new research on the release of toxic emissions, performed on the remade mix-design of the explosive, marked as ANFO – L with an oxygen balance of minus 7.0% are shown in Table 2 and Figure 1. For the purposes of comparison, the results obtained for toxic emissions from the same explosive with approximately zero oxygen balance as well as the results obtained for the ANFO-USA with an oxygen balance of minus 1.78% and the mixture 50/50% ANFO and emulsion of Sapko (2001)

Table 2. Gas emissions of booster sensitive explosives.

	Gas emissions, l/kg			
Explosives	$\sum NO_x$	CO	$\sum NO_x$ + CO	$\sum C_xH_y$
1. ANFO-L, zero % OB	21	22	43	80
2. ANFO-L remad minus 7.0% OB	11	20	31	70
3.ANFO-USA 6% DF	37	17	54	11(H_2)
4.Mix ANFO 6% DF and emulsion 50/50-USA	31	18	49	7(H_2)

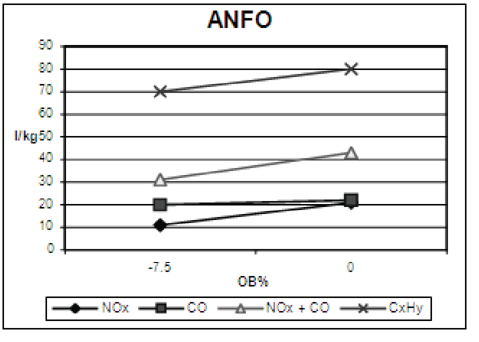

Figure 1. Dependence between the oxygen balance and the toxic emissions for the explosive ANFO–L.

The new research revealed that highly toxic nitrogen oxide emissions from the remade explosive ANFO-L with an oxygen balance of minus 7.0% were reduced nearly twice and the release of CO does not increase.

The results obtained for the modification of velocity of detonation in dependence of the oxygen balance for the explosive ANFO-L are shown in Table 3 and Figure 2.

Table 3. Velocity of detonation of the explosive ANFO-L in dependence of the oxygen balance.

Oxygen balance %	Velocity of detonation m/s
+ 3.5	2900
“0”	3600
minus 7.0	4200

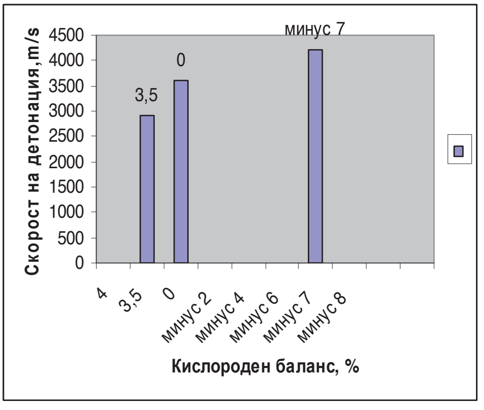

Figure 2. Dependence of the velocity of detonation and the oxygen balance for the explosive ANFO – L.

The research revealed that the oxygen balance has a significant effect on the velocity of detonation, i.e. the velocity increased with nearly 45% when the oxygen balance was modified from + 3.5 % to minus 7.0%. The above is due to the more complete explosive chemical reaction.

The research performed confirmed that due to the formation of certain quantities hydrocarbons the least possible toxic gases and improved energy parameters are obtained for the oil-ammonium explosive mixtures with a certain negative oxygen balance.

4 VERIFICATION OF THE RESULTS OBTAINED FOR THE REMADE EXPLOSIVE ANFO-L UNDER INDUSTRIAL CONDITIONS

Testing of the standard made oil-ammonium explosive ANFO-L, made with zero oxygen balance and the remade ANFO-L with minus 7% oxygen balance, produced with an increase of the fuel oil (diesel) from 5.5 % to 7.5 % have been performed under industrial conditions.

Both types of the explosive mixtures, the one with the zero oxygen balance and the one with the minus 7.0 % oxygen balance were used for industrial blasting explosion of a blasting field, which was divided into two parts. The first part of the field was charged with a traditionally balanced explosive, and the second part - with a remade explosive ANFO – L with oxygen balance minus 7.0 %. The trial explosion was carried out at the Studena quarry, and within the blasting field there are boreholes with diameter 105 mm and depth 12-15 m.

The visual effect of the explosion in the two fields is shown in Figure 3, on the left-hand side – the explosion with ANFO –L with zero oxygen balance and on the right-hand side – with the ANFO-L with minus 7.0% oxygen balance. It is visually evident that the dust cloud in the right-hand section is much smaller and much cleaner.

Figure 3. Visual effect of the gas-dust clouds of the explosion with ANFO – L with zero oxygen balance in the left-hand section and explosive with minus 7% oxygen balance in the right-hand section.

Fragments of the trial explosion are shown in Figure 4, in which the effect is more evidently represented.

Figure 4. Fragments of the development of the explosion in both sections of the Studena quarry with the explosive ANFO-L.

The large-scale industrial tests completely confirmed the laboratory tests and the correctness of the new concept of the formula for the chemical reaction of the explosion.

In compliance with the new equation for the chemical reaction of explosion we consider that analyzing and remaking the allowed explosives to a certain negative oxygen balance, including the capsule-sensitive for the underground works is recommended in order to minimize the actually released toxic gas emissions, in particular the nitrogen oxides when blasting is operated for civil purposes.

The large-scale industrial tests completely confirmed the laboratory tests.

5 CONCLUSIONS

The following may be concluded and suggested with regard the research for determining the effect of oxygen balance on the emission of toxic gases and the enhancement of energy parameters.

1. The general use of roughly dispersed booster sensitive explosives for the blasting operations in large open pits and quarries and in a number of construction sites requires the undertaking of measures for reducing the harmful gas-dust emissions, which contaminate the environment and medium, where people live.
2. The new research, complied with the new methods revealed that significant quantities of hydrocarbons are formed and free C and H_2 are liberated during the chemical reaction of explosion and with regard to the above extra oxygen is released from the so called balanced explosives with zero or nearly zero oxygen balance. Further to the above the oxygen balanced explosives in fact become unbalanced explosives, which emit large quantities of highly toxic orange nitrogen gases.
3. The analysis of results revealed that nitrogen gases are reduced twice when the explosive ANFO – L is remade from zero to minus 7% oxygen balance. Those results are confirmed for industrial conditions.
4. The remaking of oil-ammonium explosive mixtures to minus 7.0% oxygen balance definitely brings to enhancement of energy parameters, and the velocity of detonation is increase with more than 40 % compared to velocity of ANFO-L with zero oxygen balance.

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Abandoning Production in the Kanižarica Mine and Restructuring the Exploitation Area– A Case of Good Practice

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ABSTRACT With a decision from 1991, the Government of the Republic of Slovenia decided to gradually abandon and close down small mines in Slovenia. This decision affected the Kanižarica mine too. This mine had been producing brown coal for 150 years and used to be a propelling force of that region. A team of professionals, responsible for closing down the mine, had a long-term vision on future development of that region which was regarded as one of less developed parts in Slovenia. Coal production terminated in 1997, and after that the mine was gradually closed down, and old buildings were pulled down. In 2003 activities for the rehabilitation of the land started. This rehabilitated exploitation area has now assumed a completely new character and developed into a small-business zone of regional character. In the area, which extends over the surface of 95 ha, infrastructural objects have been built. New owners of plots have started building industrial and business units which will bring new jobs to the local people and enhance the development of the region which so far has been under negative impact of the mine.

1 GEOGRAPHIC POSITION OF THE KANIŽARICA MINE

The Kanižarica mine is located in the region of Bela krajina, at the south-eastern part of Slovenia. Bela krajina lies in the pocket between the Gorjanci, Kočevski rog and the Kolpa river and is mainly composed of karst. It is a region lying at the extreme south-eastern part of Slovenia, on the low karst plain of Bela krajina and extends over the neighbouring karst edge of Kočevski rob and Gorjanci hills. The geographical characteristics played an important role during human settlements in history because of relative enclosure against the surrounding regions and openness against the Panonian world. This area, located on the sunny side of the Gorjanci, is characterized by intertwining layers of Dinaric karstic features and Panonian characteristics which are reflected in special climate, local speech and traditional habits. The region of Bela krajina covers three municipalities: Črnomelj, Metlika and Semič.

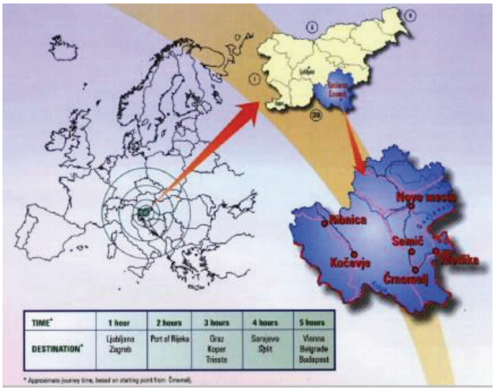

Figure 1. Geographic location of the Kanižarica mine.

2 BRIEF HISTORY OF THE KANIŽARICA MINE

The first documents on prospecting and development of the mine in Kanižarica date back to 1850. At that time it was estimated that the coal deposit was approx.15 million tons.

After the mining permit was issued on October 29, 1857, the first excavations were made 56.8 m deep, which resulted in 412 tons of coal production in l874, and 351 tons in 1875, increasing to 508 tons of extracted coal in 1876.

During WW II, Bela krajina was occupied by Italians, however the production was not disrupted until September 1942, when the partisan troops attacked the Italian invaders, and took over the mine. During the battle a fire broke out and ruined the separation unit and other wooden structures. With this event the coal production completely stopped, the cave of the mine was flooded and the mine was not operating until the end of the war.

In September 1944, the miners took actions to clean the cave and restart the production. The production of coal was very small and could only cover the needs of the local people from Črnomelj.

The mining methods used until 1958 were old fashioned, mainly longwall face, with manual transport, both in the cave and outside.

In the period from 1958 to 1963 Kanižarica was thoroughly reconstructed. During this period a new concrete haulage inclined shaft, an iron haulage tower, a haulage engine, a new reinforced concrete separation unit, and several other mine objects were put into operation.

From 1963 to 1969 a new longwall face mining method was introduced with mechanized transport of coal extraction and separation.

After the mine had been modernised the production of coal increased to 10,660 tons in 1946, and reached 143,000 tons in 1969, which was the peak in the whole history of the mine. This increasing production trend brought new job opportunities for the people: In 1946, the mine employed 94 people, while in the period when the production was flourishing, the employment increased to 400 workers. In 1974 the mine became integrated with the IGM unit which increased the number of employees from 460 to 480.

On March 1976, the mine was faced with a catastrophic inrush of water which totally flooded all mine objects: the inflow of water was approx. 1,000 litres per minute and slowly increased to 9,000 litres and eventually to 20,000 litres per minute. The production was stopped and most of the equipment remained flooded because it was impossible to rescue it with such a sudden inrush of water.

Attempts were made to pump the water out and to clean the remaining mud from the mine. Thus by June 16, 1977 approx. 300,000 cubic metres of water were pumped out.

In 1991, when Slovenia became independent, the Government of RS decided to close down smaller mines in Slovenia, and Kanižarica was one of them. According to the plans for closing down the mine coal extraction was gradually reduced and the last amount of 20,000 tons was extracted in 1996. Thus in 1997 the mining operations in Kanižarica finally stopped.

Figure 2. Aerial photography of the Kanižarica mining region in 1992.

3 CLOSING DOWN THE KANIŽARICA MINE

With government decision the activities for closing down the mine started. This project was entrusted to a group of young professionals who understood the mining tradition of the region and the problems of the local people living around the mine. The closure of the mine was divided into three phases:

- closure of the mine pit,
- rehabilitation of the land within the mine region
- human resource social management program

3.1 Closure of the Mine Pit

Within this phase, which was running from 1994 to the end of 1996, the activities for closing down the cave were running simultaneously together with some final excavations in various parts of the mines. The miners were redeployed to other works, which included:

- dismantling and removal of machines and,
- setting up hydrological barriers,
- construction of objects for monitoring.

3.2 Rehabilitation of the Land Within the Mine Region

Rehabilitation of the degraded land included the removal of all mine buildings and remediation of the exploitation area in such a way as to provide safe environment for the people, the animals and preparing grounds for the development of the elementary infrastructure for the future business park.

Figure 3. Demolition of the separation unit.

Rehabilitation of the degraded land was carried out in the following steps:

- rehabilitation of the spoil heap,
- rehabilitation of the area around the mine,
- rehabilitation of the area above the excavation fields.
- rehabilitation of the Mlake region.

Part of the rehabilitation plan was also to turn some mining objects into a mining museum.

The spatial plans of this new remediated area envisaged a business park in which the inhabitants of the region would find opportunities for developing new businesses which would consequently bring new jobs, and thus the park would become the driving power of the development of region which has otherwise been neglected for a whole century.

3.3 Rehabilitation of the Spoil Heap

As a result of mining operations, several spoil heaps have been formed in the area above coal deposit. The spoil heaps extended over the area of 22,000 m^2 without slopes. The estimated amount of debris was approx. 71,000 m^3.

According to regulatory plans from 1995, a great part of the area of spoil heaps would be used for the business park, while the remaining part would be turned into a landscape after land rehabilitation,.

Rehabilitation of the spoil heap included the following activities:

- removal of biological waste,
- regulation of leachate and water streams,
- levelling out the spoil heap and enforcement of slopes,
- recultivation of the spoil heap.

The costs for the rehabilitation of the spoil heap were approx. 490,000 €

Figure 4. Rehabilitation of the spoil heap.

3.4 Rehabilitation of the Area Around the Mine

Within the project of rehabilitation of the area around the mine, some old and useless objects, (e.g. separation unit), were removed and the area was recultivated. Some objects, e.g. the old machine room, the haulage tower and adit were refurbished and were turned into a mining museum.

Rehabilitation of a 25 ha surface around the mine included the arrangement of the landscape and building of the basic infrastructure for future users.

The costs for the rehabilitation of the space around the mine amounted to 640,000 €

Figure 5. Partially refurbished area of the previous mine in 2005.

Figure 6. Rehabilitation of the spoil heap in the region around the mine in 2009.

3.5 Rehabilitation of the Land Above the Excavation Fields

Rehabilitation of the space above the excavation fields included filling up the basins, which were formed as a result of subsidence of the land, and horticultural arrangement of the landscape. Together with the arrangement of the landscape the basic infrastructure was built for future users of plots in the business park.

The costs of rehabilitation of the land above the excavation fields amounted to approx. 525,000 €

Figure 7. Rehabilitation works above the excavation fields in 2000.

Figure 8. Remediated land above the excavation fields in 2009.

3.6 Rehabilitation of the Mlake Region

Part of the land above the excavation fields is a pond Mlaka: it was turned into a fishing farm for growing fish in a natural environment. The banks of the pond were reinforced by building slopes, the tributaries were regulated and the banks reinforced and stabilised. Nowadays the pond is being taken care by the local fishing society from Črnomelj.

The costs of rehabilitation for the pond Mlake amounted to 246,000 €

Figure 9. Remediated area of the pond Mlaka.

3.7 The Museum

Many older people still remember the times when the Kanižarica mine was in operation, providing existence to many generations. With modern developments remnants from the old times of mining could soon be lost. To preserve the memory of a 150 year mining tradition it was decided to turn some mine objects into a museum. These include: engine room of the haulage system, haulage tower and adit, which was the entrance to the mine. A roof cover where mining equipment and tools will be displayed will be built anew.

The costs for building the museum amounted to 175,000 €

Figure 10. A restaurated haulage tower with engine room.

4 HUMAN RESOURCE MANAGEMENT PROGRAMME

With closing down the mine special care was paid to the problem of surplus workers: programmes for additional education and training were introduced for redeploying the employees. With this human resource reconstruction program older workers were retired while some of them chose to receive severance pay to set up their own new businesses. Thus all miners managed to get new jobs, or were retired. This programme prevented social collapse of the region.

	Total	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004
No of employees as of Jan 1st.		191	158	94	66	45	30	23	18	18	16
Retirements	**71**	19	12	7	13	13	2	3		1	1
New programmes	**41**	8	1	20	7	1	4				
Surplus staff – severance grants	**63**	6	51	1		1	2	1		1	
Other reasons	**2**				1			1			
New staff	**-1**						1				
No of employees as of Dec 31st		**158**	**94**	**66**	**45**	**30**	**23**	**18**	**18**	**16**	**15**

From 2004 to date, the last 15 workers completed their careers in the Kanižarica mine: some of them retired, others got jobs in the new business park which emerged on the territory of the previous mine.

5 CONCLUSION: CONSTRUCTION OF A NEW BUSINESS PARK

On the remediated land of the Kanižarica mine a new business park has developed. Some companies have already built their premises and production units where more than hundred people from the region got new jobs. Considering that this region is regarded as less developed, such a development was extremely important for the people and the community as a whole.

People, who have been living in the region around the mine for more than 150 years got new opportunities for employment and can now work with modern, environmentally friendly technologies.

A group of experts from various fields, who all originated from mining tradition, understood well what closing a mine means for the economy of the region. Driven by enthusiasm and devotion they managed to create new possibilities for the development of this region.

Figure 11. Layout of the business park built on the territory of remediated mine Kanižarica.

Figure 12. Business park in 2006.

Figure 13. Business park in 2009.

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On Cleaning of Mining Waste Waters

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ABSTRACT The development of the human society leads to an increase of the water consumption, together with an increased volume of industrial and home waste waters.

The framework Directive 200/60/EC of the European Union regulates the amount of pollutants in waste waters that can be discharged into the natural emissary.

This paper presents a new method for the cleaning of waste waters. It relies on flotation with discharged air.

There is described the installation by hypobaric flotation, the laboratory tests and the improvement of the cleaning process.

With the view to getting as much useful information as possible from the flotation process, there have been used modern research tools: the test were scheduled in compliance with the suitable flowchart and an optimization of the process by the method of gradient.

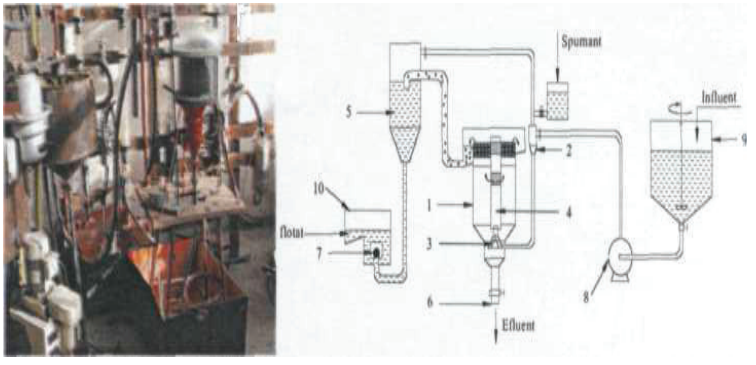
1 CLEANING INSTALLATION

There has been produced a hypobaric flotation installation for the separation of polluting particles from the industrial mining waste waters that allow cleaning by flotation.

The cleaning installation by hypobaric flotation is made of the following parts: hypobaric flotation cell; hydro-aeration device; conditioning installation; device for separation of the floated particles.

There have been produced two alternatives of this installation, i.e.:

a) hypobaric flotation installation with the supply of the influent by the centrifugal pumps.



Legend:
1-Flotation cell
2-Hydro-aerating device
3-Accelerator
4- Central adhesion tube
5- Decanting pound;
6-discharging device;
7- Valve
8- Centrifugal pump
9- Conditioning installation
10- Hydraulic closing device

Fig. 1 Cleaning installation by hypobaric flotation with the supply of the influent by the centrifugal pump.

b) hypobaric flotation installation with the supply of the influent by pressurization with compressed air

Fig. 2 Hypobaric flotation installation with the supply of the influent by pressurization with compressed air.

These two types of the installation were used during the laboratory determinations. The difference between them is the supply manner of the influent: the first alternative of the cleaning installation has the supply by centrifugal pump and the second one has the supply of the influent by pressurization with compressed air.

2 LABORATORY DETERMINATIONS ON THE CLEANING OF THE WASTE WATERS

For increasing the cleaning output of waste waters with the help of the hypobaric flotation installation, a set of 6 series of testing for the cleaning of industrial waste waters has been conceived.

The testing approach has led to the flowchart shown below:

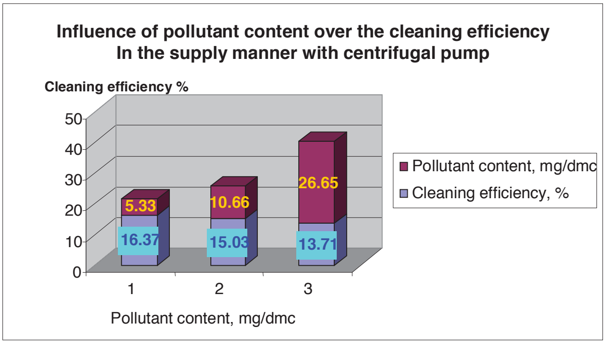
	1. Supply of the influent by the centrifugal pump		**2. Supply of the influent by pressurization with compressed air**
A	– Different content of pollutant (5, 33mg/dm^3 H_2O, 10,66 mg/dm^3 H_2O, 26,65 mg/dm^3 H_2O) – Constant supply pressure	**B**	– Different content of pollutant (5,33mg/dm^3 H_2O, 10,66 mg/dm^3 H_2O, 26,65 mg/dm^3 H_2O) – Constant supply pressure
C	– Different content of pollutant (10,66 mg/dm^3 H_2O) – Different supply pressures – (0,10 MPa, 0,2MPa, 0,3 MPa)	**D**	– Different content of pollutant (10,66 mg/dm^3 H_2O) – Different supply pressures – (0,10 MPa, 0,2MPa, 0,3 MPa)
E	– Constant content of pollutant (10,66 mg/dm^3 H_2O) – Constant supply pressure (0,2MPa) – Addition of different tensoactive substances (Gen Spumar, P1,P2, P3)	**F**	– Constant content of pollutant (10,66 mg/dm^3 H_2O) – Constant supply pressure (0,2MPa) – Addition of different tensoactive substances (Gen Spumar, P1,P2, P3)

2.1 The A Type Series of Testing

For the case of A type series of testing, the influent (made up of a mechanically mix of drinking water and suspended matters) has been supplied with a centrifugal pump. In order to be able to notice the influence of the pollutant there were cleaned out influents with a different content of pollutants (suspended matters).

2.1.1 Testing cards A

1-Type of influent: mechanically mix between suspended matters and drinking water
2-Amount of pollutant (oil)in the influent: 5,33 ; 10,66 ; 26,65 mg/dm^3
3-Presence of the ens active substance: NO
4-Type of the tens active substance: -
5-Outflow of the tens active substance: -
6-Supply pressure of the influent to the hydro aerating device: 0,2 MPa
7-Supply manner of the hydro aerating device: with the centrifugal pump.

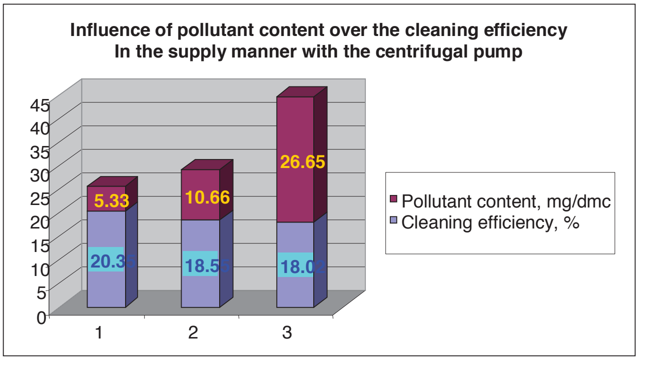


2.2 The B Type Series of Testing

For the experimental approach related to the B type series of testing, the influent simulated the waste waters came from the mining industry processing plant having different levels of pollution.

2.2.1 *Testing cards B*

1-Type of influent: mechanically mix between suspended matters and drinking water
2-Amount of pollutant (suspended matters) in the influent: 5,33 ; 10,66 ; 26,65 mg/dm^3
3-Presence of the tens active substance: NO
4-Type of the tens active substance: -
5-Outflow of the tens active substance: -
6-Supply pressure of the influent to the hydro aerating device: 0,2 MPa
7-Supply manner of the hydro aerating device: by pressurization of the influent with compressed air.

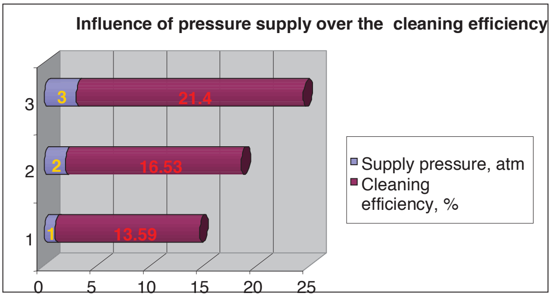


2.3 C Type Series of Testing

For the case of C type series of testing, the influent (made up of a mechanically mix of drinking water and suspended matters) has been supplied with a centrifugal pump. In order to be able to notice the influence of the supply pressure, there were cleaned out influents at different working pressures.

2.3.1 *Testing cards B*

1-Type of influent: mechanically mix between suspended matters and drinking water;
2-Amount of pollutant (suspended matters)in the influent: 10,66 mg/dm^3;
3-Presence of the tens active substance: NO;
4-Type of the tens active substance: -
5-Outflow of the tens active substance: -
6-Supply pressure of the influent to the hydro aerating device:0,1; 0,2; 0,3 MPa
7-Supply manner of the hydro aerating device: with the centrifugal pump.

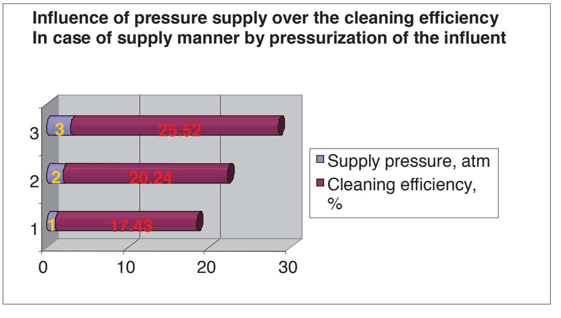


2.4 D Type Series of Testing

For the experimental approach related to the D type series of testing, the influent simulated the waste waters came from the coal processing plant. The influent was pressurized with compressed air at different working pressures.

2.3.1 *Testing cards D*

1-Type of influent: mechanically mix between suspended matters and drinking water;
2-Amount of pollutant (suspended matters)in the influent: 10,66 mg/dm^3;
3-Presence of the tens active substance: NO;
4-Type of the tens active substance: -
5-Outflow of the tens active substance: -
6-Supply pressure of the influent to the hydro aerating device:0,1; 0,2; 0,3 MPa
7-Supply manner of the hydro aerating device: by pressurization of the influent with compressed air.

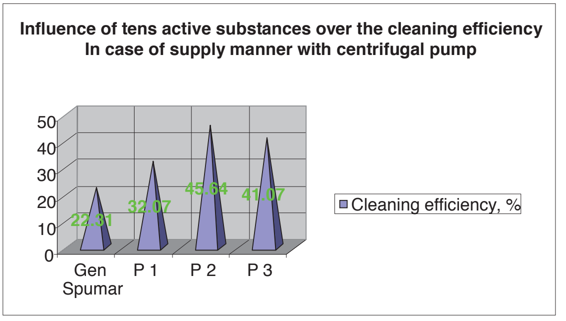


2.5 E Type Series of Testing

For the case of E type testing, the influent was made up of a mechanically mix of suspended matters and drinking water and was fed with the centrifugal pump.

2.5.1 Testing cards E

1-Type of influent: mechanically mix between suspended matters and drinking water;
2-Amount of pollutant (suspended matters)in the influent: 10,66 mg/dm^3;
3-Presence of the tens activ substance: YES
4-Type of the tens activ: Gen spumar; PI; P2; P3;
5-Specific consumption of the tens activ substance : 0,66 cm^3/dm ; 0,17 cm^3/ dm^3; 0,033 cm^3/ dm^3;
6-Supply pressure of the influent to the hydro aerating device: 0,2 MPa
7-Supply manner of the hydro aerating device: with the centrifugal pump.

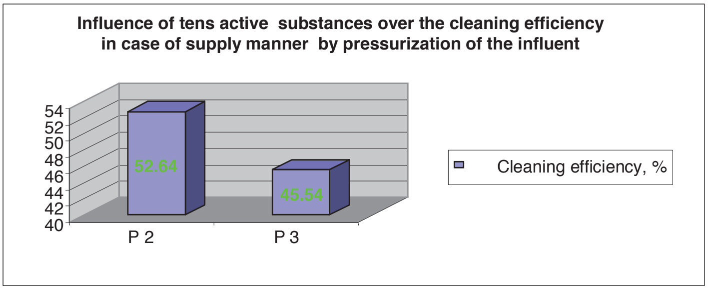


2.6 F Type Series of Testing

For the experimental approach related to the F type series of testing, the influent simulated the waste waters came from the oil processing plant. The influent was pressurized with compressed air at different working pressures.

2.6.1 Testing cards F

1-Type of influent: mechanically mix between suspended matters and drinking water;
2-Amount of pollutant (suspended matters)in the influent: 10,66 mg/dm^3;
3-Presence of the tens active substance: YES;
4-Type of the tens active substance: foaming agent P2; P3;
5-Specific consumption of the tens activ substance : 0,033 cm^3/ dm^3;
6-Supply pressure of the influent to the hydro aerating device: 0,2 MPa
7-Supply manner of the hydro aerating device: by pressurization of the influent with compressed air.



The great amount of data allowed an optimization of the cleaning method with the help of the gradient method though a three-factor experiment in a semi-replica for mining waste waters. The stages of this method are presented below:

1. Selection of the answer function for the utilization variables and of the basic level A_o;
2. Selection of the unitary changes;
3. Selection of the matrix for scheduling of experiments;
4. Determination of the coefficients of regression;
5. Determination of errors duet o experiments;
6. Checking the statistical importance of the coefficients of regression;
7. Checking the hypothesis on the linearity of the answer surface.

3 CONCLUSION

Water is the most important natural resource but, as a result of human activities, it becomes polluted with organic and inorganic and toxic substances. The optimization of the cleaning process by hypobaric flotation has increased the output of the instalation from 53,23 % to 71,73 % when the instalation is fed with a pressure of 0,33 MPa with a conditioning period of 27,6 min and a specific consumption of P2 foaming agent of 0,035 cm3/dm3.

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Kömür Açık İşletmelerinde Optimum Arazi Rehabilitasyon Yönteminin Belirlenmesi

Determination of Optimum Rehabilitation Method in Open Pit Coal Mines

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ÖZET Madencilik sektörü diğer endüstriyel işletmelerden farklı olarak, doğal çevre ile çok daha sıkı ilişki halindedir. Ülkemizdeki toplam linyit üretiminin yaklaşık %90'nın açık işletme yöntemiyle gerçekleştirildiği dikkate alınırsa, madencilik faaliyetleri nedeniyle bozulan arazilerin oldukça önemli boyutlara ulaştığı açıkça görülecektir. Geometrileri örtü kalınlığına, işletme büyüklüğüne ve seçilen açık işletme üretim yöntemine bağlı olarak bozulan bu arazilerin, yeniden düzenlenmesi ve kullanıma sunulması çevresel etki değerlendirme bakımından büyük önem taşımaktadır.

Madencilik faaliyetleri sonrası bozulan arazilerin yeniden düzenlenmesi ve kayma riski taşıyan pasa şev yüzeylerinin stabilitesi için, Müller Resonans Sıkıştırma (MRC), Ağırlıkla Dinamik Sıkıştırma (DYNIV), Titreşimli Dinamik Sıkıştırma (RDV) gibi çeşitli arazi rehabilitasyon yöntemleri geliştirilmiştir. Yapılacak rehabilitasyon çalışmaları ile şev kayma etki sahasının azaltılması, özellikle bu arazilerin yeniden düzenlenerek doğal bir peyzaj görünüm kazanması kadar, eski ekolojik ve ekonomik değerlerine kavuşturulması da mümkündür. Bu maksatla yapılan çalışmada; önemli tehlike potansiyeli içeren şev duraysızlıklarının oluşum koşulları, arazi rehabilitasyon yöntemleri ve optimum kullanım alternatifleri tanıtılacaktır.

ABSTRACT Mining, unlike other industries are much connected with its natural environments. In Turkey, since about 90 % of the overall lignite mining has produced by open pit mining technique, the magnitude of the size of these lands that require reclamation are significant. These land fields of which geometries are deformed depending on the overburden thickness, the size of the operation and the mining technique selected directly impact the effectiveness of the reclamation process and in turn proper preparation of the land for public use.

Therefore, It is highly important to predict the stability behavior of these land fields. In this study, diffrent land rehabilitation methods such as Müller Resonans Compaction (MRC), Weight Dynamic Compaction (DYNIV), Vibration Dynamic Compaction (RDV) were developped which will elucidate the risk of slide / flow behavior of these open pit mining land fields. It is our hope that this work will shed light to understand and predict the risk of slope stability so that these masssive land fields can be safely used for alternative public serve.

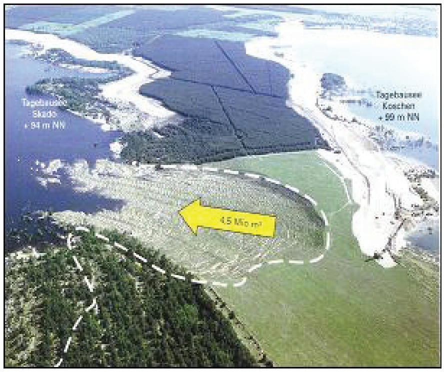
1 GİRİŞ

Ülkemizdeki toplam linyit üretiminin yaklaşık %90'nın açık işletme üretim yöntemiyle gerçekleştirildiği dikkate alınırsa, madencilik faaliyetleri nedeniyle bozulan arazilerin oldukça önemli boyutlara ulaştığı açıkça görülecektir (Ünal vd. 1994, Ünver ve Kara 1994). Gerek işletme güvenliği gerekse çevresel etki değerlendirme (ÇED) bakımından, bozulan arazilerin yeniden düzenlenmesi ve toplum yararına kullanıma sunulması yasal bir zorunluluktur. Farklı kullanım alternatiflerinin tasarlanabileceği bu arazilerin, başta jeoteknik bakımdan duraylı (stabil) ortam koşularına sahip olması gerekmektedir (Pierschke ve Boehm, 1996).

Dekapaj malzemesinin gevşek yapısal ve oluşum özellikleri nedeniyle önemli bir kısmı kayma veya akma eğiliminde olmaktadır. Bunun en önemli koşulu, boşluk suyu basıncının artmasıyla kayma gerilmesinin azalması ve statik denge durumunu bozacak yeterli büyüklükteki bir iç veya dış kuvvetin şev yüzeyine etkimesidir (Şek. 1). Çok kısa zaman dilimi (birkaç saniye veya dakika) içerisinde ani hızlara ulaşan ve büyük bir enerji boşalmasıyla sonuçlanan bu tür şev kaymaları, önemli tehlike potansiyelleri içermektedir (Kuyumcu, 2005). Özellikle bu tür kaymaların büyük bir tehlike potansiyeli içermesi, klasik kayma öncesi belirtiler olarak yorumlanan;

- şev bölgesindeki hareketlerin yavaş yavaş ilerleyerek artması,
- şev'in göbek oluşturması,
- şev gerisinde gerilme çatlaklarının oluşması gibi

alışılagelmiş belirtiler göstermemesi ve ani meydana gelen bir kaymanın zaman diliminin tam olarak tahmin edilememesinden kaynaklanmaktadır. Bu nedenle, açık işletme sonrası bozulan arazilerin rehabilitasyonu ve pasa yığınlarında oluşabilecek kayma riskinin göreceli bir tahmini için, öncelikle şev duraysızlık oluşum koşulları ve etki sahasının belirlenmesi gerekmektedir.



Şekil 1. Koschen-Skado (Almanya) linyit işletmeleri şev duraysızlığı (LMBV, 1996).

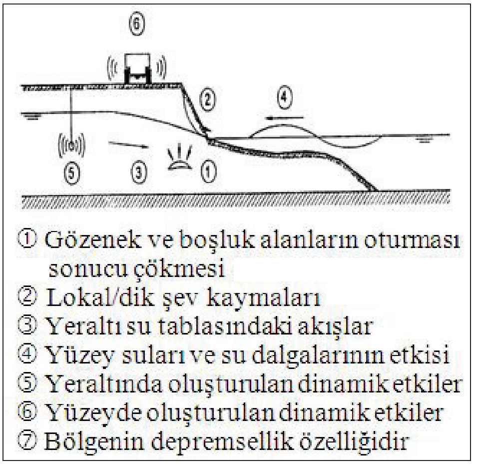
2 PASA ŞEV DURAYSIZLIK OLUŞUM KOŞULLARI VE ETKİ SAHASI

Kömür açık işletmelerinde muhtemel bir pasa şev duraysızlığının oluşabilmesi, aşağıda belirtilen koşulların açığa çıkmasıyla mümkündür. Bunlar;

- Yığın dane boyut dağılım eğrisinde bir dane büyüklüğünün baskın olarak gözükmesi,
- Yığındaki danelerin büyük bir kısmının iyi yuvarlaklık ve kaygan yüzey özellikleri göstermesi,
- Ortamın gevşek veya çok gevşek yığın halinde bulunması,
- Boşluk ve gözeneklerin tamamen su ile dolu olması,
- İç veya dış etkiyle şev'in kritik denge durumunun bozulması,
- Dinamik kuvvetlerin etki büyüklükleri ve sıklığıdır.

Eğer pasa şev duraysızlığı için yukarıda belirtilen gerekli tüm koşullar oluşmuş ise, yığının denge koşullarını bozacak bir iç veya dış kuvvetin yüzey'e etkimesiyle her an kayma başlayabilir. Kaymayı kolaylaştıran olası etki kuvvetleri Şekil 2'de verilmiştir.

Burada, sadece dinamik kuvvetlerden ⑤ ve ⑥ etkisi zamansal olarak kontrol edilebilir. Diğer etki faktörlerinin her hangi bir zamanda gerilme değişimine neden olup olmayacağı, bir kayma oluşturup oluşturmayacağı ve ne zaman meydana geleceği henüz belirsizdir. Ancak; şev kayma riski, işletme koşulları ve ortam büyüklüklerine (geometri, dekapaj sistemi, in-situ durum vb.) bağlı olarak, nitel ve ampirik bağıntılar ile göreceli olarak belirlenebilmektedir (Kızıl ve Köse, 1995; Ulusay, 2001).



Şekil 2. Açık işletme pasa şev duraysızlığına neden olan etki kuvvetleri (Förster ve Jennrich, 1999).

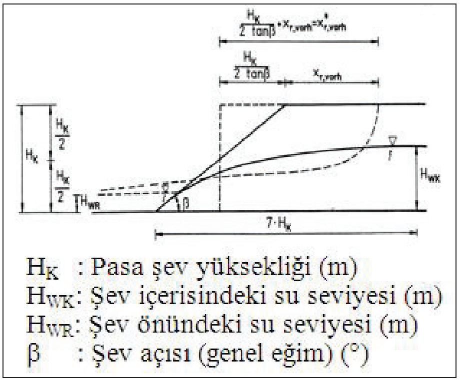
Bu amaçla olası bir kaymanın şev gerisine doğru etki sahasının yani, riskli bölgenin sayısal esaslara göre boyutlandırılması gerekmektedir. Pek çok etki büyüklüğüne bağlı bu problemin çözümü için pratik düşünceler;

- ✓ Gözlenmiş duraysızlık etki uzunlukları ile muhtemel bir kaymanın değerlendirilmesi,
- ✓ Teorik bir modelleme ile geriye dönük analiz sonuçlarının karşılaştırılmasıdır.

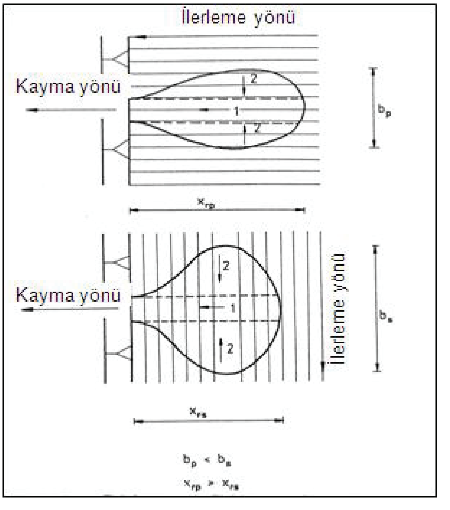
Buna göre; mevcut etki uzunluğu $X_{r,vorh.}$ (Şek. 3), belirlenen teorik etki uzunluğu ile ilişkilendirilir ve bu ilişki ilgili etki büyüklüklerine bağlı olarak ifade edilebilir.

Kaymanın belirlenmesinde önemli diğer parametrik büyüklükler ise;

- Etkinin tür ve şiddeti,
- Döküm teknolojisi (ilerleme yönüne göre kayma yönünün durumu; (p) paralel veya (s) dikey (Şek. 4),
- Kayma koşulları (Şek. 5), önlemsiz (u), önlemli (b).

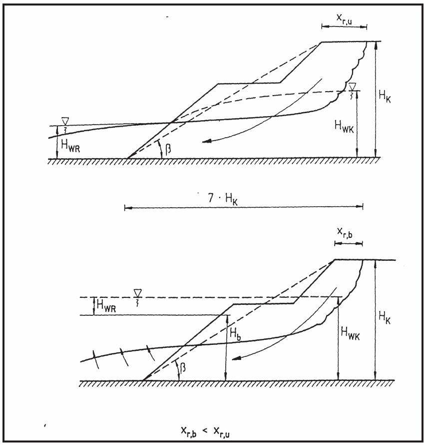


Şekil 3. Kayma etki uzunluğunun ($X^*_{r,vorh}$) belirlenmesinde parametrik büyüklükler



Şekil 4. Döküm teknolojisine göre kayma etki uzunlukları (X_r); (p) paralel, (s) dikey.

Diğer faktörler; küçük belirsizlikler içermesi, net bir korelasyon sağlamaması veya sonucun daha da iyileşmesine katkı sağlamadığından dikkate alınmamaktadır. Bu nedenle, belirleyici parametreler daha çok geometrik büyüklükler ve teknolojik faktörlerdir (Förster ve Gudehus, 1998).



Şekil 5. Etki sahasının belirlenmesinde şev koşulları, ($X_{r,u}$) önlemsiz, ($X_{r,b}$) önlemli.

3 RAZİ REHABİLİTASYON YÖNTEMLERİ

Madencilik faaliyetleri sonucu bozulan arazilerin yeniden düzenlenmesi ve ıslahında temel amaç; bu arazilerin eski ekolojik ve ekonomik değerine kavuşturulması veya daha da geliştirmesidir. Bu ise;

- Madencilik sonrası alan kullanım planlaması,
- Alan kullanım planlaması doğrultusunda yeniden düzenleme (kazı, döküm, su rejimi kontrolü, üst örtünün sellektif değerlendirilmesi vb.),
- İyileştirme (biyolojik reklamasyon),
- İzleme, bakım ve kontrol faaliyetleri ile sağlanabilir (Akpınar vd., 1993).

Öngörülen arazi kullanım şekli ve işletme koşulları, uygulanacak rehabilitasyon programının seçimini etkiler. Açık işletme sırasında ve sonrasında oluşan üretim çukurları/boşlukları, döküm harmanları ve pasa şev yüzeylerinin rehabilitasyonu için, bir yöntem seçilmesinde mevcut sahasın;

- daha önceden tamamen su ile dolu olması,
- kısmen su ile dolu olması,
- susuz olması arasında farklılıklar vardır.

Ancak; dekapaj döküm şevlerinin kayma koşullarını belirleyen kriterlerden hareketle, bir kayma riskinin önlenmesi için muhtemel tedbirler öngörülebilir. Bunlar;

- Dekapaj döküm yığın özelliklerini değiştirmek (konsolidasyon ile dayanım özelliklerini arttırmak ve yapısal değişiklikler),
- Hızlı bir işletme yöntemi ile oluşacak boşluk suyu basınçlarını azaltmak (drenajların yapılması),
- Olumsuz etki kuvvetlerinden sakınmak ve koruyucu olmak,
- Teknolojik planlamada önlemler almak (daha az üretim çukuru ve şev yüzeyine izin vermek),
- Kuvvetli geçirgen, iri daneli veya özel durumlarda bağlayıcı pasa malzemesini nihai şev önüne selektif olarak dökmek,
- Açık işletmenin tamamlanmasından hemen sonra, yeraltı su seviyesi yükselmeden şevlerin kaymaya karşı güvenli olarak boyutlandırılması şeklinde gerçekleştirilebilir.

Söz konusu en önemli arazi rehabilitasyon yöntemleri ve jeoteknik sınır koşullarına bağlı olarak kullanım özellikleri, aşağıda detaylı olarak verilmektedir (Fritz ve Benthaus 1997, Novy vd. 1999).

3.1 Patlatmayla Dinamik Pasa Stabilitesi

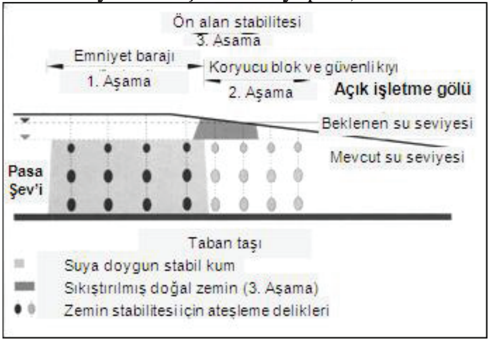
Bu rehabilitasyon çalışmaları genellikle üç etaptan oluşmaktadır. Bunlar;

I. Şev kıyısı boyunca bir emniyet barajının hazırlanması (Şek. 6),
II. Emniyet barajı denetiminde patlatmayla ön alan stabilitesi (Şek. 7),
III. Karıştırıcı veya vibratör silindir ile nihai şev'in şekillendirilmesidir.

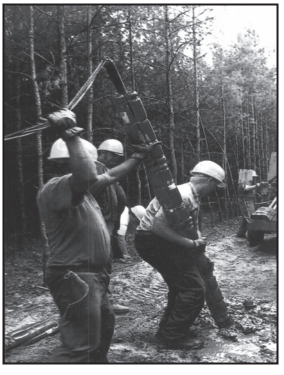
Yöntemin kullanımı için sınır koşulları;

- Dekapaj yığının suya doygunluğu,
- Kayma bölgesinde hiçbir koruyucu bloğun olmaması,
- Emniyet barajının saha statiğine göre boyutlandırılması,
- Kayma etki uzunluğunun saptanması (bkz. Bölüm 2),
- Mevcut pasa şev geometrisi (dik şev → dinamik stabilitenin başlamasından önce meyil azaltma ateşlemeleri uygulanır,

yatay şev → kaymadan sakınmak için koruyucu ateşlemeler yapılır).



Şekil 6. Patlatmayla dinamik pasa şev stabilitesi genel görünüşü (LMBV, 1996).



Şekil 7. Dinamik pasa şev stabilitesinde patlayıcının yerleştirilmesi (LMBV, 1996)

Patlatma yönteminin tercih nedenleri;

- Yüksek etki derinliği ve genişliği,
- Porozitenin azaltılması ve dayanım parametrelerinin yükseltilmesi ile mevcut yapının iyileştirilmesi,
- Patlamayla homojen sıkılık,
- Yaklaşık düz arazi yüzeyi içermesi,
- Göreceli düşük maliyet,
- Çabuk uygulanabilirliktir (Förster ve Warmbold 1992, Dähnert ve Vogt 1997).

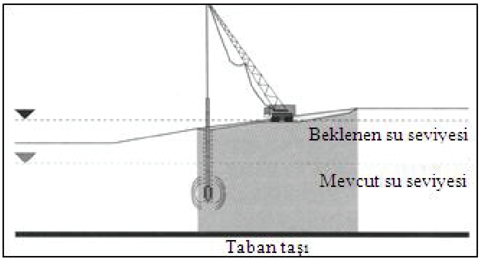
3.2 Titreşimli/Vibrasyonlu Dinamik Pasa Stabilitesi (RDV)

Yöntem; yeraltı su seviyesi yükselmeden önce, kayma riski bulunmayan pasa yığınlarının rehabilitasyonu için kullanılır. Belirli bir rizikoya bağlı koruyucu patlatmaların olduğu pasa döküm sahasında da bu yöntem kullanılabilir.

İlk önce karıştırıcı kol kendi ağırlığıyla, vibrasyon ve karıştırıcı ucundaki tazyikli su veya basınçlı hava-su karışımı yardımıyla, belirlenen derinliğe kadar indirilerek sıkıştırma yapılır. Esas sıkıştırma işi, karıştırıcının adım adım yukarı çekilmesiyle gerçekleştirilir. Dane yapısının yer değiştirmesiyle boşluk hacmi azalır ve yığın yüzeyinde sonradan doldurulacak bir huni oluşur (Novy vd., 1999).

Bölgesel heterojenliğe sahip pasa yığınlarında yöntemin iyi bir uyum göstermesi ve sıkıştırma işleminin sürekli kontrolü sağlandığından, yöntem teknolojik avantajlar sağlar. İlk defa 1990 yılında Niederlausitzer (Almanya) linyit işletmesi pasa yığınlarında uygulanan RDV, olumlu uygulama sonuçları göstermiştir.

Suya doygun şevlerin patlatma yöntemiyle, doğal şev kısmının vibrasyonla sıkıştırılması arasında bir kombinasyon kurularak, her iki yöntem etaplar halinde de kullanılabilir. Basınçlı karıştırıcı kol ile sıkıştırmanın (RDV) prensibi şekil 8'de gösterilmektedir.



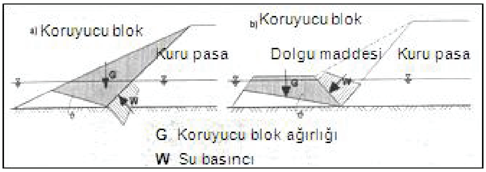
Şekil 8. Titreşimli kolla pasa şev stabilitesi (LMBV, 1996)

3.3 Şev Önünde Bir Destek Bloğu Oluşturmak

Destek bloğu; yığın olarak katkılı malzemeden, taşınmış veya yerinde oluşturulmuş ve sıkıştırılmış malzemeden inşa edilebilir (Şek. 9). Ayrıca;

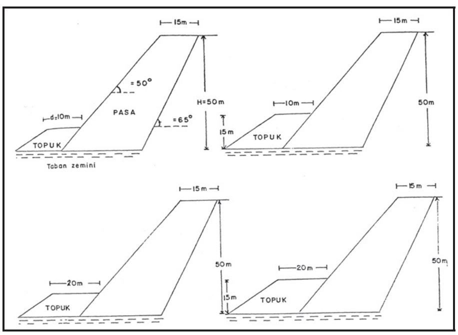
- bloğunun emniyetli boyutlandırılması ve sağlam temel üzerinde bulunması,

- yerleştirilen malzemenin akıcı olmaması ve baraj etkisinden sakınmak için yüksek geçirgenlik göstermesi,
- karıştırma işleminde suda sedimantasyon olmaması (karma yığın ve karıştırma topuğu sudan arındırılmış olmalı),
- zemin mekaniği bakımından kombine davranmayan malzemelerin kullanılması,
- karma yığının minimum yüksekliği, beklenen maksimum su seviyesine bağlı olarak belirlenmesi önemlidir.



Şekil 9. Koruyucu bloğun oluşturulması a) Karma malzeme b) Temel inşası (Warmbold ve Vogt, 1994).

Destek bloğunun yerleştirilmesi, genellikle üretim bölgesinde su seviyesi yükselmeden önce yapılır. Uygun sınır koşulları altında; inşaat molozları, kaya veya harfiyat malzemelerinden oluşan bir destek bloğu, daha önce suyla dolmuş üretim çukurlarında da yapılabilir. TKİ-GELİ Muğla bölgesinde de benzer yöntem uygulanmıştır (Şek. 10).



Şekil 10. TKİ-GELİ Muğla bölgesinde uygulanan pasa şev stabilitesi (Delibalta, 2003).

3.4 Ağırlıkla Dinamik Sıkıştırma (DYNIV)

Yöntem; yukarıdan düşürülen bir ağırlık vasıtasıyla yüzeye yakın tabakaların sağlamlaştırılması esasına dayanır. 1930'lu yıllarda otoban inşası amacıyla ve 1970'li yıllardan itibaren pek çok ülkede zemin iyileştirmede kullanılmıştır. Son yıllarda ise, kömür açık işletmelerinde kayma riski taşıyan yığınların ıslahında kullanılmaktadır. Önemli teknolojik büyüklükler; düşürülen ağırlığın yüzey alanı, kütlesi, düşme yüksekliği ve vuruş sayısıdır (Şek. 11).

Şekil 11. Ağırlıkla dinamik sıkıştırma ve oluşan konsolidasyon (LMBV, 1996).

Çarpma plakasından kısa bir süre sonra yüzeyde %100 kayma etkisine ulaşılır ve sonra hızla yavaşlar. Tekrarlanan düşmelerle zeminin sıkıştırılması ve yer değiştirmesi sağlanır. Çökme hunileri bağlayıcı olmayan zemin ile doldurulur. Daha sonraki zaman diliminde (birkaç hafta) sıkıştırma süreci birkaç kez yenilenir. Yöntemin avantajları;

- DYNIV kolay gerçekleştirilen ve çevre uyumludur (çarpma etki dağılımı büyük oranda düşey, çok az yatay yönde olur).
- Gerekli ek çalışmalar oluşan hunilerin doldurulmasından ibarettir. Bunun için bağlayıcı olmayan malzemeler kullanılır.
- Araç kullanımına göre günde 300-800m^2 alan sıkıştırılabilir.

Yöntemin dezavantaj ve kısıtları ise;

- DYNIV sadece belirli bir derinliğe kadar etkir (max.12~15m) ve bu yüzden yüzeye yakın tabakaların sıkıştırılmasında kullanılır.
- Yöntemin kullanılması için kısıtlar;
  - İnce dane oranı (<0,002mm) ağırlıkça %10-15,

- Akma sınırının %100 ~110 su içeriği,
- 10^{-7}m/s geçirgenlik katsayısı olarak belirlenmektedir.

3.5 Maden Enjeksiyonu ile Sıkıştırma

Dinamik sıkıştırma yöntemiyle kayma belirtilerinin ortaya çıktığı, daha doğrusu kontrol edilemez kaymaların meydana gelebileceği alanlarda; zeminlerin stabilitesi için enjeksiyon yöntemleri uygulanabilir. Yöntem aşağıdaki durumlarda tercih edilir.

- Bina, yol ve köprü gibi yapıların stabilitesi,
- Altyapı sistemlerinin korunması,
- Şev ve gerisinde tamamlanmış yeşil sahaların korunması.

Enjeksiyon basıncının tespitinde; geçirgenlik, gözenek dağılımı ve derinliğe bağlı gerilme gradyentleri gibi zeminin petro-fiziksel parametreleri önemli rol oynar. Genellikle karma zeminler; farklı geçirgenlik, gözenek dağılımlı ve geniş bir dane spektrumu gösterirler, öyle ki iyi enjekte edilebilir ve pratikte yatay ve düşey yönde sıkıştırılmış alanlar yaratabilirler. Enjeksiyon amacına ve pasa yığının enjekte edilebilirliğine bağlı olarak, aşağıdaki enjeksiyon sıvıları seçilebilir.

- Yaklaşık %30 bağlayıcı içerikli maden reçinesi dispersiyonu, düşük viskoziteli (5-8cp) sulu bir sıvıdır. Pasa yığınları iyi enjekte edilebilir ($k_f \geq 10^{-6}$m/s). Enjeksiyon basıncı hidrostatik basınçtan daha büyüktür. Yüksek permabilite daha doğrusu gevşek yığınlı alanlar akış yönü olarak tercih edilir ve kohezyon artışıyla stabilite sağlanır.
- Katkılı maden reçinesi dispersiyonu (%2-8 çimento veya karma bentonit), yüksek permabilite ($\geq 10^{-4}$m/s) ve gevşek yığınlı alanlar içerirler. Enjeksiyon sıvısıyla boşluklar doldurulduktan sonra, konsolidasyon yük enjeksiyon basıncıyla yapılır. Böylesi sıkıştırılmış bölgenin kesin geometrik tasarımı çok güçtür.
- Yüksek basınç enjeksiyonlarında; özel kovanlar veya kolonlar olarak silindir ve lamel gibi geometrik şekiller hazırlanır. Yöntem vasıtasıyla farklı petro-fiziksel karakteristikli pasa yığınları yüksek basınç dolayısıyla (15-300 bar) yapısal olarak yeniden dizilir, enjeksiyon sıvısında yataklanır ve sağlamlaşır.

Enjeksiyon reçinesi olarak aşağıdaki karışım uygulanabilir. %30 Securan (%30 maden reçinesi dispersiyonu), %30 su, %33,5 portlant çimento ve %6,5 tixoton (kil minerali). Bu tür enjeksiyon sıvısı ile, dayanım >10kp/cm^2'ye kadar erişebilir (Förster ve Gudehus, 1998).

Enjeksiyon yönteminin avantajları; kayma belirtileri olmaksızın zemin iyileştirmesi ve teknolojisinde görülür. Öyle ki enjeksiyon işlemleri ayarlanabilir, noktasal uyumlu deliklerde güvenilir bölgelerin yüksek derinliği (~1000m) mümkündür. Dezavantajları ise; bu tür stabilitenin toplam maliyetinde görülür (10-25 €/m^3). Ayrıca; sondaj için teknik teçhizatlar yanında, enjeksiyon malzemeleri için sarf masraflar ortaya çıkabilir.

3.6 Müller Resonans Sıkıştırması (MRC)

Çalışma prensibi; titreşimli/vibrasyonlu sıkıştırmaya (RDV) benzemektedir. Sadece, karıştırıcı kol yerine kendi ağırlığı nedeni ile zemine baskı uygulayan sıkıştırıcı bir kol kullanılır. Sıkıştırma süreci; dikey yönde kolun vibrasyonu ile başlar ve bu esnada kol ile zemin arasında kayma hareketi oluşur. Çalışma sırasında sıkıştırılan gevşek kayacın kayma frekansı bir veya birden fazla yüzeydeki alıcı sensör ile ölçülür. Değişken frekanslı vibratör, ölçülen kaymalara öyle uyumludur ki; sıkıştırılan zemin tabakasının kendi frekansındaki (resonans) kaymaya kadar erişir. Böylece sıkıştırma süreci; dane yapısının yer değiştirmesi ve danenin yeni düzen kazanmasıyla daha sıkı yığın haline gelmesinden ibarettir. Sıkıştırma normal olarak ortak davranış gösteren bir malzemeyi en fazla etkilediğinden, prensip olarak yöntem avantajlıdır. Fakat aynı zamanda güçlük, kol boyunca enerji aktarımı ve optimum sıkıştırma için spesifik enerjinin yeterli olmamasında görülebilir (yüksek sürtünme kayıpları).

MRC-Yönteminin kullanım alanları; yapısı kolay değişebilen zeminler, yani yüksek benzerlik boyutundaki düşük ince dane oranlı, doğal fakat daha çok suya doygun kum zeminlerdir. Avantajları; yabancı malzemeye gereksinim olmaması, ek çalışmalar sadece oluşan oturma hunilerinin doldurulmasından ibarettir. Dezavantajları; yöntem yüzeye yakın tabakaların sıkıştırılmasında kullanılır. 20m < derinliklere kadar etkilidir. Bu yeni yöntem, Almanya'da henüz pek az kullanılmaktadır (Förster ve Gudehus, 1998).

3.7 Boşluk Suyu Basınç Bariyerleri

Tamamlayıcı veya alternatif yöntem olarak boşluk suyu basınç bariyerleri, kayma riski bulunan pasa alanlarında stabilite veya destek için kullanılabilir. Şayet bir kayma sonucu gelişen boşluk suyu basınç artışı sınırlandırılır ise, ilgili pasa yığınlarında tamamen kesme dayanımı kaybına değil, bilakis büyüklüğü boşluk suyu basıncı üzerine etki edebilen, en düşük kesme dayanımının sağlanmasına kadar erişilebilir. Boşluk suyu basınç bariyerlerinin henüz iki türü modellenmiş ve pilot çapta denenmiştir. Bunlar;

- boşluk suyu basınç etkisinin azaltılması için hava yastığı,
- hızla boşluk suyu basıncının azaltılması için drenaj tesisidir.

Hava yastığının avantajları; sondaj ve hava basıncı için basit bir teknik teçhizat, düşük malzeme gereksinimi ve bununla göreceli düşük maliyet içermesidir. Hava katkısının desteklenmesinde yüzey aktif maddelerin kullanımı; hava doygunluk derecesini ve bununla boşluk suyu basınç bariyerlerinin sıkıştırıcılığını arttırır. Yüzey aktif madde enjeksiyonu ile boşluk suyu basınç bariyerinde hava uzun zaman lokal olarak sabitlenebilir.

Drenajların avantajları; boşluk suyu basıncının azaltılmasında sürekli bir çözüm oluşturmaları ve artan işletme maliyetlerine neden olmamalarıdır. Düşey drenajlar, pasanın dökümünden sonra düzenlenebilir. Buna karşın yatay drenajlar ise, ilerleyen açık işletmede teknolojik planlama ile eş zamanlı olarak hazırlanır.

Boşluk suyu basınç bariyerlerinin tesisinde genel avantajları; sondajlarda yığının korunan gerilmesi ve sıkıştırmada kaçınılmaz olan kuvvetli etkilerin bulunmayışıdır. Bu tespitlere göre yöntemin tercih edilen kullanım alanları;

- Patlatma veya vibrasyonlu yöntemlerde, sıkıştırılmış bölge (emniyet barajı) ve pasa şev'i arasındaki mesafe kısaltılabilir,
- Yüksek boşluk suyu basınç oluşum bölgeleri yakınında (örneğin patlatma yönteminde) hızlı işletme veya drenajlarla ani dağılım ve gevşemeyi sağlamada,
- Yığın içerisindeki boşluk suyu basınç kontrolünde, öyle ki kaymayı kolaylaştıran etki kuvvetlerden boşluk suyu basınçları, direkt çevrede kısa sürede kayıp olur,
- Rehabilitasyon bölgesinde sürdürülecek ağaçlandırma faaliyetleri ile sınırlandırılmış alanlarda, boşluk suyu basınçlarının hızlı tahliyesi veya ani dağılımı sağlanır,
- Daha önce patlatma ile sıkıştırılmış yığınların üst bölgesinde yeniden yeraltı su seviyesinin yükselmesinden sonra uygulanmasıdır.

Boşluk suyu basınç bariyerleri, şimdiye kadar tanıtılan rehabilitasyon yöntemlerinin tamamlayıcısı ve sadece özel durumlarda kullanılan bir yöntem olarak görülebilir.

3.8 Arazi Rehabilitasyon Yöntemlerinin Değerlendirilmesi

Arazi rehabilitasyon yöntemlerinin kullanım alanları;

- ➢ stabilitenin amacına bağlı olarak;
  - şev'in kendi kendine güvenilir konuma gelmesi,
  - imara uygun kullanım için kayma riskinden uzak bölgenin sağlanması,
- ➢ mevcut yeraltı su seviyesine bağlı olarak;
  - yeraltı su seviyesinin yükselmesinden önce ve sonra olmak üzere farklılıklar göstermektedir (Çiz.1).

Çizelge 1. Arazi rehabilitasyon yöntemlerinin kullanılabilirliği.

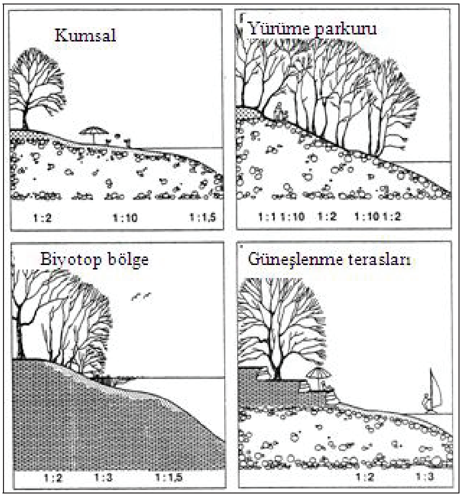
Yöntemler	**Uygulanabilirlik** Şev stabilitesinde yeraltı su seviyesinin yükselmesinden Önce	Şev stabilitesinde yeraltı su seviyesinin yükselmesinden Sonra	İmar alanı hazırlanması yeraltı su seviyesinin yükselmesinden Önce	İmar alanı hazırlanması yeraltı su seviyesinin yükselmesinden Sonra
Sellektif döküm	+ o	-	+	-
Statik sıkıştırma	+ ! o	-	+ ! o	-
Vibrasyon silindir	+ ! o	-	+ ! o	+ !
Karıştırıcı plaka	-	-	-	-
Derin titreşimli karıştırıcı (RDV)	+ !	X	+ !	+ !
Vibro flotasyon	-	X	+ !	+ !
Dinamik sıkıştırma (DYNIV)	-	-	-	+
Şok etki sıkıştırması	+ o	-	+	+
Patlatma ile dinamik sıkıştırma	+ !	-	+ !	+
Zemin (Enjeksiyon) kaplama	+ !	X	+ !	+ !
Karma destek bloğu	+	+	-	-
Dökme destek bloğu	+	X	-	-
Filtre (Drenaj) düzenleme	+	X	+ !	!

Sembollerin anlamı:
- : Yöntem uygun değildir, +: Yöntem sınırlama olmadan kullanılabilir,
X: Yöntem sadece kayma eğiliminde malzeme bulunmuyorsa kullanılabilir,
!: Yöntem yüksek maliyeti nedeniyle sadece özel durumlarda kullanılabilir,
o: Pasa döküm yığın özelliklerine göre geçerlidir.

Aynı zamanda; bozulan arazinin yeniden düzenlenmesinde güzel bir peyzaj görünüme ulaşmak kadar, buradan ekolojik ve ekonomik olarak yararlanmakta hedeflenmelidir. Bu maksatla, arazi geri dönüşüm çalışmaları;

Her defasında kullanılacak arazi iyileştirme yöntemi; ilgili sahanın jeolojik, hidrolojik ve zemin mekaniği koşullarına, üretim boşluğunun durumuna ve de öngörülen arazi kullanım planına bağlı olarak seçilmelidir. Özellikle ileriki arazi kullanım planı (Biyotop, dinlenme, ulaşım yolları, bayındırlık vs. gibi) yapılacak rehabilitasyonun kapsam ve yoğunluğunu belirler. Genel olarak; optimum arazi rehabilitasyon etkisine yalnız bir yöntem ile değil, bilakis birden fazla yöntemin uygun bir kombinasyonu ile erişilebilmektedir (Dähnert ve Vogt, 1997).

- Ziraat (tarım, bahçe, mera vs.),
- Orman (ticari ve ticari olmayan),
- Rekreasyon (eğlence ve dinlenme yerleri, parklar, halka açık alanlar vs. Şek.12),
- Su kullanımı (balıkçılık, toplumsal ihtiyaçlar için),
- Bayındırlık ve inşaat (hafif endüstriyel binalar, konut ve hizmet binaları, yol vs.),
- Yaban hayatı (biyotop, doğal koruma alanları vb.) gibi amaçlar için kullanılabilir (Dingethal vd. 1985, Köse vd. 1993).



Şekil 12. Arazi dönüşüm planı için farklı şev tasarımları (Dingethal vd. 1985).

Ayrıca; açık işletme sonrası bozulan arazilerin yeniden düzenlenmeside, teknolojik süreçte kullanılan ekskavatörler, nakliyat araçları ve dökücüler, uygulanacak arazi rehabilitasyon yöntemini geniş ölçüde etkileyebilmektedir (Derbenstedt 1996, Brauer 1997). Bu nedenle; açık işletme üretim yöntemi ve ekipman planlamasında, madencilik sonrası arazilerin yeni kullanım şekli ve dönüşüm planları da göz önünde tutulmalıdır.

Madencilik faaliyetleri sonrası bozulan arazilerin rehabilitasyonu için gerekli ekonomik göstergeler ise aşağıda kısaca verilmektedir. Örtü kömür oranı = 8:1 ve her 1000 ton üretim için 0,2 hektarlık arazinin bozulduğu bir açık işletmede, iyileştirme maliyetinin 0,32 ABD $/ton veya toplam maliyetin %7,6'sı olduğunu göstermektedir (Ünal vd., 1992). Ayrıca, Doğu Almanya linyit ocaklarının ıslahı projesi ile 7,5 milyar € harcanarak 100.000 hektardan fazla alan tekrar kullanılabilir hale getirilip çevreye entegre edilmiştir. Devam eden çalışmalar için tahmini ihtiyaç ise 1,5 milyar €civarında bildirilmektedir (Kuyumcu, 2005). Ülkemizde bu maliyetler ile ilgili rakam vermek güç olmakla birlikte, TKİ-GELI Muğla bölgesi için ortalama 58,5 TL/hektar gibi çok düşüktür (Bozoğlan 1997, Delibalta 2003). Çünkü arazi rehabilitasyonu olarak ülkemizde, genellikle ilgili sahalarda sadece teraslama, çapalama ve ağaçlandırma faaliyetleri yapılmaktadır.

4 SONUÇLAR

Kömür açık işletmelerinde, gerek üretim gerekse dekapaj faaliyetleri sırasında ve sonrasında pek çok üretim çukurları/ boşlukları ve şev yüzeyleri oluşmaktadır. Geometrileri; örtü kalınlığına, işletme büyüklüğüne ve seçilen açık işletme üretim teknolojisine bağlı olarak bozulan bu arazilerin yeniden düzenlenerek kullanıma sunulması, çevresel etki değerlendirme bakımından büyük önem taşımaktadır. Farklı kullanım alternatiflerinin tasarlanabileceği bu arazilerin, başta jeoteknik bakımdan stabil ortam koşularına sahip olması gerekmektedir.

Dekapaj malzemesinin gevşek yapısal ve oluşum özellikleri nedeniyle, önemli bir kısmı kayma veya akma eğiliminde olmaktadır. Çok kısa zaman dilimi (birkaç saniye veya dakika) içerisinde ani hızlara ulaşan ve büyük bir enerji boşalmasıyla sonuçlanan bu tür şev kaymaları, önemli tehlike potansiyelleri içermektedir. Bu nedenle, pasa yığınlarında oluşabilecek bir kayma riskinin sayısal esaslara göre boyutlandırılması gerekmektedir.

Madencilik faaliyetleri sonrası bozulan arazilerin yeniden düzenlenmesi ve kayma riski taşıyan pasa şev yüzeylerinin stabilitesi için, RDV, DYNIV ve MRC gibi çeşitli arazi rehabilitasyon yöntemleri geliştirilmiştir. Genel olarak; optimum arazi rehabilitasyon etkisine yalnız bir yöntem ile değil, bilakis birden fazla yöntemin uygun bir kombinasyonu ile erişilebilmektedir. Ayrıca; üretim çukurunun geometrik boyutu, öngörülen arazi kullanım planlaması ve işletme bölgesindeki jeolojik, hidrolojik, zemin mekaniği koşulları ve de teknolojik süreç, uygulanacak arazi rehabilitasyon yöntemini etkilemektedir.

Sonuç olarak madencilik faaliyetleri sonrası bozulan arazilerin yeniden düzenlenmesi ve iyileştirilmesin de temel amaç; bu arazilerin güzel bir peyzaj görünümü kazanması yanında, eski ekolojik ve ekonomik değerine kavuşturulması veya daha da geliştirilmesi olmalıdır. Çok yönlü disiplinler arası çalışmayı gerektiren bu faaliyetler, ancak mevcut hukuksal, ekonomik ve zamansal olanaklar ölçüsünde gerçekleştirilebilir.

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CBS ve UA Kullanılarak Muğla Mermer Ocaklarının Çevresel Etkilerinin Belirlenmesi

Determining the Environmental Impacts of the Marble Quarries in Mugla by Using GIS and RS

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ÖZET Türkiye'de mermer madenciliği ve mermerin yapı ve kaplama malzemesi olarak kullanılması antik çağlardan bu yana süregelen bir faaliyettir. Özellikle Türkiye'nin batısında bulunan Muğla bölgesi ülkenin mermer üretiminde önemli bir role sahiptir. Bunun nedeni, Muğla'da üretilen mermerlerin blok verimliliği yüksek olmasıdır. Mermerin bu ekonomik öneminin yanında, madencilik faaliyetlerinden dolayı oluşan arazi ve bitki örtüsündeki değişimlerin araştırılması günümüzde çevreye verilen önemin artması ile beraber araştırılması gereken bir konu olarak ortaya çıkmaktadır.

Bu bildiride, terk edilmiş veya üretimi halen devam etmekte olan mermer ocaklarının alansal olarak genişlemeleri ve bitki örtüsündeki yıllara bağlı olarak değişimler incelenmiştir. Bu araştırma için, Coğrafi Bilgi Sistemleri (CBS) ve Uzaktan Algılama (UA) teknolojileri ocakların çevreye olan etkilerini belirlemek amacıyla kullanılmıştır. Ocakların konumlarının belirlenmesi için MİGEM'den (Maden İşleri Genel Müdürlüğü) alınan maden ruhsat koordinatları ve araştırma için 2003–2007 yılları arası çekilen uydu görüntüleri bu çalışmanın veri kaynağını oluşturmaktadır.

ABSTRACT Marble quarrying and the use of marble as a construction or covering material is an ongoing activity in Turkey since the ancient times. Besides, Mugla region in western Turkey plays an important role in the nation's marble production. Also, the marble produced in Mugla carries a high rate of block productivity. In addition to the economic significance of marble, research about the changes to land and vegetation incurred by quarrying practices becomes a subject that should be studied alongside the contemporary increase of environmental awareness.

This paper examines the areal expansion of marble quarries either abandoned or currently under production as well as the consequent changes in vegetation over the years. For this research, Geographic Information Systems (GIS) and Remote Sensing (RS) technologies were used to determine the environmental impacts of the quarries. Mine permit coordinates obtained from MIGEM (General Directorate of Mining Affairs) were used to locate the quarries, and satellite images taken between 2003 and 2007 provides the data source of this project.

1 GİRİŞ

Antik çağlardan beri mermer üretiminin önemli bir merkezi olan Muğla ili, günümüzde de Türkiye mermerciliğinde başı çeken şehirler arasındadır.

Bölgede mermercilik faaliyetleri 1980'li yıllardan sonra yoğun olarak başlamıştır. Milas-Kalınağıl yöresinde ise üretim faaliyetleri 1970'li yıllara kadar uzanmaktadır (Yüzer ve Erdoğan, 1996). Mermer ocakları çoğunlukla Muğla'nın Yatağan, Milas ve Kavaklıdere ilçelerinin çevresinde konumlanmıştır (Yavuz vd., 2005). Maden İşleri Genel Müdürlüğü'nden (MİGEM) bu çalışma kapsamında elde edilen işletme ruhsat bilgilerine dayanılarak, Muğla'da 2008 yılı rakamları ile 119 ruhsatlı mermer sahası bulunmaktadır.

Muğla'nın Türkiye mermer potansiyeli açısından önemiyle birlikte, son yıllarda çevreye ve yeşil alanlara verilen değerin artmasından ve madencilik faaliyetlerinin çevre üzerindeki bazı olumsuz etkilerden dolayı çalışma alanı olarak Muğla ili seçilmiştir. Diğer taraftan, ildeki madencilik faaliyetlerinin yapıldığı alanların çoğunun orman arazisi olmasından ve bölgeye yönelik madenlerden ötürü oluşabilecek çevresel etkilerin araştırılması zorunlu hale gelmiştir.

Bu çalışmada, 2003-2007 arası Muğla'daki mermer ocaklarının çevresel etkilerinin, madencilik faaliyetlerinden dolayı yeşil alanlarda oluşabilecek değişimin araştırılması için Coğrafi Bilgi Sistemleri (CBS) ve Uzaktan Algılama (UA) teknolojileri kullanılmıştır. Veri kaynağı olarak; ocakların tam olarak yerinin tespiti için MİGEM'den alınan mermer ocağı ruhsat saha koordinatları ve ili kapsayan uydu görüntülerinden yararlanılmıştır. Veriyi girme ve kullanma, görüntü işleme ve analizler için MapInfo 9.0 ve ERDAS IMAGINE 9.2 yazılımlarından yararlanılmıştır.

2 COĞRAFİ BİLGİ SİSTEMLERİ VE UZAKTAN ALGILAMA

Yirmi yılı aşkın bir süredir planlama, afet yönetimi, risk analizleri, pazarlama, vb. konularda önemli bir araç olarak kullanılan CBS ve UA, çok disiplinli yapılarıyla uygulama alanlarını attırmış ve günümüzde birçok çalışmada bu teknolojilerin kullanımı zorunlu hale gelmiştir. Bu iki teknolojinin entegrasyonu, coğrafi içeriğe sahip varlıklar hakkında yeni bilgiler elde etmek ve farklı coğrafi analizler yaparak problemlerin çözümüne katkıda bulunma çerçevesinde önemlidir.

CBS, bilgisayar donanım ve yazılımları ile beraber belirli bir amaca ulaşmak, problem veya problemleri çözmek için konumsal ve konumsal olmayan verileri toplama, bilgisayarda depolama, güncelleme, paylaşma, kontrol ve analiz etme, görüntüle ve bütün bu faaliyetlerden bir veya birçok bilgi elde üretilmesine olanak sağlayan bir bilgi sistemidir (Koruyan vd., 2005). UA ise CBS'nin veri kaynağını oluşturmaktadır. Bunun yanında, UA'da görüntü işleme teknikleri ile CBS'den farklı olarak uydu veya hava fotoğrafı görüntüleri ile birçok analiz yapılabilmektedir.

Uçaklarla taşınan kameralar veya dünya yörüngesinde konumlandırılmış uydulardaki sensörler yardımıyla UA görüntüleri (verileri) elde edilmektedir. Alınan görüntülerin işlenmesi ve verilerden yeni bilgiler elde edilmesi çeşitli bilgisayar donanım ve yazılımları ile mümkün olmaktadır.

UA çalışmalarına ilk adım, Dünya yörüngesine 1972'de ABD tarafından Earth Resources Technology Satellite-1 (ERTS-1, 1975'de ismi Landsat-1 olarak değiştirilmiştir) uydusunun uzaya gönderilmesiyle atılmıştır (Harper, 1983). UA, birçok farklı çevresel disiplinlerde (coğrafya, jeoloji, zooloji, tarım, ormancılık, botanik, meteoroloji, oşinografi) kullanılmaktadır (de Jong vd, 2004). Günümüzde ise, ilerleyen teknoloji ve artan ihtiyaçlarla değişik platform ve sensör çeşitleri ile yeryüzündeki değişimler incelenmektedir.

Aşağıda, kullanılmakta olan bazı platform/sensorlar verilmektedir (Kerle vd., 2004):

- Yüksek çözünürlüklü pankromatik sensorlar (Piksel boyutu 0,5 m ila 6 m). (OrbView/PAN, Ikonos/PAN, QuickBird/PAN, EROS/PAN, IRS PAN, Spot/PAN).

- Maltispektral sensorlar (4 m ila 30 m mekansal çözünürlükte). (Landsat/ETM+, Spot/HRG, IRS/Liss3, Ikonos/OSA, CBERS/CCD, Terra/Aster).
- Birçok meteoroloji uyduları ve diğer düşük çözünürlüklü sensorlar Piksel boyutu 0,1 km ila 5 km). (GOES/Imager, Meteosat/Seviri, Insat/VHRR, NOAA/AVHRR, Envisat/Meris, Terra/MODIS, Spot/VEGETATION, IRS/WiFS).
- Radar amaçlı görüntüleme (8 m ila 150 m mekansal çözünürlükte). (Envisat/ASAR, ERS/SAR, Radarsat/SAR).

Çalışma kapsamında veri kaynağı olarak ASTER (The Advanced Spaceborne Thermal Emission and Reflection Radiometer) Level 3A01 (3D Ortho Data Set) uydu görüntüleri kullanılmıştır. ASTER Aralık 1999'da Terra uydusuna monte edilerek fırlatılan ileri maltispektral bir görüntüleyicidir. ASTER; yüksek mekansal, spektral ve radyometrik çözünürlükle görünürden termal kızılötesine doğru 14 bant ile geniş bir spektral alanı içermektedir. Bu geniş spektral alan üç VNIR (Visible and Near Infrared Radiometer) bandı, altı SWIR (Short Wave Infrared Radiometer) ve beş TIR (Thermal Infrared Radiometer) bandına sahiptir. Ek olarak, bir teleskop da steroskopik özellikte olup, geriye bakışlı yakın kızılötesi spektral bandını barındırmaktadır (3B bandı) (Çiz.1). Bununla birlikte, ASTER Level 3A01 görüntüsü, Level-1A ve Level-4A01 (DEM-Digital Elevation Model veri seti) ürünlerinin çeşitli düzeltme ve işlemlerden sonra elde edilmektedir (ERSDAC, 2004). DEM verisi ise topografya veya arazinin dijital olarak sunumudur.

ASTER verisinin avantajı, görünür yakın-kızılötesinden kısa-dalga kızılötesine, bundan da termal kızılötesinde geniş bir spektral aralığın ve yüksek mekânsal çözünürlüğün özgün bir kombinasyonu olmasıdır (Gad & Kusky, 2007). ASTER verisi çoğunlukla bitki ve ekosistem araştırmaları, afetleri izleme, jeoloji, iklimbilim, hidroloji ve yüzeydeki değişimlerin incelenmesi gibi araştırmalarda kullanılmaktadır (Fujisada, 1998).

Çizelge 1. ASTER spektral bant geçişleri.

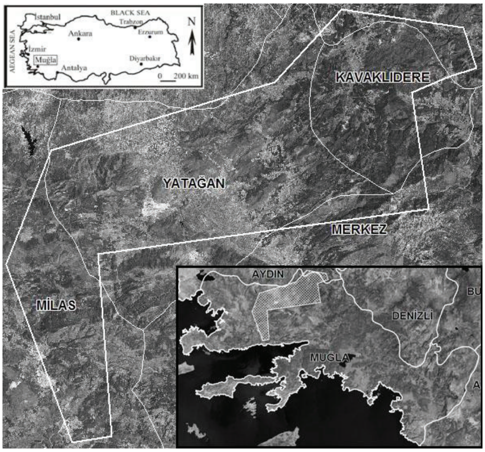
Bantlar	Bant No.	Spektral Aralık	Çözünürlük
VNIR	1	0,52 - 0,60	15 m
	2	0,63 - 0,69	
	3N	0,78 - 0,86	
	3B	0,78 - 0,86	
SWIR	4	1,600 - 1,700	30 m
	5	2,145 - 2,185	
	6	2,185 - 2,225	
	7	2,235 - 2,285	
	8	2,295 - 2,365	
	9	2,360 - 2,430	
TIR	10	8,125 - 8,475	90 m
	11	8,475 - 8,825	
	12	8,925 - 9,275	
	13	10,25 - 10,95	
	14	10,95 - 11,65	

3 ÇALIŞMA ALANI

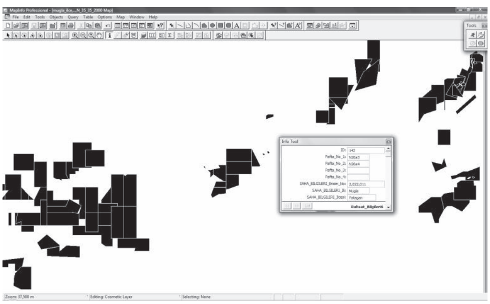
Çalışma alanı Muğla'nın Milas, Yatağan, Kavaklıdere ilçeleri ve Merkez ilçeyi kapsayacak şekilde sınırlandırılmıştır. Mermer ocaklarının çoğunun Milas-Yatağan-Kavaklıdere güzergâhında olması, uydunun istenilen zaman aralığında il sınırlarını kapsayacak şekilde Muğla'dan görüntü almamış olması, hava koşulları, bulutluluk vs. yüzünden il sınırları içersinde kalan işletme ruhsatı bulunan sahaların tümü çalışma kapsamına alınamamıştır. Bu yüzden 2003, 2005 ve 2007 yıllarında çekilmiş görüntüler veri kaynağını oluşturarak Şekil 1'de gösterilen çalışma sahası içinde kalan alan üzerinde yoğunlaşılmıştır.

4 METODOLOJİ

Çalışma başlangıcında MİGEM'den Muğla'da yer alan mermer ocaklarının ruhsat koordinatları tedarik edilmiştir (Haziran 2008). Koordinatlar ve ruhsat sahası ile ilgili veriler MapInfo yazılımı yardımıyla coğrafi veri tabanı haline getirilmiş ve Muğla mermer ruhsat sahası haritası oluşturulmuştur. Her nokta kapalı poligon haline getirilerek Şekil 2'de görülen ruhsat saha haritası elde edilmiştir.



Şekil 1. Çalışma alanı.

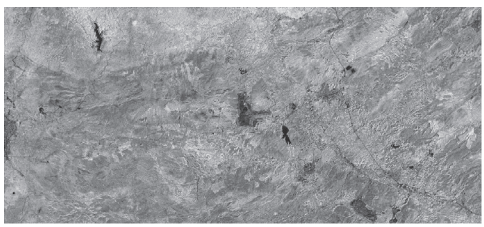


Şekil 2. Mermer ruhsat sahaları.

Uydu görüntülerine çalışma öncesinde bazı düzeltmeler yapmak çalışmanın sağlıklı yürütülmesi açısından gerekli bir süreçtir. Mather'a (1999) göre proses öncesi çalışmalar, geometrik ve radyometrik eksikliklerin düzeltilmesi ve hatalı verilerin giderilmesidir. Çalışmada kullanılacak uydu görüntülerine geometrik ve radyometrik düzeltmelerin yapılması çalışmanın sağlıklı ilerlemesi açısından önemlidir. Geometrik düzeltmeler Muğla ilini kapsayacak 1/25000'lik paftalar yardımıyla (köprü, yol kavşakları vb referans alınarak) yapılmıştır. Bunun yanında, 2003-2007 arası uydu görüntülerindeki histogram farklılıklarından dolayı histogram eşleştirme uygulanarak her bir görüntü belirli bir standarda getirilmiştir.

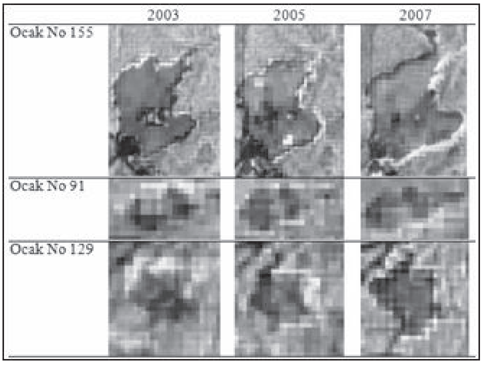
Ocak sınırlarının doğru saptanması ve ocakların iki boyutlu düzlemde ne kadar genişlediklerinin hesaplanması için bant oranları ile yapılan yöntemlerden yararlanılmıştır.

ASTER bant oranı görüntülerinin jeolojik haritalama ve özellikle jeolojik materyalin tanımlanması için yararlı olduğu kanıtlanmıştır (Abdeen vd., 2001). Litolojik olarak tanımlanmış yapıların haritalanmasına yönelik geleneksel yaklaşımlar, elektromagnetik spektrumun VNIR ve SWIR bölümlerindeki farklı kayaç tiplerindeki yansıma farklılıkları temel alarak kullanılmaktadır. Bantlar veya bant oranları RGB (Kırmızı, yeşil, mavi) renk kombinasyonu görüntüleri olarak gösterilmektedir (Abrams vd., 1983; Chavez vd., 1982; Chavez ve Bowell, 1984; Sheffield, 1985; Sultan vd., 1986; Sultan vd., 1987; Beauchemin ve Fung, 2001; Kusky ve Ramadan, 2002; Abdeen vd., 2002). Abdeen vd.'nin 2001'de yaptığı çalışmada; Abram vd.'nin (1983) (4/7-3/4-2/1) ve Sultan vd.'nin (1987) [4/7-4/1-(2/3)*(4/3)] oranları kullanılmış, bunların litolojik birimlerin ve özellikle serpantin, granit ve mermerin tanımlanmasında çok kullanışlı olduğu belirtilmiştir. Yapılan bu çalışmada da 2003, 2005 ve 2007 yıllarına ait ASTER görüntülerine, yukarıda geçen her iki bant oranları uygulanmış ve işletilen mermer ocaklarının net bir şekilde belirlenmesinde en iyi sonuç Sultan vd.'nin (1987) oranı ile sağlanmıştır (Şekil 3).



Şekil 3. Sultan vd. (1987) oranı ile oluşturulan görüntü.

Muğla uydu görüntüleri yardımı ile mermer ocaklarının iki boyutlu düzlemde ne kadar alanı işgal ettiği ve zamana bağlı olarak sınırlarını ne kadar genişlettiklerinin tespiti için ocak sınırları baz alınarak alan hesaplamaları yapılmış ve Şekil 4'de üç ocak için örnek verilmiştir.



Şekil 4. 2003, 2005 ve 2007 yıllarında seçilen üç mermer ocağının bant oranları kullanılarak elde edilen görüntüleri.

İkinci bir çalışma olarak da bölgede mermer madenciliğinden dolayı oluşan yeşil alanlardaki değişim incelenmiştir. Bitki örtüsü değişimi ve araştırmalarında sıklıkla bitki örtüsü indeksleri (Vegetation Index: VI) kullanılmaktadır.

VI'ler tipik olarak bitki biokütlesiyle ve zirai alanlardaki yaprak alan indeksiyle ilişkilidir ve bitkilerin çoğunlukla sağlıklı olup olmadıklarını ifade etmektedir (Pinter Jr. vd., 2003). Birçok araştırmacı yıllardan beri VI'ler ile ilgili araştırmalar yapmışlar, kırka yakın VI geliştirilmişlerdir (Qinjiu ve Xiangjun, 1998). Bu indekslerden önemlileri aşağıda sıralanmıştır:

Basit Bitki İndeksi (Jordan, 1969) (SVI) = NIR / R (1)

Normalleştirilmiş Bitki Örtüsü Fark İndeksi (Rouce vd., 1973) (NDVI) = (NIR – R) / (NIR + R) (2)

Dönüştürülmüş Bitki Örtüsü İndeksi (Deering vd., 1975) (TVI) = $(NDVI + 0.5)^{1/2}$ (3)

Geliştirilmiş Bitki Örtüsü İndeksi (Anderson, 1997) (EVI) = [(NIR - R) / (NIR + 6 x R – 7,5 x B + 1)] x 2,5 (4)

Dikey Bitki Örtüsü İndeksi (Curran, 1983) (PVI) = sin(a)NIR- cos(a)R (5)

Burada;
NIR: Yakın kızıl ötesi bandından kazanılan veri,
R: Spektrumun Kırmızı Bölümünden kazanılan veri,
B: Spektrumun Mavi Bölümünden kazanılan veri,
a: Güneşin Geliş Açısı'dır.

Bu çalışmada ise mermer ocakları çevresindeki bitki örtüsündeki değişimin araştırılması için birçok bilimsel araştırmada da sıklıkla kullanılan NDVI'den yararlanılmıştır. NDVI biofiziksel bir özellik olup, bitki örtüsünün fotosentez faaliyetiyle bağıntılıdır. Buna ek olarak, bitkinin canlı olup olmadığının göstergesi koşuldur. (Wang ve Tenhunen, 2004). Tucker'ın (1979) araştırmalarında NDVI değerlerinin -1,0'le +1,0 arasında değiştiğini fakat bitki değerlerinin 0,1 ila 0,7 arasında değerler aldığı ve yüksek indeks değerleri canlı bitki örtüsüyle orantılı olduğunu belirtmiştir.

5 SONUÇLAR VE TARTIŞMALAR

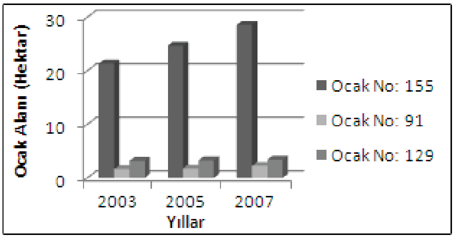
Muğla; mermer üretimi açısından Türkiye'nin önemli merkezlerinden biridir. Ayrıca, ilin büyük bir kısmı ormanlarla kaplıdır. Türkiye'de son yıllarda çevreye ve yeşil alanlara verilen önemin artması madencilik faaliyetlerinin çevreye etkilerinin bilimsel olarak araştırılmasını zorunlu hale getirmiştir.

Bu çalışmada, Muğla yöresindeki işletilmiş ve halen işletilmekte olan mermer ocaklarının çevreye olan etkilerinin CBS ve UA destekli olarak araştırılması yapılmıştır. Bu amaca yönelik olarak da, mermer ocaklarının yıllara bağlı olarak ocağın ne kadar genişlediği ve ne kadar yeşil alanın bozulduğu gibi konular araştırılmıştır. Bu araştırma için veri kaynağı olarak 2003-2007 yılları aralığında çekilmiş ASTER Level 3A01 uydu görüntüleri kullanılmıştır. Bunun yanında, ocakların konumları belirlenmesine yardımcı olması açısından MİGEM'den 2008 yılına ait ruhsat koordinatları tedarik edilmiştir.

Çalışma alanını Milas, Yatağan ve Kavaklıdere'de bulunan ocakların birçoğunu kapsayacak şekilde 107.568 hektarlık bir alan olarak seçilmiştir.

Çalışmanın veri kaynağını oluşturan 2003, 2005 ve 2007 yılına ait ASTER görüntüleri başta bazı düzeltmeler uygulanarak standart hale getirilmiştir.

Ocakların iki boyutlu düzlemde ne kadar alanı işgal ettiği ve belirli bir zaman aralıklarında sınırlarını ne kadar genişlettiklerinin tespiti için ocak sınırları temel alınarak alan hesaplamaları yapılmıştır. Bu hesaplamaları yapmak için ise belirli bant oranları kullanılmış ve en iyi sonuç Sultan vd.'nin (1987) oranı ile sağlanmıştır. 4/7-4/1-(2/3)*(4/3) RGB düzeninde mermer ocakları veya mermer jeolojik oluşumları net bir şekilde belirlenebilmiştir. Ocakların iki boyutlu düzlemde ne kadar genişlediklerinin belirlenmesi için örnek olarak üç ocak seçilmiştir. Şekil 4'de bu ocaklar gösterilmiştir. Görüntü üzerindeki ölçme teknikleri ile alanlar hesaplanmıştır (Şekil 5). Çizelge 2 ise bu üç ocağın 2003 – 2005 ve 2003 – 2007 yılları arasındaki ocakların kapladıkları alanın değişimini yüzde olarak göstermektedir. Şekiller ve tablo'dan görüleceği gibi ocakların genişlemeleri net bir şekilde yazılım ve uydu görüntüleri yardımı ile takip edilebilmektedir.

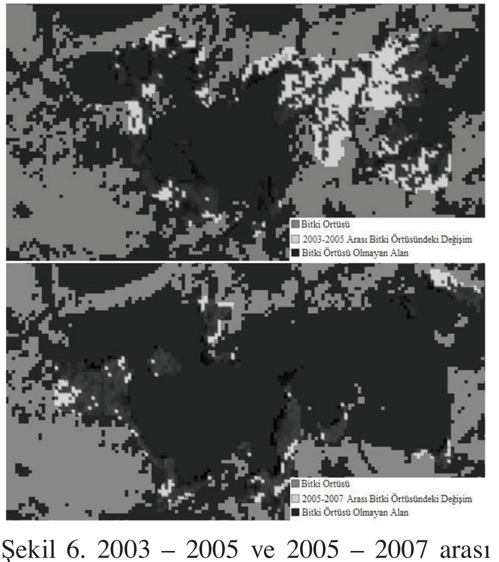


Şekil 5. 2003, 2005 ve 2007 yıllarında seçilen üç mermer ocağının iki boyutlu düzlemde alansal olarak genişlemeleri.

Diğer bir çalışma olarak da mermer madenciliğinden dolayı oluşan yeşil alanlardaki değişim incelenmiştir. Bitki örtüsündeki değişimin hesaplanabilmesi için ise Rouce vd.'nin (1973) geliştirdiği ve birçok çalışmada sıklıkla kullanılan NDVI'den yararlanılmıştır. NDVI ile bitki örtüsünün hesaplanması için yine 2003, 2005 ve 2007 yıllarına ait ASTER görüntüleri kullanılmıştır. (NIR – R) / (NIR + R) hesaplamalarından sonra 2005 yılına ait NDVI görüntüsü 2003 yılına ait NDVI görüntüsünden (2005NDVI – 2003NDVI) ve aynı şekilde 2007 yılına ait NDVI görüntüsü 2005 yılına ait NDVI görüntüsünden çıkarılmıştır (2007NDVI – 2005NDVI). Bu şekilde her yıl arasındaki değişim net bir şekilde belirlenmiştir. Kontrol amaçlı olarak da elde edilen değişim görüntüsü önceki yılın görüntüsü ile karşılaştırılmıştır (Şekil 6). Alansal değer olarak üç yıla ait bitki örtüsündeki değişim mukayese edilmiş ve sonuçlar Çizelge 3'de verilmiştir.

Çizelge 2. 2003 – 2005 ve 2005 – 2007 yılları arasında seçilen üç mermer ocağının alansal olarak genişlemelerinin yüzde miktarları.

	2003 - 2005	2005 - 2007
Ocak No: 155	%15,60	%15,67
Ocak No: 91	%7.44	%31,26
Ocak No: 129	%1,86	%4,70



Şekil 6. 2003 – 2005 ve 2005 – 2007 arası mermer ocakları faaliyetlerinden dolayı bitki örtüsündeki değişime örnek.

Çizelge 3. Alansal değer olarak üç yıla ait bitki örtüsündeki değişimi.

	Yıllar		
	2003	2005	2007
Toplam Bitki Örtüsü (Hektar)	46,244	51,157	49,549
Mermer ocağı faaliyetleri dolayısıyla oluşan bitki örtüsündeki değişim	2003 - 2005 Yılları arası değişim (hektar)	2005 - 2007 Yılları arası değişim (hektar)	
	40	31	

107.568 hektarlık çalışma alanının 2003 yılında %43'ü, 2005 yılında %48'i, 2007 yılında ise %46'sı yeşil alanla kaplıdır. Çizelge 3'den görüleceği üzere mermercilik faaliyetlerinden dolayı bitki örtüsünde oluşan değişim çok düşüktür. 2003 – 2007 yılları arasındaki değişim 40 hektar ve 2005 – 2007 yılları arasındaki değişim ise 31 hektardır.

Maden ocaklarının çevreye olan etkilerinin izlenmesi ve değerlendirilmesi CBS ve UA teknolojileri kullanarak mümkün olmaktadır. Bunun yanında, UA ile jeolojik araştırmalar ve maden aramaları birçok bant oranı sayesinde yapılabilmektedir.

Çalışmada kullanılan uydu görüntüleri farklı zamanlarda veya koşullarda çekildiğinden dolayı verilerde tam bir standarda erişilememiş olması olasıdır. Bu yüzden arazi çalışmaları bu tür araştırmalar için zorunlu olmakta ve hatanın en aza indirilmesi için uydu görüntülerinin yanında yerinde alınacak verilerin de kullanılması gerekmektedir.

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Selection of the Method of Coal Exploitation According to the Environmental Protection Criteria

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ABSTRACT The exploitation of coal deposits causes some damages, despite all benefits for the society. These damages are particularly expressed in relation to the environment. The influence of coal exploitation on the environment depends on many factors, especially on the method of exploitation. This paper deals with the analysis of the environmental impact on the method of coal exploitation.

1 INTRODUCTION

The coal exploitation, whether underground or open-cast, represents a significant activity in the energy industry of every country. In spite of the industrial, social and security interconnection of all the European countries, each country must have a certain level of energy independence, i.e. it has to be ready to utilize its own energy resources.

On the other hand, any exploitation of natural resources, and even mineral resources, is endangering the environment. The attitude that, in order to preserve the environment, we should discontinue the exploitation of natural resources is wrong; because over 70% of objects and facilities made by a man derive from mineral resources, and almost 80% of energy needs are satisfied by using mineral resources as natural energy sources (coal, crude oil, gas, uranium). A contemporary society, in utilizing natural resources, should apply the basic principles of the sustainable development: cost-effective exploitation with the highest environmental protection.

The objective of this Paper is to indicate the basic differences between the open-cast and underground coal exploitation from the aspect of the environmental protection. Certainly, in opting for a method of mining, other parameters should be also taken into account, but modern trends in mining industry increasingly comply with the needs for the environmental protection in all its activities, as well as in the coal exploitation.

2 RELATIONSHIP BETWEEN THE COAL EXPLOITATION AND THE ENVIRONMENTAL PROTECTION

The environmental protection is endangered by human factors with regard to:

- Exhaustion of natural resources,
- Destroying the natural environment and
- Polluting the environment.

Mining, as the industrial activity, endangers the environment in regard to all of the three aspects. The needs for mineral resources are evident, so it is necessary to adapt such exploitation, preparation and refinement technologies to the goal to minimize the environmental endangerment level. This requirement is particularly topical in the exploitation of coal, which is today, and will be in the future, an important energy resource.

Exhaustion of natural resources, when speaking about mineral raw materials, and also coal, has a certain severity, because it is about non-renewable natural resources. One ton loss in the exploitation is mainly a permanent loss. It is estimated that, due to inappropriate mining technologies, over one billion tons of coal are permanently lost in mines all over the world. Just for a comparison, such loss corresponds to five-year electricity demand of a developed country, such as France.

The coal exploitation results also in the **destruction of the natural environment** which arises in opening, preparation and extraction, and also in the process of tailing dumping. Mass production of low quality coal (lignite) means large open-cast mines, which causes large degraded areas of the mines and outside dumps. Destroying areas for the purpose of mines gives also rise to the intersection of underground watercourses, displacement of rivers, creeks etc.

In the present practice, the most attention is paid to **the environmental pollution** in mining exploitation, as the most visible aspect of the environmental endangerment. The coal extraction pollutes the soil, water and air. Erosion of tailing dumps, release of noxious matters, sedimentation of mechanical particles on the surface of the ground, exposure of soil to toxic matters resulted from the process of coal exploitation, purification and alike, lead to a lower or higher pollution of land in the vicinity of mines. Release of noxious matters and mechanical particles into neighbouring watercourses results in the physical and chemical pollution of waters located near or in the region of mines. In coal exploitation, the air is polluted due to the increase in the dustiness, exhaust gases, coal oxidation etc.

Making the decision on the method of coal deposit exploitation depends on many factors: from natural and geological conditions to economical and sociological effects. In the whole variety of these factors it is necessary to include also the environmental protection parameters, as one of the most important factors in decision-making. In is certain that, in decision-making, not all of the parameters can be taken into account equally, and the most difficult task in the phase of preliminary considerations is precisely the determination of the impact level of certain factors on the final decision.

Each coal deposit has its own specificities and it is defined by its position, occurrence, quality, reserves, prospective market etc. Some of these indicators can define, from the very start, the method of exploitation (for example depth of occurrence), but it is often necessary to make a detailed analysis with taking into account all of the criteria which are relevant to the determination of an exploitation method. In order to assess correctly the environmental impact of such method, it is necessary to envisage, in each particular case, the endangerment aspect in the underground or open-cast deposit exploitation.

The main objective which should be sought for in coal mining is a maximum recovery of a deposit in the process of the exploitation. In this way, the mitigation of the first environmental endangerment factor – **exhaustion of natural resources** - is reached. On the basis of many years' experiences in coal mining, we can state the following:

- Higher recovery of a coal deposit is reached by the open-cast exploitation for deposits of high and medium thickness (power) (80-90%). For lower thickness seams, from 2 to 5 m, this recovery ratio rate is lower, but it is still higher than in underground mining,
- In underground coal mining, by longwall face extraction methods, the deposit recovery ratio rate may range from 55 to 70% which depends on the mechanization type, thickness and regularity of the coal seam, preparation works etc.
- Pillar drilling methods, typical for steep and small deposits, have the lowest deposit recovery ratio (35 to 50%). It is typical with these extraction methods that a lot of losses arise from coal oxidation and mine fires.

Destruction of the natural environment exists both in the underground and open-cast coal mining. This aspect of the environmental endangerment has a number of forms, but the following two are prevailing: typical destruction in the process of the extraction and degradations of the land surface. Destruction of the natural environment in coal mining appears in the following cases:

- By open-cast coal exploitation in the process of coal extraction and stripping of overburden,
- By underground mining in the process of extraction and tunnelling, opening and preparation,
- By subsidence and damaging of the terrain due to creating an empty cavity or breaking in mines,
- By building accompanying facilities of a coal mine (channels, trenches, constructions and alike)
- By intersecting underground and surface watercourses and by modifying the environmental hydraulic situation etc.

Ground degradation is much more present in open-cast than in underground coal mining. The most frequent cases of the ground degradation appear:

- Due to the occupancy of land covered by open-cast mines,
- Due to setting up tailing dumps,
- Due to building up accompanying mine facilities (roads, railroads, pillars etc.)

Environmental pollution is also more present in open-cast than in underground coal mining. Harmful impacts can be divided into emission from a mine to the soil, water and air, and to immission from the polluted soil, water and air into the environment.

The soil is threatened by:

- Sedimentation of dust arising from the process of exploitation,
- Outflow of oil, lubricants and alike,
- Sedimentation of waste solid particles and solid matters,
- Technical and waste waters which are discharged into the soil and alike

In coal mining, water may be polluted as follows:

- Physical making watercourses turbid,
- Mixing with waste oils and lubricants,
- Discharge of technical and waste waters from drying and separation plants,
- Appearance of solid matters due to erosions of tailing and coal dumps etc.

The air in the vicinity of mines is often polluted due to the consequences of technological processes of coal exploitation. This pollution is incomparably higher in the vicinity of open-cast mines and it arises from:

- Escape of dust in the process of extraction, conveyance and dumping,
- Gases resulting from coal oxidation,
- Exhaust gases as a consequence of the operation of diesel drive machines and alike

It is typical that the polluted air often discharges harmful ingredients into the environment (emission) and thereat pollutes the soil and the water.

For each deposit it is necessary, before making a decision on the method of coal exploitation, to take the following measures in order to make a complete assessment of the environmental impact and to make a correct decision:

a) Determine the category of the environment where a deposit is located,
b) Identify potential environmental damages caused by certain mining technologies,
c) Define measures for moderating and remedying damages and include economic parameters,
d) Make a harmfulness quantification and determine technical coefficients for each of them,
e) Determine the best solution using one of the methods of multi-criteria decision-making.

The results thus obtained may serve as one of the criteria for determining a method of coal exploitation because they are comparable to other numeric indicators (exploitation costs, amount of investments, coal price and alike).

3 GENERAL ASSUMPTIONS IN SELECTING A DEPOSIT EXPLOITATION METHOD

In selecting a method of mineral exploitation, the satisfying solution is usually searched by considering a number of possible variants. Decision makers mostly build up their knowledge on the results obtained from the techno-economic and multi-criteria analysis.

By techno-economic analysis we obtain the exploitation method with minimum specific costs. This procedure takes also into consideration, in addition to costs of assets T_s and operating costs T_r, environmental protection costs T_{zo}, that is:

$$c_i = \frac{T_s + T_r + T_{zo}}{Q_{god}} \tag{1}$$

Environmental protection costs, in theory, may also be negative, considering the fact that total environmental protection costs are reduced for the profit generated from the environmental protection:

$$T_{zo} = T_{sz} + T_{rz} - D_z \tag{2}$$

Where T_{sz} and T_{rz} - costs of assets and operation related to the environmental protection.

The profit generated from the environmental protection may be expressed in dependence on the loss coverage ratio; such losses appear in the environment due to the mining exploitation G:

$$D_z = (1 - k_i)\, G \tag{3}$$

Where k_i - loss coverage ratio, it ranges from 0 to 1.

The multi-criteria analysis is more complex because it includes a number of factors which have smaller or greater influence on the selection of the best exploitation method. The selection of criteria is free and depends on the specificity of the given deposit. It is however important that the same criteria imply to all variants. In opting for the most suitable solution, the following criteria are usually taken:

- Costs of the obtained useful mineral unit,
- Amount of initial investments,
- Required manpower,
- Labour safety,
- Environmental protection,
- Power supply etc.

Environmental protection is in this case twice evaluated: as a separate criterion and through specific costs of the broken ground.

In general, the problem of multi-criteria decision-making may be put as follows:

$$\mathrm{Max}\left\{ k_1(a), k_2(a), \ldots k_p(a) \middle| a \in A, \right\} \tag{4}$$

Here k_1, k_2, ... k_p represent criteria which were previously selected, while A is the finite set of available variants which are to be ranked with a view to finding an optimal solution.

One of the most important tasks in all methods of multi-criteria analysis is determining weight coefficients ω_k which define the impact level of each criterion. Thereat, the following condition should be met:

$$\sum_{k=1}^{p} \omega_k = 1 \tag{5}$$

For concrete decisions, PROMETHEE, MATRIX and ELECTRA methods are commonly used. For each of them it is very important to determine quantitative indicators, especially in the domain of environmental protection parameters. That is the reason why classifications of environments are made and that individual impacts of certain influences on each of them are determined.

4 CONCLUSION

The environment is endangered by the exploitation of mineral deposits, as non-renewable natural resources. In making the decision on the need for and the method of exploitation, it is necessary to include, as one of the most significant criteria, the environmental protection. In order to properly apply this criterion, it is necessary for decision-makers to be well informed about all forms of environmental endangerment by mining activities.

Analyzing all possible aspects of the environmental endangerment, quantifying them and putting them into equal position as other criteria, lead to assumptions which will result in making the least erroneous decision. The existing methods of techno-economic and multi-criteria decision-making provide enough possibilities to reach the best solution with respect to the method of exploitation of particular mineral deposits. A special significance thereat has the reliability of input data, on the basis of which it is possible to get a picture on the environmental endangerment degree.

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Heavy Metals in the Water of the River Kalnistanska and the Vicinity

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ABSTRACT The paper presents the results and conclusions from the investigations carried out on the water contamination with heavy metals in the River Kalnistanska and its vicinity. For the increased content of heavy and toxic metals in the near vicinity of river Kalnistanska, especially in the water systems, large contribution have increased ore mineralization of the area, sewage waters from the feces of the town and neighboring villages, and industrial objects (battery factory etc.).

The examinations are an attempt to determine the real and the physical state of heavy metals in the river water and its tributaries. Samples were analyzed with ICP-AES method. Earlier experiences indicated that the following elements should be expected: Mn, Fe, Al, Zn, As, Cd, Cu, Ni, Co, Ag, Cr, Ti with possible occurrence of increased concentrations of maximum allowable amounts.

Analysis and data interpretation yielded increased concentrations of Mn, Fe, Al, Pb, Zn, As, Cd and Cu. The entire drainage system in the Kalnistanska in the Municipality of Probistip has been polluted. It was found that the major pollutants are the Zletovo Mine and the discharge of waste waters containing heavy metals.

1 INTRODUCTION

Investigated area is located in the eastern part of the Republic of Macedonia near to town of Probistip. River Kalnistanska flow through mountain, lowland and canyon terrains. There are many hills above the Probistip where the river Kalnistanska is situated and it is lay down along the fault lines.

After the exit at village Kalniste, along the river Kalnistanska valley alluvium is wide spread with width of 300 to 1000 m. Left bank is flatland with characteristic little uplifts.

Hydrographic net of this area belongs to the river Bregalnica which has water through the whole year. Left affluent of river Bregalnica is river Zletovska in which river Kalnistanska influents. These rivers are with permanent flow through the whole year. In the rainy days there are enough water, and in rainless summers water potential decreased. Only river Kalnistanska could dry up (rarely).

Over the past few decades environmental pollution was not given sufficient attention. However, the issue was given a priority over the few years. Heavy and toxic metals in drinking water has been of special concern. It is of note that the wells from which the population in several municipalities receive water for the water supply system are located in the River Kalnistanska alluvion. This has been a serious problem for the people in the Municipality of Probistip.

People in the area use the waters from several tributaries for drinking water and household needs. They also use the water for irrigation, agriculture particularly in the production of many kinds of vegetables. Investigations carried out so far point out possible water pollution.

Another reason for the implementation of the project is the strict laws on the quality of the human environment.

The studied area included the Rivers Kalnistanska and Zletovska as far as its estuary to the Bregalnica (Figure 1). The area marked with an oblong is the place where water samples were collected.

Figure 1. Map of the Republic of Macedonia with location of investigated area.

2 METHODS OF WORK

Preliminary field activities were carried out in order to obtain data on the situation of the terrain.

Field sampling also defined measurement points and cross-section lines. Topography of sampling stations was defined including the cross section lines for sample collection. The initial phase consisted of collecting water samples in the middle of the river course in a 1-l. clean plastic vessel. Sample collection also included filtering through paper the openings being 45 μm. Before closing the vessel acidifying with 0.4 ml nitric acid (HNO_3) of 50 % was done to avoid possible metal settling on the wall and the bottom of the vessel. Laboratory examinations included ICP-AES method and their interpretation.

3 RESULTS AND DISCUSSION

The results obtained from flowing water samples are given in Table 1, with comparison between maximum allowable (MAC) concentration of heavy and toxic metals for III – IV classes.

Data shown in Table 1 make it possible to define the amount of heavy metals in the Kalnistanska River water and its tributaries and the reasons for their occurrence.

Based on the data presented in Tab. 1, certain constants can be given concerning the presence of particular heavy metals in the waters of the Kalnistanska River.

The data for the concentration of zinc (Table 1) indicate its increased presence in most of the samples which were analyzed. The greatest concentrations of zinc are found in the sample Kr-9, which are 0.28 mg/l. The increased concentrations of zinc are also found in the samples: Kr-1, Kr-3 and Kr-5. In the remaining samples the concentration of zinc is present in amounts less than the standard ones. Generally, it can be concluded that the entire research area is contaminated with zinc. The increased concentrations of zinc are a result of the active working of the Mine for lead and zinc Zletovo.

Table 1. Contents of heavy metals in flowing waters in the Kalnistanska River (mg/l).

Sample	Fe	Mn	Zn	Pb	Cu	Cd	As	Ag
Kr-1	0.290	0.016	0.1700	0.013	0.007	0.0120	0.014	0.0029
Kr-2	0.050	1.150	0.0130	0.011	0.005	0.0007	0.018	0.0041
Kr-3	0.043	0.210	0.2400	0.024	0.034	0.0008	0.028	0.0022
Kr-4	0.029	0.017	0.0580	0.019	0.019	0.0012	0.044	0.0040
Kr-5	0.072	1.310	0.1800	0.043	0.023	0.0010	0.026	0.0025
Kr-6	0.110	0.025	0.0072	0.010	0.005	0.0180	0.070	0.0015
Kr-7	0.180	0.120	0.0440	0.010	0.010	0.0010	0.055	0.0043
Kr-8	0.240	0.025	0.0310	0.029	0.016	0.0011	0.022	0.0015
Kr-9	0.280	0.218	0.2800	0.006	0.003	0.0110	0.016	0.0017
Standard	1.0	1.0	0.2	0.03	0.05	0.01	0.05	0.02

The increased concentration of lead are noticed in the samples Kr-5 (0.043 mg/l), Kr-8 (0.029 mg/l), Kr-3 (0.024 mg/l) and Kr-4 (0.019 mg/l). It is obvious that increased concentrations of lead are found in most of the samples, but the great sample concentrations of lead are found in those samples where increased concentrations of zinc were also found, which confirms the statement of the great influence of the hydro-waste dump and the active working of the Mines Sasa for the pollution of the environment with these metals.

In all the samples cadmium is found in extremely larger concentrations compared to maximum allowed concentration. The great sample concentrations of cadmium are noticed in the samples Kr-1 (0.12 mg/l) Kr-6 (0.018 mg/l) and Kr-9 (0.011 mg/l). This behaviour of the cadmium is due to its geochemical characteristics (easily soluble, low mobility). The increased concentrations of cadmium follow the parts which are contaminated with zinc because it geochemically follows the minerals in the zinc.

The increased concentrations of copper are found in most of the samples, but the great sample concentrations of copper found in the samples Kr-3 (0.034 mg/l), Kr-4 (0.019 mg/l), Kr-5 (0.023 mg/l) and Kr-8 (0.016 mg/l), while in the remaining samples the concentrations of copper are close or less than in maximum allowed concentration. The presence of the copper in the waters of river Kalnistanska is a result of the occurrence of chalcopyrite in an association with the minerals of the lead and zinc.

Iron is present in concentrations lower than maximum allowed concentration and is not a significant contaminant of the drainage system of the river Kalnistanska.

Attention should be paid for the high concentrations of the manganese in the samples: Kr-2 (1.15 mg/l) and Kr-5 (1,31mg/l). The reason for the occurrence of the manganese in high concentrations in the waters of the river Kalnistanska as a presence of the waste waters from the Mines Zletovo.

4 CONCLUSION

Data obtained with laboratory examinations on heavy metal contents in the waters of the Kalnistanska and its tributaries indicate that a number of heavy metals occur in increased amounts, some of them in amounts lower than the maximum allowable.

The most important contaminants are lead, zinc, cadmium, silver, arsenic and silver.

The increased concentrations of heavy metals are a consequence of the geological composition of the terrain, anthropologic activities such as early mining and stockpiling of waste material, the use of fertilizers in agriculture as well as the physical and chemical character of the water solutes.

It can be said in the end that the largest distribution and resulting contamination with heavy metals were found in the mine and flowing into the Kalnistanska. From ecological point of view this entails the need of rehabilitation of the area in order to prevent flora and fauna intoxication in the water medium an the surrounding and prevent the negative effect on the health of the population.

The metal concentration in flowing waters is also influenced by their geochemical characteristics and pH and Eh factors.

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Harşit (Giresun) Granitlerinin Mineralojik - Petrografik Analizi ve Granitlerin Bina Malzemesi Olarak Kullanımı için Doğal Radyoaktivite Seviyesinin İncelenmesi

Investigation of Harşit (Giresun) Granites Mineralogical-Petrography Analysis and the Level of Natural Radioactivity of Granites Used as Building Materials

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ÖZET Dünya mermerciliğinde hem üretim hacmi, hem de çeşitlilik açısından ülkemiz önemli bir konumdadır. Gelişen mermercilik sanayisi ülke ekonomisine büyük bir katma değer ve istihdam olanağı yaratmaktadır.

Bu çalışmada, Harşit granitlerinin kapalı mekanlarda döşeme ve kaplamada kullanılması için doğal radyoaktivite seviyesi tayin edilmiş ve granitin mineralojik petrografik özellikleri araştırılmıştır.

Mineralojik ve petrografik analiz için polarizan mikroskopta optik incelemeler yapılmıştır. Aynı zamanda bu çalışmada Harşit granitlerinin, HPGe detektörü kullanılarak örneklerin doğal radyoaktivite seviyesi tayin edilmiştir. Gama spektometrisiyle doğal radyoaktivitesi belirlenen örneklerin özellikle ^{226}Ra, ^{232}Th ve ^{40}K içeriği ölçülmüştür. Ölçümler sonucunda radyoaktivite değerlerinin bina malzemesi için belirtilen seviyenin altında olduğu bulunmuş. Ayrıca granitlerin ^{226}Ra, ^{232}Th ve ^{40}K aktivite konsantrasyonu dünyanın değişik granit çeşitleri ve bina malzemeleriyle karşılaştırılmıştır.

ABSTRACT In world marble industry, Turkey is an important country regarding variety and production volume. The improving marble industry has been contributed to the national economy and provided employment.

In this study, level of natural radioactivity of the use of Harşit Granites were determined for closed areas flooring and coatings. Moreover, Mineralogical- Petrography properties were investigated .

Mineralogical, petrographical and optical analyses were carried out by using a polarisan microscope. Meanwhile, in this study, the level of the natural radioactivity in samples has been determined using a HPGe (high purity Germany detector) detector. The natural radioactivity of the samples was measured by gamma-ray spectrometry. In particular the content of ^{226}Ra, ^{232}Th and ^{40}K in each sample were measured. These radionuclide's were found to be in low levels for the uses of building materials. Moreover, the activity concentrations of ^{226}Ra, ^{232}Th and ^{40}K of granites are presented and compared to those of other building materials as well as other granite types used all over the world. In order to assess the radiological impact from the granites were also investigated.

1 GİRİŞ

Dünyanın en zengin ve kaliteli mermer yataklarının büyük bir kısmı Türkiye'nin içinde bulunduğu Alp kuşağı içindeki ülkelerde yer almaktadır. Türkiye'nin geniş alanlarını kaplayan mermer oluşum rezervleri henüz kesin olarak ortaya çıkarılmamıştır. Çeşitli kaynaklar taranarak ve jeolojik etüd raporları incelenerek Türkiye'nin mermer rezervleri; 5,2 milyar m^3 (13,9 milyar ton) toplam mermer rezervine sahiptir. Bu durumda dünya mermer rezervlerinin yaklaşık % 33'ünün ülkemizde bulunduğu tahmin edilmektedir (Onargan vd., 2004).

Türkiye'deki doğal taş rezervlerinin büyük bir bölümü Afyon, Balıkesir, Muğla, Eskişehir, Denizli, Tokat, Çanakkale, Konya, İzmir, Kırşehir ve Elazığ illerinde bulunmaktadır.

Çizelge 1'de doğu Karadeniz bölgesinde illere göre doğal taş potansiyelini incelediğimizde, belirlenen alanlarda tespit edilen toplam rezerv 246 milyon m^3'tür. Rezervin % 88.32 gibi önemli bölümünü granit, % 7,8'ini kireçtaşı, % 3.86'sını traverten ve çok az % 0.02 kısmını da metamorfik yataklar oluşturmaktadır.

Çizelge 1. İllere göre doğaltaş potansiyeli (Yılmaz, 2006).

İl	Muhtemel Rezerv ($x10^3$ m^3)	Genel Özellik
Giresun	115.965	Blok verebilen, az çatlaklı
Ordu	64.025	Blok verebilen, çeşitli renk ve desenlerde
Rize	32.100	Çatlaklı ve kırıklı
Trabzon	14.760	Az çatlaklı-kırıklı
Bayburt	9.840	Az çatlaklı-kırıklı
Gümüşhane	9.300	Genelde sert, gevşek ve bozunmalar da var

Doğu Karadeniz Bölgesi, Ordu ilinden Artvin iline kadar uzanan geniş alanda başta granit olmak üzere değişik renk ve desenlerde önemli ölçüde doğaltaş rezervine sahiptir. Bu doğaltaş rezervinin yaklaşık değeri 90 milyar dolar civarındadır (Korkmaz, 1996).

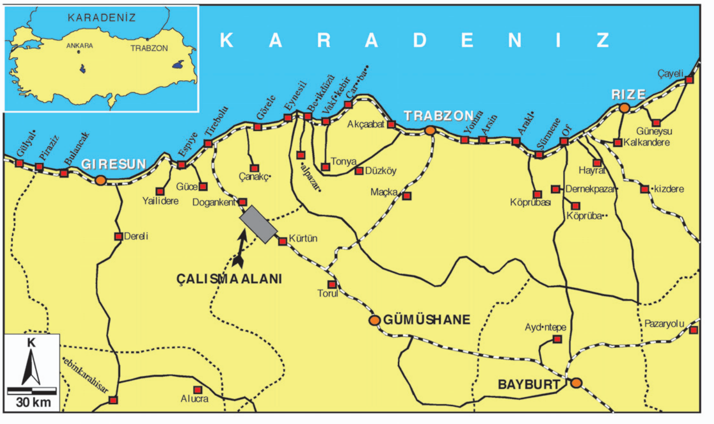
Bilindiği üzere insan yaşamının önemli bir kısmı (ofis, ev) kapalı mekanlarda geçmektedir. Binalar; dışarıdan gelen kozmik ve kıtasal orijinli radyasyona karşı koruyucu olmakla beraber, bazen yapı malzemelerindeki radyasyon içeriğine bağlı olarak içerdeki radyoaktivite dışarıdan fazla olabilir. Yapı malzemeleri gerek doğal yollarla (yapı taşlan, mermerler v.b), gerekse de çeşitli ham maddeler kullanılarak yapay olarak üretilir (çimento, kompoze taş, mermerit v.b). Her iki durumda da hammaddenin özelliklerinden kaynaklanan radyoaktivite içeriği söz konusudur. Bu nedenle gelişmiş ülkeler, insan sağlığını korumak ve bina malzemelerinin spesifik aktivitelerini karşılaştırmak amacı ile tüm dünyada ortak bir indeks kullanmaktadırlar (Türkmen, 2003).

İnsan ve çevre sağlığının radyasyondan etkilenmesi 3 temel nedenden kaynaklanmaktadır. Bunlar; toprak, atmosferde bulunan doğal radyoaktif elementler, nükleer silah denemeleri ve nükleer reaktör kazalarıdır. Toprak ve atmosferde bulunan doğal radyoaktif elementlerin çevreye yaydığı radyasyonu önlemek mümkün değildir. Ayrıca. toprak kökenli yapı malzemelerinde de (kum, çimento. tuğla vb.) bir miktar radyasyon vardır. Miktarı az olan malzemelerdeki radyasyon insan sağlığını önemli ölçüde etkilemez. Fakat radyasyonun hangi kaynaktan ve ne kadar sürede alındığı önem taşımaktadır (Dyson, 1993).

Granitin ana kullanım alanı dekoratif taşlara ihtiyaç duyulan inşaat sektörüdür. Binaların iç cephesinde; yer döşemesi ve duvar kaplamasında, basamaklarda, sütunlarda, şöminelerde, mutfak ve banyolarda, binaların dışında ise; dış cephe kaplamasında yaygın olarak kullanılmaktadır. Ayrıca dış zeminlerde parke olarak kullanıldığı gibi bir çok dekoratif eşya yapımında ve mezar taşlarında da kullanılmaktadır. Hijyenik koşulların gerekli olduğu umuma açık ortamlarda veya üretim tesislerinde granit kullanımı artmaktadır (örneğin hastaneler, hava alanları vb). Granit özellikle hem desen (dekorasyon) hem de dayanıklılık açısından titizlik gerektiren işlerde kullanılmaktadır (DPT, 1996).

1.1 Çalışma Alanı

Çalışma alanı, Doğu Karadeniz Bölümü'nün Giresun iline bağlı Doğankent ilçesi merkezinin 4 km kuzeyinde 10 km güneyine kadar Harşit vadisi boyunca uzanmaktadır. Sahilden 30 km içerde bulunan Doğankent ilçe merkezi dağlarla çevrilmiştir. Doğankent'e ulaşım Karadeniz sahilinde Trabzon-Giresun karayolu üzerinde yer alan Tirebolu (Giresun)'dan 33 km uzunluğundaki yolla sağlamaktadır. Şekil 1'de Çalışma alanına ait yer bulduru haritası verilmiştir.



Şekil 1. Çalışma alanına ait yer bulduru haritası.

1.2 Harşit Granit Ocağının ve Çevresinin Genel Jeolojisi

Çalışma alanı granitleri, ilk defa Schultze-Westrum (1959) tarafından Harşit graniti olarak adlandırılan birim daha sonraları Gedikoğlu (1978) tarafından Harşit granit karmaşığı olarak adlandırılmıştır. Bunlar granit, alkali granit, alkali feldspat granit, granodiyorit, kuvarslı diyorit ve diyorit gibi kayaçlardır.

Çalışma alanı ve çevresinin genel jeolojisine baktığımızda, yaşlıdan gence doğru Hamurkesen Formasyonu, Berdiga Kireçtaşı, Yavuzkemal Formasyonu ve Harşit Granitoidi ile Kuvaterner yaşlı alüvyonlar yüzeylenme vermektedir.

Hamurkesen ismi Doğu Karadeniz'de yapılan çalışmalarda ilk kez Ağar (1977) tarafından kullanılmıştır. Hamurkesen Formasyonu andezit, bazalt ve bunların piroklastitlerinlerinden oluşmakta olup çalışma alanında Harşit Granitoyidi tarafından kesilmiştir.

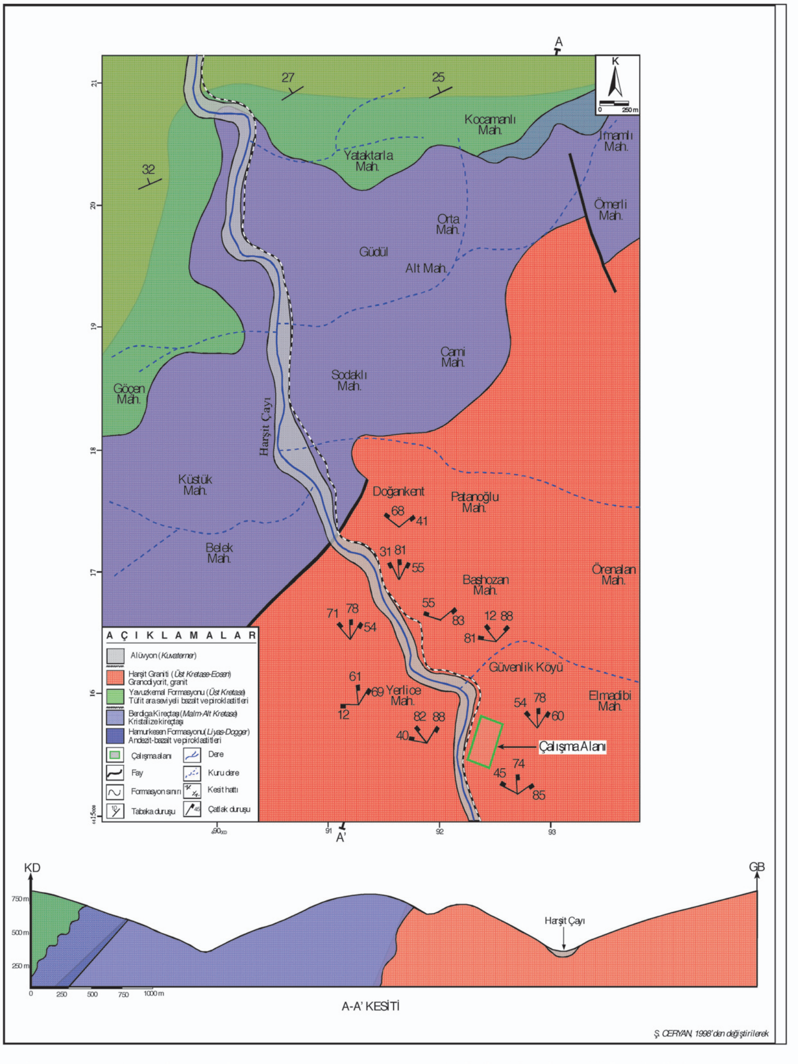
Berdiga Kireçtaşı ismi ilk kez Pelin (1977) tarafından Alucra-Giresun yöresindeki kireçtaşı yüzeylemelerine dayanarak adlandırılmıştır. Formasyon, genellikle kristalize kireçtaşından ve dolomitik kireçtaşından oluşmakta olup uyumlu olarak Hamurkesen Formasyonu üzerine oturmaktadır. Berdiga Kireçtaşları çalışma alanının kuzeydoğusunda çok az bir alanda yüzeyleme vermektedir.

Yavuzkemel Formasyonu ismi ilk kez Doğanket'in batısında yer alan Dedeli-Giresun yöresinde çalışan Boynukalın (1991) tarafından kullanılmıştır. Formasyon volkano-tortul nitelikli olan tüfit ara seviyeli bazalt ve piroklastitlerinden oluşmakta olup uyumsuz olarak Berdiga Kireçtaşları üzerine oturmaktadır.

Harşit Granitoyidi ismi diyoritten granite kadar bileşim değişikliği gösterdiği için ilk kez Köprübaşı (1992) tarafından kullanılmıştır. Genellikle granodiyoritten oluşmakta olup Hamurkesen Formasyonunu keserek yerleşmiştir.

Çalışma alanının içinde yer alan mermer ocağındaki şevlerden yapılan hat etüdü ölçümleri ile ortama süreksizlik ara uzaklığı 95 cm olarak belirlenmiş ISRM (1981) tarafından yapılan tanımlamalara göre ocak alanında yüzeyleyen granodiyoritlerin "geniş ara uzaklıklı" sınıfına girdiği belirlenmiştir. Bu da ocak alanından maksimum 95 cm genişliğinde blokların üretilebileceğini göstermektedir.

Çalışma alanındaki en genç oluşukları ise Harşit vadisi boyunca kumlu, siltli, çakıllı ve bloklu malzemelerden oluşan alüvyonlar oluşturmaktadır. Şekil 2'de (Ceryan, 1999)'dan küçültülmüş Harşit (Doğankent) granit ocağı ve çevresine ait 1/25000 ölçekli jeoloji haritası gösterilmiştir.



Şekil 2. Harşit (Doğankent) granit ocağı çevresine ait 1/25000 ölçekli jeoloji haritası

1.3 Mineralojik ve Petrografik Özellikler

Kayaçların çeşitli iç yapı özelliklerinin tanınmasında kullanılan petrografik analizler, mermerlerde ocaktan kullanım yerine kadar, mermer karakterizasyonunda önemli bir rol oynamaktadır. Mermerlerde sertlik, kırılganlık, kesilebilme, parlatılabilme, cila alma gibi parametreler malzemenin iç yapısıyla ilgilidir.

Mikroskop ve X-ışınları yardımı ile petrografik analizler yapılarak mermerlerin makroskobik olarak belirlenemeyen bileşenleri belirlenebilmektedir. Petrografik analiz ile mermerlerde iç yapı (dokutestür) özelliklerinin tanınması yanında içerdiği mineral tür ve boyutları belirlenir.

1.3.1 Mikroskobik Özellikler

Harşit Granitlerinin mineralojik bileşimini belirlemek amacıyla, granitin özelliğinin temsil edecek şekilde 6 adet parlak kesit optik mikroskopda incelenmiş, 11 adet ince kesit polarizan mikroskopta incelenmiştir.

Kayaçlar ne kadar ince taneli ise sertliği o kadar fazladır. Kayaçlar arasında görülen bu sertlik farkı tane boyutundan ve içinde bulundurduğu mineralden kaynaklanmaktadır. İnce taneli granitler, iri taneli granitlere oranla daha serttir. Mermerin içine kalsitin yanı sıra dolomit de girerse sertlik biraz artar. Mermer içinde silikat minerallerinin görülmesi ile sertlik 4'ün üzerine çıkmaya başlar. Özellikle kayaçta kuvars bulunması sertliği daha yukarıları tırmandırır. Sert mermer grubuna giren granit, siyanit, gabro gibi silikat minerallerince zengin kayaçların sertliği 6-7 arasında değişmektedir. Bu değişim o kayaçları oluşturan feldspat, kuvars, piroksen, amfibol minerallerinin oranının değişimi ile gerçekleşir. Mermeri oluşturan minerallerin birbirleri ile olan ilişkileri bu minerallerin düzenli veya düzensiz yapıda olmaları kayacın dokusunu oluşturur. Tane sınırları düzgün ve iç içe grift değilse mermer daha yumuşaktır. Tane sınırları dantel gibi girintili çıkıntılı ve diğer tanenin içine girecek şeklinde uzanan parçalar içeriyorsa bu tip mermer daha serttir (Kun, 2000).

Bazı granitlerden yazı strüktürü görülmesi, bu tür granitler diğer granitlere nazaran daha az kuvars içerirler. Bunların kimyasal bilişimi, kuvars ve potasyumlu feldispatın ötektik karışımına çok yakındır ve dolayısıyla yazı strüktürü çoğu kez bu iki mineralin ötektik eriyikten beraberce kristallenmesine bağlıdır (Aslaner, 1989). Harşit granitlerinin ince kesitlerinin incelenmesi sonucunda yazı dokusu olan kesitler görülmüş ve bu granitlerin kuvars içeriği minerallerin yaklaşık %15-20'sini oluşturmaktadır.

İnce kesitleri yapılan granitlerden aşağıdaki sonuçlar çıkarmıştır. Harşit granitlerine ait Şekil 3'de parlak kesitler, Şekil 4'de ince kesitler verilmiştir. Kesit incelemeleri sonucunda, Harşit granitlerinde bulunan mineraller ve özellikleri aşağıda verilmiştir

Harşit granitleri gri renkte ve ince taneli yapıya sahiptirler. Kesit incelenmesin yarı öz şekilli taneli dokular görülmüştür.

Plajiyoklaz; kayaçta en çok bulunan mineral olup, açık renkli minerallin yaklaşık % 60'nı oluşturur. Bazen ters zonlu yapıda olup dış kesimleri genellikle andezin, iç kısımları albit bileşimlidir. Yer yer ortoklaz kristalleri tarafından çevrelenirken, genellikle dengesiz dokular içerir.

Ortoklaz; kayaçta yaklaşık % 20-25 oranında bulunur. Genellikle plajiyoklazları çevreleyen pozisyondadır. Ayrışma yok pertifik özellik gösterip içlerinde alkit inklüzyonları taşırlar ve kuvarsla birlikte diğer kristallerin arasını doldurmaktadır.

Kuvars, açık renkli minerallerin yaklaşık % 15-20 sini oluştulur. Öz şekilsiz olup yer yer yazı dokusu oluşturacak şekilde ortoklazla beraber bulunur.

Hornblend; Kayaçta yaklaşık % 15-20 oranında, yarı öz ve öz şekilsiz olarak bulunurlar. Yer yer ayrışarak opak mineralleri (limonit- manyetit) dönüşürler yer yerde kloritleşmişler.

Biyotit; Kayaçta % 3-5 oranında yer yer kloritleşmiş, yer yer de opaklaşmış olarak görülür.

Opak mineral; Bir bölümü birincil bir bölümü de ikincil olarak belirlenir. Birincil olanlar öz şekilli, ikincil olanlar ferromagnezyon mineralleri (hornblend, biyotit) ayrışma ürünü olarak gelişmektedir. Opak mineler olarak pirit ve manyetittir.

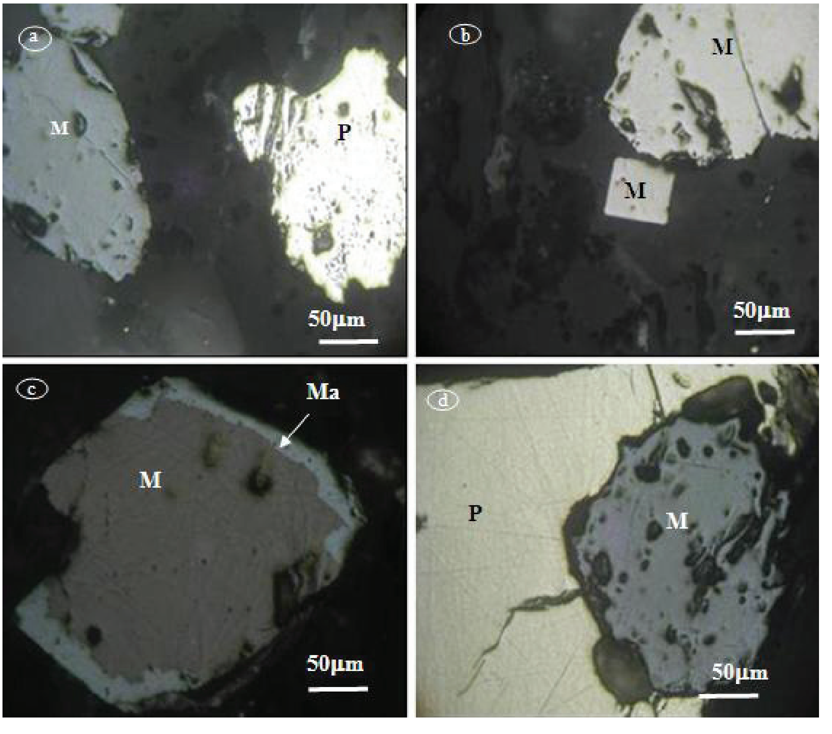
Klorit; kayaçta ikincil mineral olarak genellikle hem hornblend hem de biyotitin ayrışma ürünü olarak bulunur.

Mikro çatlaklı yapıları; kayaç içindeki mineraller de mikro çatlak oranı son derece düşüktür. Kayaçta klorit dışında önemli bir ayrışma minerali yoktur. Çizelge 2'de Çeşitli ülkelerdeki granitlerin petrografik özelliklerinden mineralin rengi, dokusu, kayaç çeşidi ve içerdiği minelerler verilmiştir.

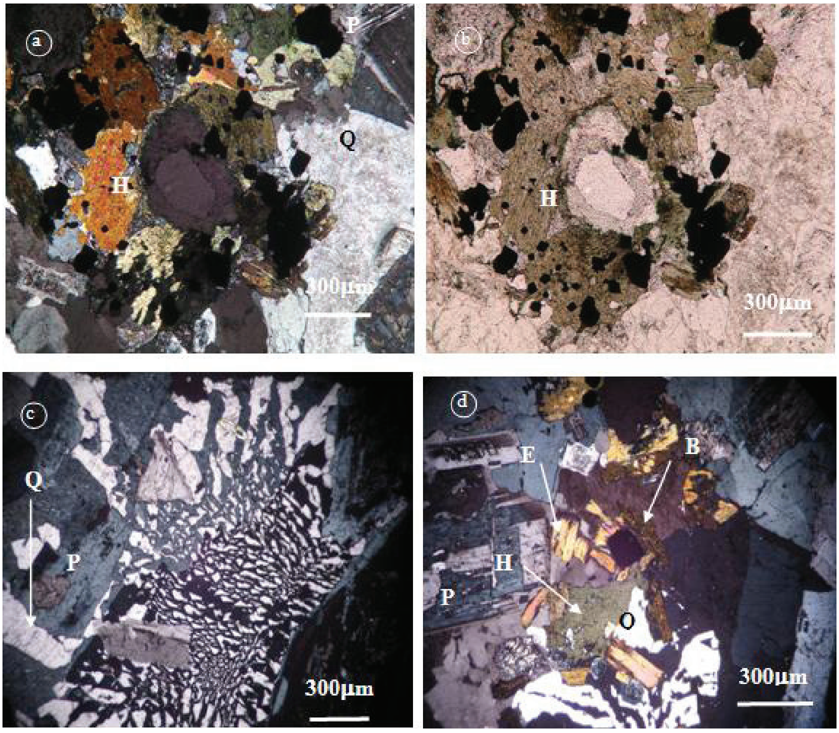
1.4 Harşit Granitlerindeki Radyoaktivite Seviyesi

Radyoaktivite seviyesinin belirlemesi için, ocağın 6 farklı noktasında alınan numuneler, çekiçle küçük parçalara kırıldıktan sonra 100±5°C'de etüvde kurutulduktan sonra, halkalı değirmende 2 dakika öğütülüp çapı 6 cm ve yüksekliği 5 cm olan plastik kutuların içine doldurup kutuların ağızları sıkıca kapatılarak 1 ay süreyle bekletilir. Böylece ^{238}U ve ^{232}Th ürünleri arasındaki radyoaktif dengenin oluşması sağlanır ve daha sonra numuneler sayıma hazır hale gelir. Şekil 5'de içinde radyoaktivite seviyesi ölçek için alınan numuneler kaplar içinde ki gösterimi verilmiştir.

Alınan numuneler, Yüksek Saflıkta Germanyun Dedektörü kısaca HPGe Detektörü olarak adlandırılan gama spektroskopik ölçümleri için ORTEC GEM55P4 Model HPG'e detektörü kullanıldı. Bu detektör özden yarı iletken prensibine göre çalışan yarı iletken kristalden oluşmaktadır (Şek. 6, Şek. 7.).



Şekil 3. Harşit granitinin parlak kesit incelenmesi; a) yarı öz biçimli ve öz biçimsiz, pirit ve manyetitler mevcut, b) öz şekilli ve öz şekilsiz manyetitler, c) ortada manyetit etrafında hematite dönüşmüş martitleşmeler var, d) manyetit piriti sarmış, piritler büyük ve kırıklı yapıya sahiptir. (P: Pirit; M: Manyetit; Ma: Martitleşme)



Şekil 4 İncelendiğinde a) Çift nikoldeki farklı renklerde hornblendler, kuvars ve plajoklazlar mevcut, b) Tek nikoldeki tek mineral hornblend, c) Çift nikoldeki yazı dokusu üzerindeki kuvars ve plajoklazlar, d) çift nikoldeki koyu yeşil hornblend, plajoklazlardaki zonlanma, epitot ve biyotit gösterilmiştir. (P: Plajoklaz; Q: Kuvars; H: Hornblend; E: Epitot; B: Biyotit).

Rn-226 ve kısa yan ömürlü ürünlerinin neden olduğu içsel maruz kalmanın yanında bina malzemelerinde bulunan ^{40}K, ^{232}Th, ^{226}Ra radyonüklidlerinden yayımlanan gamalar da bina içinde dışsal maruz kalmaya katkıda bulunurlar. Bu nedenle Ra, K, Th içeren bina malzemelerinin spesifik aktivitelerini karşılaştırmak için radyum eşdeğer aktivitesi Ra(eq) adı verilen ortak bir indeks kullanılmaktadır ve Ra (eq) aktivitesinin 370 Bq/kg'mı geçmemesi istenmektedir (Petropouios vd., 2002).

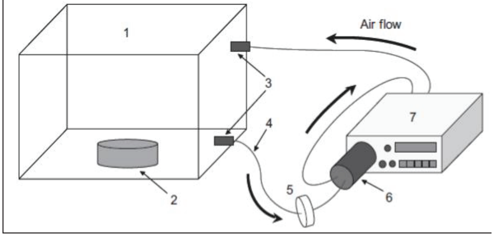
Şekil 5. Radyoaktivite seviyesi ölçek için hazırlanan numuneler

Çizelge 2. Bazı granitlerin petroğrafik özellikleri (Pavlidou vd. 2006).

Granit ismi	Ülke	Renk	Doku yapısı	Kayaç Çeşidi	Mineral içeriği
Harşit graniti	Türkiye	Gri	Orta-ince taneli	Granit	Qz, PI, Bi, Hem, Py, Ep, Op
Savatierra	Ispanya	Açık gri	İri taneli porfirik	Alkali granit	Qz, PI, K-Bi, All, Chl, Kaol, Ser, Tit, Zr
Rosa porrino	İspanya	Pembemsi	İri taneli porfirik	Alkali granit	Qz, Pl, KeF, Bi, Zr, Al, Kaol, Ser
Blanco real	İspanya	Beyaz	Orta-iri taneli granitik	Alkali granit	Qz, Pl, KeF, Bi, Mu, Zr, Mz, Ap, Chl, Kaol, Ser
Topazio	Brezilya	Sarı açık kahve	İri taneli granitik	Granit	Qz, Pl, KeF, Bi, Grt, Zr,Op, Ac, Kaol, Ser, Hem
Yellow cecilia	Brezilya	Sarı	İri taneli porfirik	Granit	Qz, Pl, KeF, Bi, Grt, Zr
Blanco crystal	İspanya	Beyaz	Orta-iri taneli granitik	Granit	Qz, Pl, KeF, Bi, Tit, Zr, Mz, Chl, Kaol, Ser
Napoleon	Brezilya	Sarı	İri taneli porfirik	Granit	Qz, Pl, KeF, Bi, Grt, Mu, Zr, Kaol, Ser
Balmoral	Finlandiya	Açık kırmızı	İri taneli granitik	Granit	Qz, Pl, KeF, Bi, Mu, Zr, Ap, Fl, Mz,
African red	Güney Afrika	Koyu kırmızı	İri taneli granitik	Granit	Qz, Pl, KeF, Zr, Chl, Kaol, Ser
Multicolour	Hindistan	Açık kırmızı-koyu gri	Orta-iri taneli granitik	Granit	Qz, Pl, KeF, Bi, Mu, Tit, Zr, Ap, Op, Kaol, Ser
Baltic Brown	Finlandiya	Koyu kahve	İri taneli	Granit	Qz, Pl, KeF, Bi, Amph, Zr, Ap, Op, Kaol
Gris perla	İspanya	Beyaz açık gri	İri taneli porfirik	Granit	Tit, Zr, Ap, All, Chl, Kaol, Ser
Emerald	Norveç	Renkli siyah	İri taneli granitik	Kuvarslı monzonit	AeF, Bi, Amph, Cpx, Ap, Op, Py, Hem
Marina pearl	Norveç	Renkli mavi-gri	İri taneli granitik	Kuvarslı monzonit	AeF, Bi, Cpx, Ol, Zr, Ap, All, Ep, Op, Py
Zimbabwe	Güney Afrika	Siyah	Orta-iri taneli granitik	Kuvarslı monzodiyorit	Qz, Pl, Opx, Cpx, O, Py Hem
Africa nero	Güney Afrika	Siyah koyu kahve	Orta-iri taneli granitik	Kuvarslı gabro	Qz, Pl, Opx, Cpx,

Qz = Kuvars, PI = Plajoklaz, K-F= K-feldispat, A-F= alkali feşdispat, Bi= biotit, Amph = anfibol, Ol= olivin, Mu= Muskovit, Zr= zirkon , Ap = apatit FI= Florit , All= albit , Ep= epitot, Chl= klorit, Kaol Kaolinit, Ser= serizit, Tit = titan, Hrm= hematit, Py= pirit, Op= opak minerali

Şekil 6. Ortec GEM55P4 Model HPGe dedektörü.



Şekil 7. Radon ölçümü için kullanılan alet: (1) Numune yerleştirilen kutu, (2) örnek, (3) supap, (4) kablo, (5) hava filtresi, (6) ışıldama hücresi, (7) radyasyon monitörü (Sakoda vd, 2008).

1.4.1 Radyum Eşdeğer Aktivitesi (Ra_{eq})

^{226}Ra, ^{232}Th ve ^{40}K'ın numunelerdeki dağılımı birbirlerinden farklıdır. Farklı miktarlardaki bu spesifik aktivitelerinden radyasyona maruz kalmayı standartlaştırmak için radyum eşdeğer aktivitesi (Ra_{eq}) aşağıdaki ifade ile tanımlanmaktadır (Beretka ve Mathew, 1995).

$$Ra_{eq}(Bq/kg) : C_{Ra} + 1.43\ C_{Th} + 0.07\ C_K \qquad (1)$$

Burada C_{Ra} , C_{Th} ve C_K sırasıyla ^{226}Ra, ^{232}Th ve ^{40}K'ın Bq/kg biriminde spesifik aktiviteleridir. Örneklerin ölçülen Ra_{eq} değerleri 370 Bq/kg'den küçük olmalıdır (Petropouios vd., 2002).

1.4.2 Dış Tehlike İndeksi (H_{ex})

Beretka ve Mathew, (1995) tarafından tanımlanan dış tehlike indeksi;

$$H_{ex} = C_{Ra}/370 + C_{Th}/259 + C_K/4810 \ \leq 1 \qquad (2)$$

ifadesiyle verilmektedir. Burada C_{Ra} , C_{Th} ve C_K sırasıyla ^{226}Ra, ^{232}Th ve ^{40}K'ın Bq/kg biriminde spesifik aktiviteleridir. Radyasyon zararının önemsiz olabilmesi için H_{ex}'in değeri 1 den küçük olmalıdır.

1.4.3 İç Tehlike İndeksi (H_{in})

Beretka ve Mathew, (1995) tarafından tanımlanan iç tehlike indeksi;

$$H_{in} = C_{Ra}/185 + C_{Th}/259 + C_K/4810 \qquad (3)$$

ifadesiyle verilmektedir. Burada C_{Ra} , C_{Th} ve C_K sırasıyla ^{226}Ra, ^{232}Th ve ^{40}K'ın Bq/kg biriminde spesifik aktiviteleridir. Radyasyon zararının önemsiz olabilmesi için H_{in}'in değeri 1'den küçük olmalıdır. Çizelge 3'de Harşit granitlerinin radyoaktivite deney sonuçları verilmiştir.

Çizelge 3. Harşit granitlerinin ^{226}Ra, ^{232}Th, ^{40}K, $Ra_{eq,}$ H_{ex} ve H_{in} aktivite konsantrasyonları

Deney no	^{226}Ra, Bq/kg.	^{232}Th,Bq/kg	^{40}K, Bq/kg.	Ra_{eq}, Bq/kg	H_{ex}	H_{in}
1	54±4	57±5	770±30	194	0,52	0,67
2	52±3	80±7	774±23	225	0,61	0,75
3	39±3	63±5	671±20	182	0,49	0,60
4	48±4	47±4	770±22	175	0,47	0,60
5	25±2	59±5	617±19	157	0,42	0,49
6	50±3	50±5	747±29	179	0,48	0,62
Ortalama	**45 ±3**	**59 ± 5**	**725 ± 24**	**185**	**0,50**	**0,62**

Çizelge 3 incelendiğinde alınan granit örnekleri üzerinde Gama Spektrometrik Analizi sonucu saptanan Ra (eq) aktivitesi değerinin ortalaması 185 Bq/kg, olarak bulunmuştur. Örneklerin ölçülen Ra_{eq} değerleri bina materyalleri için 370 Bq/kg'den küçük olmalıdır (Petropouios vd. 2002). Bulunan sonuçlar bina materyalleri için öngörülen 370 Bq/kg değerinin altındadır.

Beretka ve Mathew, (1995) tarafından tanımlanan, bina materyallerinin radyasyon zararında önemsiz olabilmesi için H_{ex} ve H_{in}'in değeri 1 den küçük olmalıdır. Çizelge 3'de verilen $H_{ex,}$= 0,5 ve H_{in} =0,62 değerlerini göz önünde bulundurduğumuzda Harşit Granitlerinin bulunan H_{ex} ve H_{in} değerleri 1'den küçük olduğu bulunmuştur.

Çizelge 4'de dünyanın değişik yerlerinde 17 değişik granitin ^{226}Ra, ^{232}Th ve ^{40}K ve $H_{ex,}$ değerleri verilmiştir. Harşit granitin verilen granitlerin radyoaktivite değerleriyle karşılaştırıldığında ^{40}K ve H_{ex} 'in değerleri Zimbabwe ve Afrika siyahı dışındaki bütün granitlerden düşük olduğu ^{226}Ra, ^{232}Th'nın değerlerinin ortalama bir değerde olduğu görülmüştür. Çizelge 5'de Dünyanın farklı yerlerindeki doğaltaştarın radyoaktivite seviyesine baktığımızda özellikle Japon granitiyle Mısır çölünün kuzey ve güneyindeki granitlerin yüksek seviyede ^{226}Ra, ^{232}Th ve ^{40}K içerdiği gözlenmiştir.

Çizelge 4. ^{226}Ra, ^{232}Th and ^{40}K ($Bq\ kg^{-1}$) Pavlidou, vd, 2006).

Granite ismi	**^{226}Ra, Bq/kg.**	**^{232}Th, Bq/kg.**	**^{40}K, Bq/kg.**	**$H_{ex,}$ mSv**
Harşit graniti	45 ±3	59 ± 5	725 ± 24	0,50
Savatierra	118± 2	77±2	1320 ± 33	1.4
Rosa porrino	59±1	109 ±2	1420 ± 36	1.4
Blanco real	117 ±1	95 ±1	1233 ± 30	1.4
Topazio	29 ±1	44 ±1	1327 ± 33	0.8
Yellow cecilia	19±1	30 ±1	1020 ± 26	0.6
Blanco crystal	163±2	91 ±2	1190 ± 30	1.6
Napoleon	11±2	46 ±1	1200 ± 30	0.7
Balmoral	170±1	354 ±3	1592 ± 30	3.2
Afrika kırmızı	80± 1	121 ±1	1421 ± 32	1.5
Multicolour	11±1	84 ±2	926 ± 23	0.8
Baltic Brown	60±1	57 ±1	1350 ± 34	1.0
Gris perla	70 ±1	43 ±1	1340 ± 34	1.0
Emerald	55 ±1	63 ±1	1053 ± 24	1.0
Marina pearl	35 ±1	37 ± 1	894 ± 22	0.7
Zimbabwe	20 ±1	32 ± 1	332 ± 14	0.4
Afrika siyahı	1.6 ±0.3	< MDA	49 ± 4	0.0

Çizelge 5. Doğal taş radyoaktivite seviyesi (Sakoda vd, 2008, Krsti´c vd, 2007, Arafa ve Environ, 2004, Türkmen, 2003, Shershaby, 2002).

Granit-yer adı	**Ülke**	**^{226}Ra ($Bq\ kg^{-1}$)**	**^{232}Th ($Bq\ kg^{-1}$)**	**^{40}K($Bq\ kg^{-1}$)**
Harşit graniti	Türkiye	45 ±3	59 ± 5	725 ± 24
Misasa	Japon granit	895 ± 50	21 ± 2	779 ± 14
Badgastein		7064 ± 325	64 ± 15	1169 ±18
Gabal El Majal	Mısır Çölünün Doğu ve Kuzey-Doğu kısmı	198	30	681
El Misikat		1184	45	672
El Aradiya		125	24	480
Kuzey Homret Waggat		489	109	1590
Güney Homret Waggat		706	147	2144
Gable Gattar II		6018 ±181	113 ± 3	1140 ± 34
Granit	Hong-Kong	180 ±31	122 ±5	1248 ±15
Mermer	Ürdün	20 ±1,9	114 ±19	850 ±21
Mermer	Suriye	146 ± 18	7 ±2	41 ±4
Granit	Yunanistan	42 ±2	42 ± 2	630 ±28

Çizelge 6'da Türkiye'nin 7 ayrı bölgesinde alınan mermer ve granit numunelerindeki ^{226}Ra, ^{232}Th ve ^{40}K aktiflik konsantrasyonları verilmiştir. Çizelge 6'ya baktığımızda ^{226}Ra 20±8 (Bq/kg) ve ^{232}Th 11±4 (Bq/kg) değerleriyle en yüksek bölgemiz Karadeniz İç Anadolu Bölgemizden sonraki en yüksek ikinci bölgemizdir. Çizelge 6'daki Karadeniz Bölgesinin radyoaktivite seviyesi yüksek olmasına rağmen Çizelge 4 ve Çizelge 5'deki dünyanın farlı yerlerindeki granit ve mermerin radyoaktivite değerleriyle karşılaştırdığımızda bu değerlerin daha düşük olduğu görebiliyoruz.

Çizelge 6.Türkiye'deki Mermer ve granit numunelerindeki ^{226}Ra, ^{232}Th ve ^{40}K aktiflik konsantrasyonları (Damla, 2009).

Bölgeler	^{226}Ra (Bq/kg)	^{232}Th (Bq/kg)	^{40}K (Bq/kg)
Marmara	20±8	11±4	105±46
Ege	16±4	10±4	82±26
Akdeniz	20±7	9±3	102±41
Karadeniz	33±20	25±33	201±109
İç Anadolu	26±10	19±21	273±266
Doğu Anadolu	22±7	10±4	110±39
Güney Doğu Anadolu	21±7	12±4	68±46

2 SONUÇLAR VE ÖNERİLER

Mineralojik ve petrografik özelliklerine göre

- Yapılan parlak kesit sonuçlarında Harşit Granitinin içinde, Pirit (FeS_2), Hematit (Fe_2O_3) ve Manyetit (Fe_3O_4) olduğu gözlendi. Pirit, manyetit ve hematit minerallerinin bileşiminde yer alan (Fe) demirin bozuşması sonucunda ve çok fazla suyla temas etmesi sonucunda döşemede kullanıldığında granit yüzeylerinin paslanma tehlikesi olabileceği düşünüldü. Buna binaen fazla suyla temas eden yerlerde kullanılmaması daha iyi olur.
- İnce kesit sonucunda kayacın içinde plajoklaz, kuvars, hornblend, epitot ve biyotitlerin olduğu gözlendi. Kayaç içindeki mineraller de mikro çatlak oranı son derece düşüktür. Ayrışmanın çok az olduğu ve klorit dışında önemli bir ayrışma minerali bulunmamaktadır.
- Radyoaktivite seviyesine sonuçlarında
- Harşit granitlerinde saptanan Ra (eq) aktivitesi değerinin ortalaması 185 Bq/kg olup, bina materyalleri için öngörülen 370 Bq/kg değerinin altındadır.
- Beretka ve Mathew (1995) tarafından tanımlanan iç ve dış tehlike indeksinde, radyasyon zararının önemsiz olabilmesi için H_{ex} ve H_{in}'in değeri 1 den küçük olmalıdır. Sonuçlar göz önünde bulundurduğumuzda Harşit Granitlerinin bulunan H_{ex} ve H_{in} sırasıyla 0,5 ve 0,62 olarak bulunmuş ve bu değerlerin 1'den küçük olduğu görülmüştür.
- Bulunan bu değerler Harşit Granitlerinin bina malzemesinde iç cephede ve dış cephede yer döşemesinde duvar kaplamasında, basamaklarda, sütunlarda, şöminelerde, dış cephe kaplamasında yaygın olarak kullanılabileceği aynı zamanda kayaç sağlam olduğunda ve ayrışma yok denecek kadar az olduğunda agrega malzemesi olarak ta kullanılabilir.

TEŞEKKÜR

Bu çalışma Karadeniz Teknik Üniversitesi Bilimsel Araştırma Fonunca desteklenmiştir. Proje No: 2007.112.008.1.

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Urfa (Harran) Bazda Ancient Underground Marble Quarying

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ABSTRACT Archeological reserches have shown that natural stones have been produced and used in meny residental areas by old civilizations that lived in Anatolia. One of the oldest known stone obeliks is at Göbekli tepe-Urfa B.C. 10.000-12.000.

In Anatolia there were many ancient (B.C.) open marble quarries. But the first known underground marble quarry is İzmir – Belevi in the literature. The extraction method is room and pillar mining.

Today the most famous marble quaries were opened near or around old quarries especially at western Anatolia and its inner parts.

The second underground marble quarry is Urfa-Bazda which is called a cave by living people in its environs.

Th e type of rock is limestone. Room and pillar mining method was used to produce marble blocks. The height of pillars change between 12-18 meters.

According to archeological and historical studies, produced marble blocks were used to built Harran and Şuayp city B.C. 1.000-1.200.

In this paper , some parameters related to underground marble mining method and applied cutting technology used at Bazda underground marble quarry are given.

1 INTRODUCTION

Ancient Harran city is located at Holly Land Harran plain and on the Silk Road caravan. It is 44 km. far from the land of prophets Southeastern of Şanlıurfa City. According to the legend, when Adam, his wife (Adam & Eve) was expelled from paradise, the first place they arrived at on the earth was the sacred Harran plateau. According to Muslims, Christians and Jews beliefs Halil Ibrahim (Abraham), Eyyüp (job), Şuayp prophets were lived in this region. The holly Handkerchief of Isa (Jesus) was in Harran city and, the first officially Christianity religious was accepted by Harran Osreane Kingdom in history (Erdem, 1998). Natural stone is the main resource ancient architectural building materials, but also is important ancient stone quarries. In our country many local studies have pointed out aspects of extraction and production stone materials in ancient times (Kulaksız, 2008.)

Natural stones are employed not only in monuments and works of arts, but also ordinary buildings which are part of our historical, architectural and cultural heritage. In Anatolia (Türkiye) ornamental stone first were used at Göbekli Tepe (B.C. 12000-10000) later Hittites Empire Hattutaş, in Western Anatolia (Lyda, Karia, Lycia) B.C. 900 – A.C. 100 and especially Roman Empire.

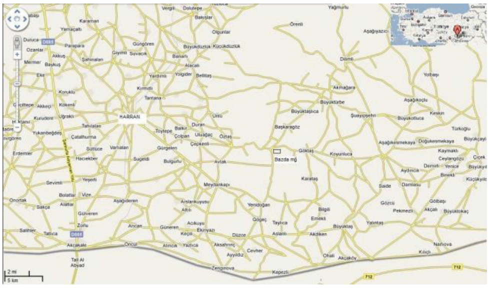


Figure 1. Location Map of Harran and Bazda Underground Ancient Mine (Bazda Caves).

1.1 Geology of Harran

Harran plain is covered with loose materials that contain, silt, clay, sand and gravel and talus of and slopes wastes. The structure Harran possibly graben structure.

The tabular materials overlay over Miocene-Eosene aged Midyat- Formation. The Eocene limestones extend over a large area in Southeast Anatolia. Their most characteristic development is represented of Midyat Plateau from the name Midyat Formation. (Tolun, N 1962, Sütçü, Y.F. 2008)

The upper part onsist of slightly fossiliferous, chalky, medium-soft limestone. Eocene limestone are more chalky of Urfa region than the other areas. Ancient Bazda Underground mine, dense cream-beigo colored massive limestone that contain also chalky limestone zone. So that Bazda Eocene limestone marble are separated into two zones; Cream, hard , fossiliferous massive sometimes thick badded limestones and chalky limestones (Figure 2).

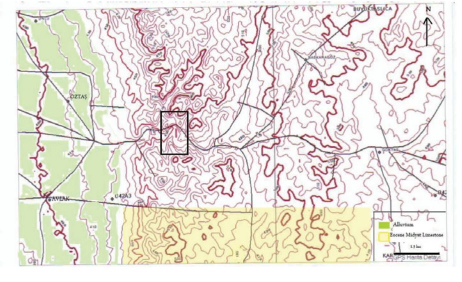


Figure 2. Topographic and Geologic Map of Ancient Marble Mine of Bazda

1.2 History of Harran and Ancient Bazda Underground Marble Mine

Ancient Harran city was founded by Sumerians before the Christ 2000 years ago and prosperous until 16 th century as a Vital Silk Road Caravan and Cultural city for marchanet and travelers. The holly book Tevrat(Torah) and Bible and some Islamic historian wrote that it was established the prophet Hz. Ibrahim brother Aran (Harran) Mardin – Urfa – Diyarbakır region ancient peoples believed the moon god Sin of Harran. The moon god Sin was one of the most ancient gods of the Mesopotamia. Mitanni Kingdom, Aramaeans, Assyrians and many Arami states, they all believed in the moon god Sin. Harran and Sin Temple were destroyed by Medes. Later Babylon King Banonid and Macedonian King Alexander reconstructed temple and city.

Harran Culture was mixed with Hellenistic Culture. Later Harran city were occupied Arabic tribes and other immigrant people continued to believe, moon god, (Sin), sun god, (Shamas) during the spread Christianity and then Islam in the region.

Harran was also capital the Emevi Kingdom (Umayyad Caliphate). The Sin Temple was destroyed by the Fatimids at A.D. 1032.

As a result we can say that Ancient Sumer, Egypt, Babylonian, Assyrian, Hittite, Mitanni, Macedonian, Selecuids, Abgar, Roman, Selçuks, Ottoman Age kingdom were ended but Harran name was not changed. Period Harran continued as harran. [Karlış. (1998), Çelik, (2000) Durukan, (1999)].

Bazda underground marble mining was started with the foundation of Harran City and Şuaypşehri (The city of Şuayp prophet) in second millennium. There isn't any rock outcrops in Harran plain. Bazda marble mine region is 15 km far from Harran city and 20 km far from ancient Şuayp şehri.

The underground marble mining operation was continued up to Anatolian Selçuks Kingdom (A.C. 1300). One of the main advantages of this area Chalky limestone was easily cut by chisel & miner picks by equipments and man forces.

2 BAZDA UNDERGROUND MINING METHODS AND EXTRACTION TECHNOLOGY

Harran stone carvers (masters) were also good marble miners. Some of the questions about them are “How to work underground marble mine and plan, what kind of equipments were used, How to carry marble blocks to Harran, what was geological knowledge about stones?

Ancient marble blocks production and mine planning generally choice of underground (tunnels) entrance places at outcrops, opening (gallery/tunnel) preparation, steps were observed. According to situation, openings had were replaced hillside, so overburden excavations were not so much. Entrance (opening) geometry is nearly square with changing dimensions (4x4 – 4x6 m). All the openings were excavated in massive, chalky limestone (Midyat Formation).

Figure 3. Entrance of marble mine and massive limestone.

These surfaces so adjusted to that, cuts in marble mine any discontinuity were not followed. Room and pillar mining system were applied. Irregular pillars were common and sometimes sill pillar of tunnel type rooms are exit. Pillars were randomly spaced and their design followed by extraction variously pillar conditions for the room and pillar operations. The size of pillars and openings (rooms) vary 3 x 3 m. widths and up to 20 meter heights (Figure 4).

Figure 4. Regular pillars of Bazda Mine.

After opening the rooms, some pillars partially had been extracted from bottom or reverse top (Figure 5).

Figure 5. Pillars recovering.

The most common marble block extraction methods descending slices, rarely ascending slices. This means that a progressive removal from top to bottom (Figure 5) of rock mass, also laterally from the entrance. Extracting of marble blocks low steps with virtual slices (Figure 6).

Marble block production regards a single slice there is only one bank (step), if more than one slice several different levels are exist. The most configurations are step with risers. One question “Are there any relationship between block size, step or level height?”

Figure 6. Marble block production from top to bottom.

In one of the Bazda underground marble mine there is an aeration (natural ventilation) system. Tunnel is connected with a rise (shaft) up to surface. (Figure 7).

Figure 7. Natural ventilation of Bazda Mine.

Besides this there is also mixed type (underground-open cast) marble mine (Figure 8).

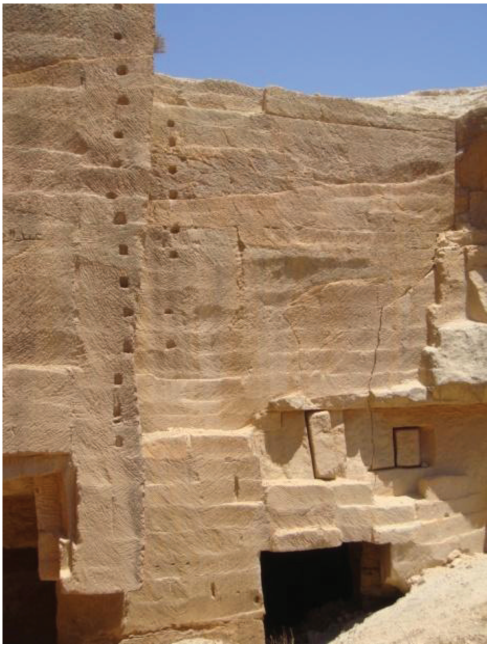


Figure 8. Mixed types marble mine.

In our studies of site investigation of Bazda underground marble, to cutting the blocks from massive limestone, are hand digging equipments which are miner pick, crowbar, chisel bar and cold chisel hammer, sledge hammer, different type of axes. The most interesting thing is wire cutting tracks at the top of room (Figure 9) of underground mine.

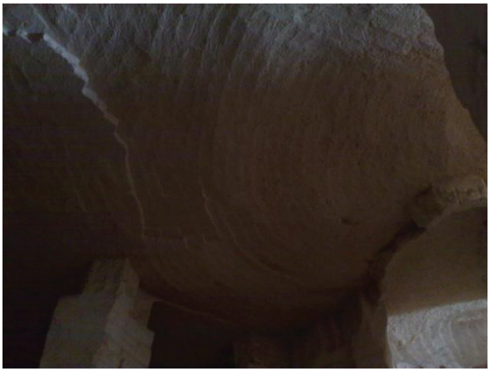


Figure 9. Probable wire-cutting tracks at the roof of mine.

As seen in Figure 10, they were used cushion to prevent while marble blocks cutting down.

Figure 10. Cushion.

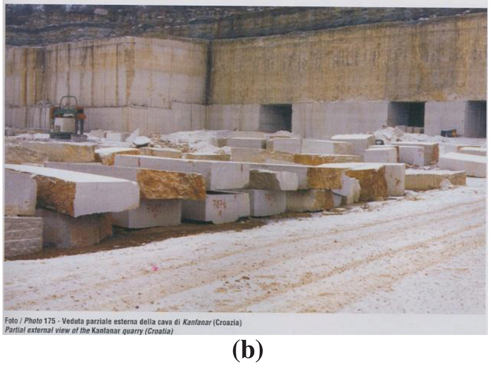
3 COMPARISON OF ANCIENT AND CONTEMPORARY UNDERGROUND MARBLE METHOD & TECHNOLOGY

The main monuments and buildings of Harran old city were made with local Bazda light beige. Eocene carbonate rocks that are mostly limestone.

Ancient Harran city is cultural heritage of civilizations, also this ancient stone underground mine too. We observed same block extraction and mining methods similarities Bazda underground marble mine and contemporary some marble mine in the world.

a. Applied room and pillar mining method at Bazda, today also are used in Italy, Portugal, Brazil, Croatia, Spain etc. (Fig11, Entrance).
b. Low step, multiple benches, several ditches left-right (laterally), up and down (vertically) (Fig.12).
c. Ventilation system and tunnel connection
d. Configurations of block cutting depend on where they were used.
e. Opening the hollow system similar today systems that are used.
f. Ripping of pillar is used in modern mining system.
g. We accept that wire cutting technology a new and started in 1970. But, as seen from cutting tracks of wire is similar to modern wire cutting technology.

(a)

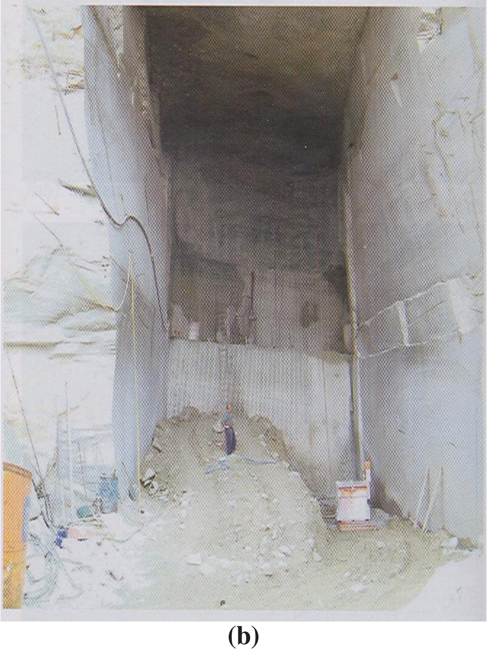


(b)

Figure 11. Underground marble mine entrance at Bazda (a) and Italy (b).

Roman Emperor period, there were many open-underground marble mine in Anatolia. Similar marble block cutting observations were found by İleri (1988) at Ephesus ancient city (Turkey) andKulaksız (2009) at ancient Sagalassos city (Turkey). Therefore marble block cutting with wire technology is not new as it was dated in 1970.

(a)



(b)

Figure 12. Different benches of room and pillar mine at Bazda (a) and in Italy (b).

4 RESULTS

* Bazda caverns (caves) were not real karstic openings. They are men made underground openings that are ancient marble quarries. The numbers of mine openings are over 24.
* Marble mining method is randomly room and pillar, low step methods. Sometimes regular room and pillar and tunnel type used.
* At the same places open pit and underground mining method are applied together.
* Observations were done in eight different entrances.
* Archeologically, this old marble mine must be researched and will be open to tourism.

These underground mines must be investigated from the rock mechanics and old underground mine planning must be prepared.

ACKNOWLEDGEMENT

Authors thanks to assistant Emre Yılmazkaya, Fırat Atalay and Özcan Yıldırım Gülsoy, Chairman of Mining Engineering Department Hacettepe University.

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Müzelerin Eğitime Katkısı: "Kütahya Jeoloji Müzesi"
Educational Contribution of Museums: "The Geological Museum of Kütahya"

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ÖZET Okullarda konular çoğu zaman kavranamayacak kadar soyut işlenir ve öğrenciler konuları algılamada zorluk çekerler. Kayaçların, minerallerin nasıl oluştuğunu ve maden çalışanının zor çalışma koşullarını dinlemek bir tür bilgi oluşturur; ama müzelerde oluşturulmuş bir maden ocağındaki bire bir galeri örneğine girip karanlığı görmek, oradaki sessizliği dinlemek, kompresör sesinin, bu sessizliği nasıl bozduğunu işitmek, mineral örneklerine dokunmak, somut olarak fosilleri görmek öğrencilerin kavrayışını daha da derinleştirecek ve daha kolay empati yapmalarını sağlayacaktır. Bu bağlamda müzeler, öğrencilerin öğrenmeye olan merakını ve isteğini geliştirir. Çok ilginç buldukları sergi parçalarını araştırma ve inceleme fırsatını yakalarlar. Katılım artar. Çocuklar yaşadığı çevreyi ve dünyayı bu şekilde araştırırken oldukça eğlenir ve kendileri etkin oldukları için de merakları kamçılanır. Bu çalışmanın amacı; ilköğretim öğrencilerini dağların, vadilerin, kayaçların, minerallerin, kristallerin, fosillerin gizemli dünyasıyla tanıştırıp, doğaya karşı ilgilerini artırarak, jeoloji-jeofizik-maden mühendislikleri ile ilköğretim öğrencileri arasında duygusal bir bağın kurulmasını sağlamada "Yer Bilimleri Müzeleri"nin önemini vurgulamaktır. Ülkemizde mesleğine aşık yerbilimcilerin yetişebilmesi için, öğrencilerin ilgilerini daha ilköğretim çağındayken bu alana çekmek gerekir. Bunun için "Yer Bilimleri Müzelerinin" sayılarının artması ve üniversitelerimizin jeoloji-jeofizik-maden bölümlerinde seçmeli dahi olsa "müze" konusunda derslerin verilmesi faydalı olacaktır.

ABSTRACT Subjects and content in schools are mostly taught too abstract to comprehend so students have difficulty to understand. Listening how to form rocks-mineral and heavy working conditions of mineworkers creates a sort of knowledge but; going into a textbook example of the drift mine formed in museums will develop students' insight and enable them to show empathy easily, as well as; to see the darkness- to listen to silence there- to hear the changed tone of the compressor-sound in this silence- to touch the mineral samples- to watch the concretized fossils. In connection with this; the students improve their motivation *and* curiosity for learning. They take the opportunity of research and investigation making exhibition pieces interesting. Participants in number increase considerably by all means. Children have a good time to repletion while exploring the earth and the environment where they live, and their curiosity encourage themselves by virtue of being active. The aim of this study; to introduce primary school students to a mystical world of valleys, mountains, rocks, minerals, crystals and fossils and to establish an emotional bond between geology - geophysics - mining engineering and primary school students by attracting their attention for the nature, after all; to emphasize the significance of geological museum. For training more geoscientists being very fond of their profession in our country, it must arouse students'

interests as they're in primary schools; therefore in case of increasing the quantity of "geological museums" and optional subjects as well, it will be beneficial in being given lectures relating to "museum" in the departments of geology – geophysics - mining in our universities.

1 GİRİŞ

Bireylerin, iyi bir kariyer ve başarılı bir geleceğe sahip olmaları için salt üniversite eğitimi yeterli değildir. Bireyler ancak gönül verdikleri ve yapmaktan haz duydukları işte başarılı ve verimli olabilirler. Ancak ülkemizde gençlerimiz mesleklerini seçerken, ilgi duydukları ve kabiliyetleri olan alanlara yönlendirilmek yerine, toplumda prestiji olan ve iyi para getirdiğine inanılan bölümlere odaklandırılmaktadırlar. Dolayısıyla çocuklarımız belki yapmaktan hiçbir zaman mutlu olamayacakları ve içselleştiremeyecekleri meslekleri tercih etmektedirler. ÖSS'de dereceye giren çocuklarımızın hep belirli bölümleri seçmeleri bunun en bariz örneğidir.Öyle ya, birgün ÖSS'de ilk ona, ilk yüze giren öğrencilerden birkaç tanesi, ilk tercihlerini maden, jeoloji ya da jeofizik v.b. bölümlerinden yana kullansalar ailesi, çevresi, onu yetiştiren okul ve dershane öğretmenleri ne derler? Neler yazılır, neler çizilir? Peki ÖSS'de derece yapmış ve tercihini bu bölümlerden yana kullanmış bir gencimizin, okulu başarıyla bitirdikten sonra işsiz kalması ya da çok iyi şartlarda iş bulamaması söz konusu olabilir mi? Öyleyse, neden sıralamada başı çeken çocuklarımız yerbilimlerini tercih etmiyorlar? Yerbilimleri, onların ilgi alanlarına girmeyecek kadar basit ve işlevsiz bölümler midir? Bu ülkenin kalkınmak ve daha ileri gitmek için yeraltı ve yerüstü zenginliklerini en iyi şekilde değerlendirebilecek, yurdumuzun zeki öğrencilerine ihtiyacı yok mudur? Ülke kalkınmasında, o ülkenin yeraltı ve yerüstü zenginliklerini değerlendirmek kadar, hatta ondan da önemlisi, insan kaynaklarından en güzel şekilde yararlanmak ve sahip olduğu beyin gücünü kabiliyeti ve ilgisi olan alana kanalize edebilmektir. Bunun en güzel yolu da, daha ilköğretim çağından başlamak üzere kişinin kendisini tanımasına, yetenek ve ilgi alanlarının ortaya çıkmasına olanak tanıyacak düzenlemelerin yapılmasıdır. Eğitim alanında yapılacak böylesi açılımlar sayesinde bireylerin eğitsel, düşünsel ve sosyal koşulları iyileşecek, böylelikle sağlıklı kişisel gelişim ve kendini gerçekleştirme süreçleri daha erken başlayacaktır. Bu sayede de sadece bireylerin değil, ülkenin gelişimine de katkı sağlanacaktır. Bütün bunların gerçekleştirebileceği en iyi ortamlar müzelerdir. Etimolojik olarak "müze" sözcüğünün kökeninde Yunan Mitolojisi yer alır. "Mousa" ilham perisi sözcüğünden türemiş, "mouseion" ilham perilerinin yeri ya da bilimler tapınağı anlamında kullanılmıştır. Eski Atina'daki Mousa'lar tapınağına atfen İskenderiye'de II. Ptolemaios Philadelphos'un (Mö 285-246) kurduğu kütüphane ve sanat koleksiyonuna verilen addır (Nişanyan, 2002). Tarihte koleksiyonculuktan müzeciliğe geçiş bilimsel etkinliklerin doğal sonucu olarak ortaya çıkmıştır. Yayınlara göre; "koleksiyonların gelişmeleri ve yeniden değerlendirilmeleri, sınıflandırılmaları, belgelendirilmeleri ve korunmalarıyla, bireysel uğraş, bilimsel ve kuramsal bir çalışmaya dönüşmüş; toplumsal ve siyasal bilinçlenmelerin katkılarıyla kamulaştırılan koleksiyonlarla müzeler oluşturulmuştur." (Şener, 2005).

2 MÜZE VE EĞİTİM

Müzeler; "Toplumun bilimsel ve kültürel geçmişini yansıtan ve geleceğini biçimleyecek öğeleri araştıran, toplayan, koruyan, sergileyen, belgeleyen, yaşatan ve yönlendiren yaygın eğitim kurumlarıdır." (Atagök, 1990).

"Doğa bilimlerinden güzel sanatlara kadar çok geniş bir alana yayılan koleksiyonculuk çabaları, bugün artık tasnif etme, kataloglama, yerleştirme, tanıtma ve eğitici işlevlere açık tutma gibi, yalnız müzeciliği ilgilendiren uzmanlık düzeyindeki bilgilere dönüşmüştür. Bu bilgiler, genellikle geçen yüzyılda hızlanan endüstri devriminin ortaya çıkardığı yeni uzmanlık alanlarıyla da yakından ilgilidir. Böylece müzecilik birtakım eşsiz parçaları toplamaktan ibaret olan geleneksel dar ölçülerin dışına taşmış , yeni ve çağdaş bir müzecilik kavramı, bu yüzyılın başlarından itibaren ortaya çıkmıştır"(Özsezgin, 1985). Koleksiyonlar insanların "biriktirme" merakından doğmuştur. Avrupa'da koleksiyonların ziyarete açılmasıyla müzeler bir kurum olarak ortaya çıkmış, endüstri devriminin etkileriyle de gelişme göstermiştir.

20.yy'ın ikinci yarısından sonra müze eğitiminde bilimsel yaklaşımlar önem kazanmıştır. Bunun en büyük etkenlerinden biri UNESCO ve ICOM gibi uluslararası organizasyonların kurulmasıdır (Tezcan Akmehmet ve Ödekan, 2006). Geleneksel müzecilik daha çok arama, toplama, koruma ve sergileme anlayışıyla sınırlıyken, buna karşılık çağdaş müzecilik iletişim kurma ve eğitme işlevlerini vurgulamaktadır. Böylece edilgen bir müzecilik anlayışından etkin, dinamik ve katılımcı bir müzecilik anlayışına geçilmiştir (Onur, 2000). Müzecilikte asıl amaç, kültür ve bilimin toplumun tüm kesimine aktarılması olarak gelişmiş, bu nedenle eğitim, toplama, koruma, inceleme, değerlendirme ve sergilemenin önünde yönlendirici etkinlik olarak biçimlenmiştir. Günümüzde müzeler birer yaygın eğitim kurumu olarak halkı eğitmeyi, kültür ve bilimi topluma aktarmayı hedefleyerek, iletişim ve halkla ilişkileri başlıca yöntemler olarak kullanmaya yönelmiştir. Müze sadece bir bina ve koleksiyon değildir; müze toplar, fakat bir depo değildir; müze korur, fakat bir buzluk değildir; müze belgeleri oluşturur, fakat bir kütüphane değildir; müze eğitir, fakat bir okul değildir. Müzeler üstlendikleri görevleri nedeniyle, birer açık üniversite, herhangi bir ailenin tüm fertlerinin eğlenerek öğrenebileceği, öğrenmenin bir zevk olabileceği kültür merkezidir." (Atagök, 1990).

Müzelerde ve galerilerde eğitim yapmak; müzenin koleksiyonları ile müze ziyaretçisinin gereksinmeleri ve ilgileri arasında ilişki kurmaktır. Her grubun ve her bireyin gereksinmeleri ve ilgileri farklıdır (Greenhill, 1999). Vurgu nesnelerin üzerinden insanların üzerine kaymıştır.
Müzeler bir yandan konusu gereği topladığı, depoladığı, arşivlediği, koruduğu eserlerle kültür ortamı olma amacını gerçekleştirirken, diğer yandan sergileme vb. yolla eğitim amacını da gerçekleştirerek bu anlamda bir bütün olma özelliğine ulaşmaktadır.

Böylece bu amaçları doğrultusunda müzeler; bir şeyler öğretirken, kişinin yaratıcı güçlerini de geliştirerek kişiyi topluma hazırlamalı ve aynı zamanda müze faaliyetleri, boş zamanlarla ilgili faaliyetlerle birleştirilmelidir (Rebetez, 1969).

Tüm bunların yanında, amaçlarını gerçekleştirebilmek, görevlerini tam anlamıyla yerine getirebilmek için müzelerde çeşitli faaliyetler yürütülmektedir. UNESCO 'nun 1962'de "Müzelerin Eğitimdeki Rolü" hakkında düzenlediği bölge seminerinde; basit şekli ile sadece müze galerilerinde yapılan, fakat gelişmiş şekli ile müzenin diğer kısımlarında da yapılabilen rehber eşliğinde ziyaretler, tamamen müze içinde oluşan eğitim çalışmasıyla karıştırılmaması gereken konferanslar, kurslar ile yaratıcı sanat faaliyetleri, müze ve müzenin eğitim servisleri tarafından gezi ve seyahatler ki, müzenin türüne göre değişik şekillerde olan bu geziler eğlendirici ve aynı zamanda eğitici içerikte olabilir, okullara ve kültür kuruluşlarına ödünç eser verme gibi faaliyetler, müzelerde yürütülebilecek faaliyetler olarak sıralanmıştır (Rivier, 1962).

"Amerika Birleşik Devletleri'nde, Texas Üniversitesi'nde yapılan bir araştırmanın sonuçlarına göre, zaman faktörü sabit tutulduğunda insanlar okuduklarının % 10'unu, görüp işittiklerinin % 50'sini, sadece işittiklerinin % 20'sini, görüp işittikten sonra söylediklerinin % 80'ini, gördüklerinin % 30'unu, yapıp söylediklerinin % 90'ını hatırlamaktadır." Çağdaş eğitim anlayışı, ezberciliğe tamamen karşıdır. Bunun yerine çocuklarda hayal gücünün ve yaratıcılığın gelişmesini ön planda ele alır. Bu yeteneklere sahip bir kimse okul yaşamı boyunca bir takım bilgileri ezberleyen bir başkasından daha verimli olacaktır. Çünkü hayal gücü kuvvetli, yaratıcılığı gelişmiş bir birey öğrenmeye daha açıktır ve ezber her zaman unutulmaya mahkumdur. Bu nedenle yaratıcı gücü gelişmiş , estetik duyguya sahip bireyler yetiştirmek eğitimin başlıca amacı olmuş ve bu amacını gerçekleştirmek için yararlandığı çeşitli kurumlar arasına müzeleri de katmıştır. Müzeler estetik duygunun, yaratıcılığın, hayal gücünün gelişmesini sağlamada ideal kurumlardır.

"Eğitim ortamı; eğitsel etkinliklerin meydana geldiği, öğretme öğrenme süreçlerindeki iletişim ve etkileşimin oluştuğu personel, araç gereç,tesis, organizasyon vb… oluşturduğu çevredir. Eğitim bilimcilere göre öğrenme fiziksel , sosyal ve psikolojik yönlerden uygun ve hoş bir çevrede oluşabilir." (Alkan, 1979). Bu açıdan bakıldığında müzeler; kazandırılması düşünülen bilgilerin somut olarak görüldüğü bir yer olarak fiziksel; yaşamda yeri olan bir kurum olması ile sosyal; sınıf ortamı olmadığından çocukların kendilerini rahat hissedecekleri psikolojik bir çevredir. Fiziksel, sosyal ve psikolojik niteliklerin hepsini üzerinde toplamış bulunan müzeler, ziyaret ve alan gezileri şeklinde bir eğitim ortamı olarak nitelendirilebilinir.

3 OKUL - MÜZE İLİŞKİSİ

Müze eğitiminin okuldaki eğitimi zenginleştirici bir potansiyele sahip olduğu ortadadır. Eğitim zihinsel kavrayış yanında empatik bağlar kurmayı, merak etmeyi, eleştirel bakmayı, pratik beceriler kazanmayı da sağlayabilmelidir. Müze ziyaretlerini öğretimle bütünleştirmek, okulu müzeye götürmek kadar müzeleri dışa açmak, kısacası müze-okul işbirliğini gerçekleştirecek düşünceleri geliştirmek, etkinlikler yapmak gerekmektedir. Batı ülkelerinde pek çok müze çocuklara ve yetişkinlere programlı olarak bilim, kültür, sanat kursları düzenlemektedir. Bu yönüyle yaratıcılığı, düşgücünü, soru sormayı, yeni bilgiler üretmeyi, sentez yapmayı özendiren, geliştiren bir etkiye sahiptir. Kitapların soyut metinleri müzenin somut ve görsel yapısı ile birleştirilebilirse, öğretim daha ilginç olur ve öğrenciler müzeleri kaynak olarak kullanmada daha istekli olurlar. Eğer öğrencilerin çok ilginç buldukları sergi parçalarını araştırmalarına ve incelemelerine izin verilirse, büyük olasılıkla güdüleme artar. Dolayısıyla öğretmen eğitimciden çok, bir rehber olarak öğrencileri destekleme ve bilgiyi nerede ve nasıl bulacaklarını onlara gösterme işlevini görür. Ancak ülkemizde okul-müze arasındaki buluşma seyrek bir özelliğe sahiptir. Çoğu zaman müzelere ziyaretler, okul çalışmalarında sağlam bir temele dayanmayan geziler olma özelliğine sahiptir; öğretimin bütünleştirilmiş bir parçası olma yerine, okulu kırma gibi görülmektedir. Bu bağlamda öğretmenlerin müzelere bakış açısını ortaya koymak üzere yapılmış bir araştırmaya 4 devlet okulundan 50 öğretmen katılmıştır. Yapılan ankette öğretmenlerin müzelere bakışı sorulmuştur. Ankete katılanların çoğunluğu (%86'sı) müzeleri eğitim ortamı olarak gördüklerini belirtmişlerdir. Bu sonuç müze eğitimi açısından sevindiricidir. Fakat eğitim ortamı olarak müzelerden yararlanıp yararlanmadıkları sorusuna çoğunluk (%36) müzelerden yararlanmadıklarını belirtmişlerdir. Bu durum bize öğretmenlerin teorik olarak müze eğitimi kavramını bildiklerini, duyduklarını ama o ortamı eğitim ortamı olarak kullanmadıklarını ortaya çıkarmaktadır. Müzelerden yararlananların ise en çok hayat bilgisi (%65) ve sosyal bilgiler dersinde (%52) yararlandıkları ortaya konmuştur. Müzelerden yararlanmayan öğretmenlere neden yararlanmadıkları sorulduğunda, "öğrencileri müzelere götürmek için formalitelerin çok fazla olduğunu" belirtmişlerdir. Ülkemizdeki müzelerin eğitim etkinliklerini nasıl buldukları sorulduğunda ise çoğunluk (% 54) yetersiz bulduğunu belirtmiştir. Bu da okul-müze iletişiminin zayıflığından kaynaklanmaktadır. Okullarla müzeler işbirliği içinde olsa, müfredat programları müzedeki nesnelerle bağlantı kurularak yapılsa, öğrenciler açısından çok daha yararlı olacaktır. Öğretmenlere gezdiği müzelerde, eğitim açısından gördüğü en büyük eksiklik sorulmuştur. Bilgilendirmenin olmadığını vurgulamışlardır. Müzelerdeki bilgilendirme etiketleri daha dikkat çekici ve bilgi verici olmalıdır. Okul-müze iletişiminin geliştirilmesi için neler yapılmalıdır, sorusu yöneltilmiştir. Ankete katılanların çoğu (%50) "Müzelerin bilgilendirmeye ve reklama önem vermesi gerekir" demiştir. Müzeler kendilerini tanıtmalı, eğitim etkinlikleri ve seminerler düzenlemelidir. Tanıtıcı broşürler, levhalar bastırarak reklamlarını yapmalıdırlar. Klasik müzecilik anlayışından çağdaş müzecilik anlayışına bir an önce geçilmelidir. Milli Eğitim müfredat programında değişiklikler yapılmalıdır. Konular müzelerde işlenebilecek şekilde tasarlanmalıdır. Özellikle Resim – İş, Hayat bilgisi, Sosyal Bilgiler gibi derslerde müzelerden rahatlıkla yararlanılabilinir. Müzelerden yararlanmak öğretmenin de işini kolaylaştıracaktır. Bütün bunlar için gerekli alt yapılar hazırlanmalı, formaliteler en aza indirilmelidir. Öğretmenlerimizin de belirttiği gibi MEB – Kültür Bakanlığı iş birliği içinde olmalıdır. Müzelerde müze eğitimcileri çalışmalı, öğrencileri onlar bilgilendirmelidir (Gökmen, 2004).

Okul, bütün öğrencilerin öğrenmeye olan merakını ve isteğini geliştirmeye çalışmalıdır. Deneysel atölyenin temel ilkelerinden biri katılım kavramıdır ve öğrencilerin deneme ve yanılma yolu ile kendi bilgilerini oluşturma fikrine dayanır. Bunları müzelerde gerçekleştirmek mümkündür. Okullar, öğrencilerin yaratıcılık yeteneklerini ve değişik ifade araçlarını kullanma alanını geliştirmelidir. İlk anda okul, sözlü kültürün bir arenasıdır. Çocuklar kendilerini konuşarak ve yazarak ifade etmeyi öğrenirler. Öğretmenlerin öğrencilerin yetenekleri hususunda sabit fikirleri vardır. Az zeki olduklarını düşündükleri çocukların sınıfın dışında, müzede ya da doğada, problem çözmede ve gerçeklikle baş etmede ne kadar yaratıcı, yenilikçi, hayal gücü kuvvetli ve enerjik olduğunu gördüklerinde şaşırırlar. İlköğretim müfredatında yerbilimlerinin çalışma konularını içine alan bir çok konu başlıkları vardır. Örneğin 4.sınıf fen ve teknoloji dersinin 7.ünitesi "gezegenimiz dünya", 5.sınıfın sosyal bilgiler dersinin 3.ünitesi "bölgemizin yüzey şekillerini tanıyalım ve yurdumuzdaki doğal afetler", 6.sınıfın fen ve teknoloji dersinin 8.ünitesi "yer kabuğu nelerden oluşur", 8.sınıfın fen ve teknoloji dersinin 8.ünitesindeki "doğal süreçler, evren ve dünyamız nasıl oluştu", yer kabuğunu etkiliyen levha hareketleri," v.b. gibi ilköğretimin bütün dönemlerinde yerbilimleri ile ilgili bir çok konunun olması yerbilimleri müzelerini, diğer müzeler içersinde bir adım daha öne çıkarmaktadır. Öğrenciler sergileri inceleyerek çok şeyler öğrenirken, müzeye yeni nitelikler ekleyebilirler. Çok değişik dönemlerden kalma nesneleri görebilir, inşa edilmiş ortamları keşfedebilir, slayt gösterilerine katılabilir ve filmleri izleyebilir; meslekleri deneyebilir ve diğer etkinliklere katılabilir. Okul kitapları, yaşadığımız dünyayı soyut olarak anlatırken, yerbilimleri müzeleri çocuklarla dünya arasında duygusal bir bağ kurmasına yardımcı olabilir, aynı zamanda çocuklarda sağlam bir çevre bilinci oluşmasını sağlayabilir.

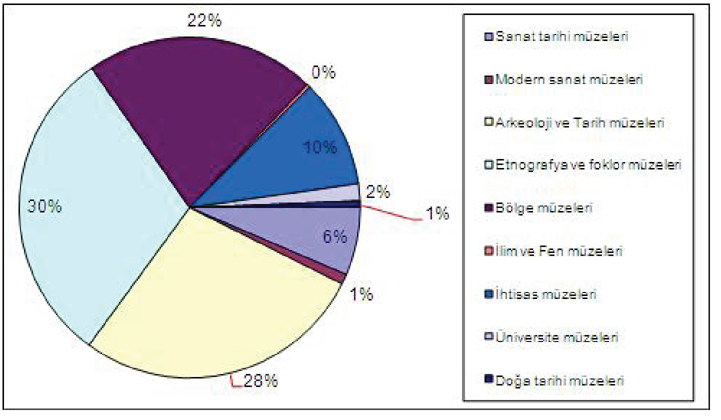
4 TÜRKİYE'DEKİ DOĞA TARİH MÜZELERİ

Türkiye'de bugün itibariyle 1481 adet koleksiyon, 127'si özel olmak üzere toplam 311 adet müze vardır. 17 vilayette hiç müze bulunmazken, müzelerin %43'ü beş şehirde toplanmıştır (İstanbul, Ankara, İzmir, Bursa ve Konya). Özel müzeler, Kültür ve Turizm Bakanlığı Kültür Varlıkları ve Müzeler Genel Müdürlüğü'nün, sadece denetimine bağlıdır. Milletlerarası Müzeler Komitesi (ICOM) tüm müzeleri, koleksiyon içeriğine göre; Sanat Tarihi, Modern Sanat, Arkeoloji ve Tarih, Etnografya ve Folklor, Tabii Tarih, Bölge, İlim ve Fen, İhtisas ve Üniversite Müzeleri olarak sınıflandırmıştır. Arkeoloji ve etnografya müzelerinin Türkiye'deki toplam müzelere oranı %58'ken, Doğa Tarihi müzelerinin oranı ise sadece %1'dir. Doğa Tarihi müzelerini oluşturmak diğer müzeleri oluşturmaya göre hem çok daha ucuz ve hem de çok daha kolaydır. Müzelerin gücü ziyaretçi sayısı ile yakından ilgilidir. "Doğa

Tarihi Müzeleri" çağdaş, gelişmiş ülke normlarına ve okul müfredatlarına göre tanzim edilse, öğrencilerin kendilerini evlerinde gibi hissedebilecekleri belli bir hareket alanı oluşturulsa, ziyaretçileri hiç eksik olmaz (Çizelge 1, Şekil 1).

Çizelge 1. Kültür Varlıkları ve Müzeler Genel Müdürlüğü'ne kayıtlı müze sayıları.

	Sanat Tarihi	Modern Sanat	Arkeoloji ve Tarih	Etnografya ve Foklor	Bölge	İlim ve Fen	İhtisas	Üniversite	Doğa Tarihi	Toplam
Müze Sayısı	20	3	86	94	68	1	32	5	2	311



Şekil 1. Kültür Varlıkları ve Müzeler Genel Müdürlüğü'ne Kayıtlı Müzelerin Dağılımları.

Müzelerin tarihi gelişimi içinde Tabiat Tarihi Müzelerinin önemli bir yeri vardır. Doğanın milyonlarca yıl öncesine dayanan karanlık yönleri, yapılan jeolojik, paleontolojik, paleoantropolojik ve prehistorik çalışma ve araştırmalarla aydınlanmaktadır. Bunun önemini kavramış olan Avrupa ülkelerindeki ilk Tabiat Tarihi Müzeleri'nin kuruluşu 16. yüzyıla kadar inmektedir. Böylesine değerli bulguları içeren bir Tabiat Tarihi Müzesi'nin 20. yüzyılda ülkemizde hala kurulmamış olması belirgin bir eksiklik olmuştur. İlk Tabiat Tarihi Müzesi, MTA Genel Müdürlüğü bünyesinde 1968 tarihinde kurulmuştur. 10.800m^2 kullanım alanına sahip olan müzede, dinozor Allosaurus, fillerin ilk temsilcilerinden Mastodon, Maraş Fili, güncel bir balina iskeleti, Türkiye florası ve faunasına, omurgalı-omurgasız fosiller, doğal mağara modeli, yaklaşık 2000 adet mineral ve kayaç örneği, süs taşları sergilenmektedir.

Türkiye'nin ilk akademik müzesi olma özelliğini taşıyan Ege Üniversitesi Tabiat Tarihi Müzesi aynı zamanda ülkemizin ikinci büyük Doğa Tarihi Müzesidir ve ilk kez 1973 yılında sahip olduğu objeleri sergilemeye başlamıştır. Ege Üniversitesi Tabiat Tarihi Müzesi, yaklaşık 2500 m^2'lik bir alana yerleşmiştir. Toplam 7000'e yakın objenin 4000 adet tanılı olanları, altı farklı galeride sergilenirken, diğerleri ise karşılaştırma materyali olarak merkez laboratuarındaki dolaplarda yer almaktadır.

5 KÜTAHYA JEOLOJİ MÜZESİ

2005 tarihinde kabul edilen 5393 sayılı "Belediyeler Kanunun" 14. maddesine göre yerel yönetimlerin kültür, sanat, turizm, sosyal hizmetler, meslek ve beceri kazandırma gibi eğitsel konularda görev ve sorumlulukları vardır. Bu bağlamda, Kütahya Belediyesi restorasyon uygulamasının tamamlanmasından sonra; doğa ve doğa tarihine ilişkin objelerin korunmasını, saklanmasını, onlarla ilgili bilimsel çalışmalar yapılabilmesini ve onların gelecek kuşaklara aktarılmasını; bilhassa ilköğretim çağındaki öğrencilerin eğitim ve öğretimine katkı sağlayarak, bu çalışmaları ulusal ve uluslararası kurumlarla paylaşıp araştırıcı ve yaratıcı doğa bilimcilerin yetişmesine katkı sağlamayı ve doğa tarihini çağdaş müzecilik anlayışı içinde toplumla bütünleştirmeyi misyon edinmiş bir kurum olarak, "Şengül Hamamı ve Yahya Efendi Konağı'na Kütahya Jeoloji Müzesi" şeklinde fonksiyon verilmesini kararlaştırılmıştır. Korunması gerekli Kültür varlığı olarak tescilli olan Şengül hamamının tarihi 17. yüzyıla kadar uzanmaktadır. 4,4 milyar yıllık dünyanın geçirdiği evrimi anlamayı, öğrenmeyi ve sorgulamayı; doğanın taşlar, mineraller, fosiller, güneş sistemi, çevre gibi tüm çeşitliliğini göstermeyi amaçlayan müzede, ziyarete gelenlere evrenin, gezegenlerin ve yerkürenin oluşumunu çeşitli konferanslar, film ve slayt gösterileri gibi kanıtlarla anlatarak, özellikle ilköğretim ve lise öğrencilerini çevremizdeki tüm doğa olayları hakkında bilgilendirmeyi, doğayı sevmeyi, korumayı ve onun bir parçası olduğumuz bilincini yerleştirmeyi amaçlıyor. Bu amacın gerçekleşmesine, ilköğretim 4, 5, 6, 7 ve 8. sınıf müfredatında yerbilimlerinin çalışma konularına ait bir çok konu başlığının olması da katkı sağlıyor. Müzedeki etkinliklere katılan öğrenciler, burada doğayla birlikte kendilerini de keşfederek ilgi alanlarını ortaya çıkarırken, gelecekte kendilerini adayabilecekleri, zeka ve becerileriyle katkıda bulunabilecekleri mesleği de belirleme imkanı buluyorlar. Belirledikleri meslek yerbilimleri ile alakalı bir bölüm olmasa bile, çocuklara kazandırılmış sağlam bir çevre bilinci ile hangi meslekte olurlarsa olsunlar, doğayı koruyan ve gelecek kuşaklara en az düzeyde bozulmuş bir çevre bırakmayı amaçlayan bireyleri topluma kazandırıyor. Bu amaçlarla oluşturulan "Kütahya Jeoloji Müzesi" 610m² iç, 530 m² dış kullanım alanıyla, Kültür ve Turizm Bakanlığı'nın 03.12.2008 tarihli ve 219618 sayılı oluru ile çalışmalarına başlamıştır. Müze, iki katlı, 8 odalı idari bina (Yahya Efendi konağı) ile, 9 bölmeden oluşan ve örneklerin teşhir edildiği sergi salonundan (Şengül hamamı) oluşmaktadır. Müzede yaklaşık 700 adet örnek sergilenmektedir. Sergilenen objelerin envanter çalışması henüz tamamlanmamıştır (Şekiller 2, 3, 4, 5, 6, 7, 8 ve 9).

Şekil 2. Kütahya Jeoloji Müzesi'nin görünümü.

Şekil 3. Kütahya Jeoloji Müzesi'nin uydu görüntüsü.

Şekil 4. Kütahya Jeoloji Müzesi'nin iç mekanı.

Şekil 5. Kütahya Jeoloji Müzesi'nde bir kömür galerisi.

Şekil 6. Kütahya Jeoloji Müzesi'nin seramik hammadde standı.

Şekil 7. Kütahya Jeoloji Müzesi'nde maket iş makineleri.

Şekil 8. Kütahya Jeoloji Müzesi'nde tanımlanmamış fosil örneği.

Şekil 9. Kütahya Jeoloji Müzesi'nde süs taşları standı.

6 SONUÇ VE ÖNERİLER

Türkiye'de "Doğa Tarihi Müzeleri" yaygınlaştırılıp, ilköğretim çağındaki çocuklarımızı yerbilimleri ile tanıştırmalı, ilgisi ve kabiliyeti olan çocukları bu bölümlere kazandırmalıdır. Bunu gerçekleştirebilecek elemanların istihdam edilebilmesi için de, yerbilimleri bölümlerinde müze ile ilgili derslerin seçmeli de olsa okutulması gerekmektedir.

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