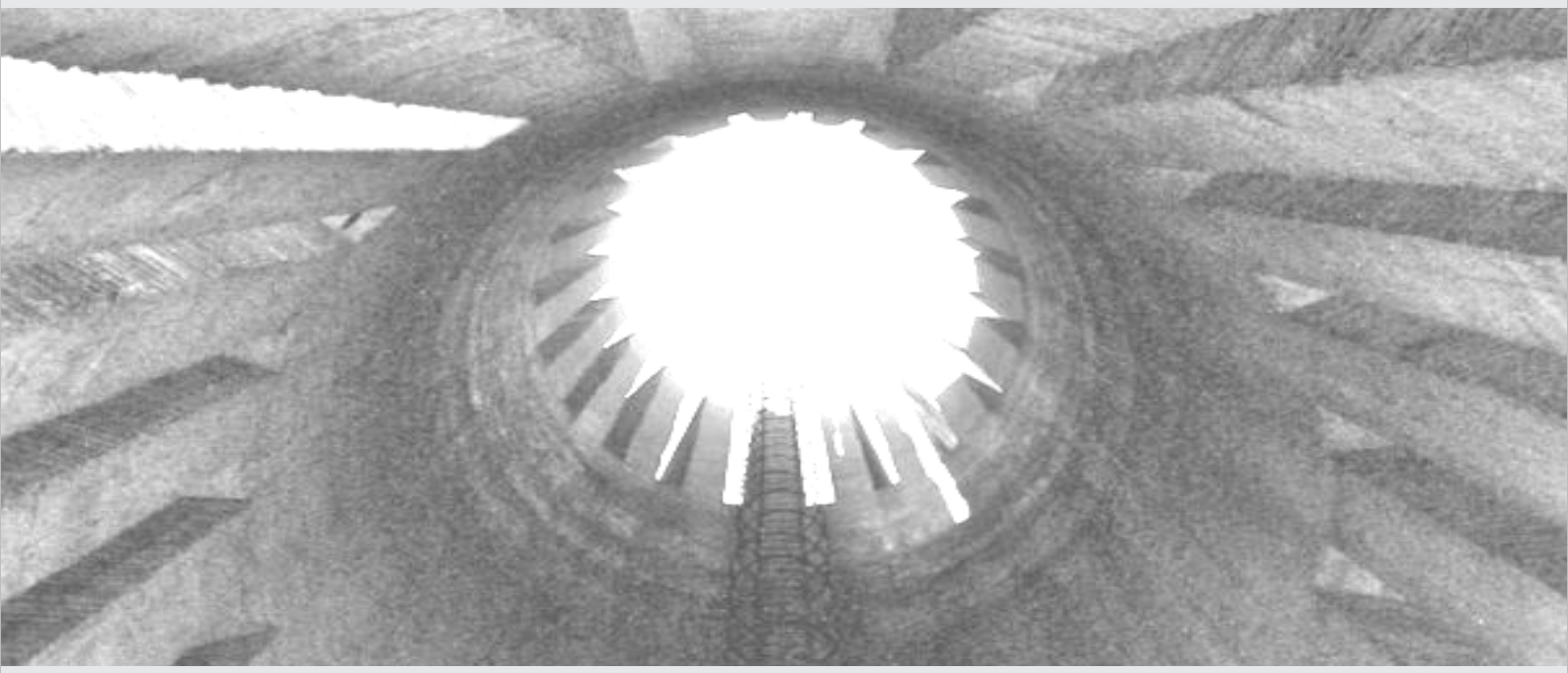
7th Balkan Mining Congress



"BALKAN MINING FOR THE FRIENDSHIP AND PROGRESS"

Book of Proceedings
I



11-13 October 2017, Prijedor

Year 7, No.7 (2017) ISSN: 2566-3313

Proceedings
BALKANMINE

BALKANMINE 2017
7th Balkan Mining Congress

BOOK I

Prijedor, October 11-13, 2017.

CIP - Каталогизација у публикацији
Народна и универзитетска библиотека
Републике Српске, Бања Лука

622:55(497)(082)

BALKAN Mining Congress (7 ; 2017 ; Prijedor)
Balkanmine : Proceedings. Book 1 / 7th Balkan Mining Congress, Prijedor, October 11-13, 2017. ; [Editors Slobodan Vujić, Vladimir Malbašić]. - Prijedor : University of Banja Luka, Faculty of Mining ; Belgrade : Mining Institute, 2017 (Banja Luka : Mako Print). - 382 str. : ilustr. ; 30 cm

Kor. nasl.: Balkan Mining for the Friendship and Progress. - Na nasl. str.: Year 7, No.7 (2017) ISSN: 2566-3313. - Tiraž 200. - Bibliografija uz svaki rad. - Registar. - Abstracts.

ISBN 978-99955-681-7-7 (Faculty of Mining)

COBISS.RS-ID 6803736

7th Balkan Mining Congress
PROCEEDINGS

Congress Organizers:

UNIVERSITY OF BANJA LUKA

FACULTY OF MINING PRIJEDOR

BALKAN ACADEMY OF MINING SCIENCE

ACADEMY OF SCIENCE AND ARTS OF REPUBLIC OF SRPSKA

UNION OF ENGINEERS MINERS AND GEOLOGISTS OF REPUBLIC OF SRPSKA

Proceedings Publishers:

University of Banja Luka Faculty of Mining Prijedor
Save Kovačevića bb, 79101 Prijedor, RS/BiH

Mining Institute Belgrade Ltd
Batajnički put 2, 11080 Beograd, Zemun, Serbia

Editors:
Academician Slobodan Vujić
Prof. dr Vladimir Malbašić

Technical Editor:
Prof. dr Lazar Stojanović

Design, text capture and processing by:
Lazar Stojanović
Dražana Tošić
Miodrag Čelebić

Printed by:
MAKO PRINT d.o.o. Banja Luka

Issued: October 2017

Circulation: 200

www.balkanmine2017.com
www.rf.unibl.org/

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VI Balkan Mining Congress

September 20-23, 2015.
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IN MEMORIAM



Assistant professor Tomo Benović was born on January 06, 1958 in Bogutovo Selo in Ugljevik. From 01.02.1982.Tomo Benović was employed in Rudnik i termoelektrana Ugljevik in the following works: trainee, shift manager, technical manager of coal production, assistant director (for mines and technical business), manager for mining and geological service, director of RiTE Ugljevik, coordinator for coordination with Regulatory Authorities, Team Leader of Project Implementation and realization of investments and projector for the mine. Tomo Benović was the first Mayor of Municipality Ugljevik and in the period 2000-2002 he had been a member of the National Assembly of Republic Srpska. Tomo Benović had been in the following scientific and professional organizations and associations: the Chairman of the Alliance of Engineers and Technician of Mining - Geological and Metallurgy Profession, the membership of the International Coordination Committee of the Balkan Mining Congress from Bosnia and Herzegovina in two mandates.

At the Senate of the University of Banja Luka session, held on August 25, 2016, Tomo Benović PhD in mining was elected as Assistant professor for scientific research - Surface exploitation of the mineral raw materials. Assistant professor Tomo Benović tragically died on 27 November 2016 in a traffic accident.

Preface

Dear Colleagues,

On behalf of the University of Banja Luka, the Faculty of Mining Prijedor and the International Coordination Committee of the 7th Balkan Congress, we welcome you as respected and dear guests of the University and Faculty, Prijedor, Republic Srpska and Bosnia and Herzegovina. The 7th Balkan Mining Congress has a motto **"Balkan mining for the friendship and progress"**, which speaks enough about the basic idea of organizing and holding this event. This Conress has been held biennial in the Balkan countries.

This international meeting is an opportunity for Congress participants - authors of works, sponsors, exhibitors, representatives of institutions and companies to meet each other, exchange experiences in solving problems and issues related to the development of mining, geology, and the work of companies. Every opportunity to hear something new, something that is applied in other countries and conditions is the chance to find a chance in this transition period which is difficult for the work and development of mining companies. The exploitation of mineral resources could be beneficial, for the producers themselves,and for local communities and countries where are these mines located.

In contemporary trends in the mining and geology development, there are dilemmas to reconcile certain, at first sight, completely opposed and incompatible activities: mining, environmental protection, optimal economic effects of mining activities for the concedents and concessionaires. The Balkan Mining Congress is a unique opportunity to talk about these issues, exchange experiences, find solutions and align certain models of more rational solutions.

Wishing to feel comfortable and pleasant in Prijedor, and after the end of the Congress, you go home happy and with the view that it was worth being here, I greet you in my personal name and the name of the University in Banja Luka-Mining Faculty Prijedor and others co-organizer of the Congress.

Prijedor, October 2017.

Assoc. prof. Vladimir Malbašić

Chairman of the Organizing Committee
of the 7th Balkanmine Congress

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EXPLOITATION OF MINERAL RAW MATERIALS AND TECHOLOGICAL PROCESSES

DOI: 10.7251/BMC170701001G

VALORIZATION OF OVERBURDEN ON THE OPEN PIT OF RUDNIK UGLJA-PLJEVLJA, MONTENEGRO

Miodrag GOMILANOVIĆ[1], Miodrag KALUĐEROVIĆ[1]

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ABSTRACT

The open pit "Potrlica" of Rudnik uglja- Pljevlja, Crna Gora, has a capacity of 1.5 million tons of coal per year and about 6 million m^3 s. m. overburden. The overburden consist the natural cement marls as part of complex of the Miocene lake sediments. The average quality of cemented marls is quite balanced by the mineral and chemical composition suitable for cement production.

The paper aims to point out to the extraordinary economic validity of a cement plant building on the base of overburden that is waste now - the cost of coal production and the use of coal as a technological fuel in that factory. The paper will also point to the necessary measures of environment safety in the view of the climatic characteristics of Pljevaljska kotlina.

Reserves of this raw material provide a possibility of a cement plant capacity of 1.5 to 3.0 million tons per year. An additional argument is the significant need for cement as a product for the construction of infrastructure and other facilities in Montenegro, as well as for export.

Key words**:** overburden, marl, cement, coal, ecology.

1. INTRODUCTION

The Rudnik uglja- Pljevlja is located in the north of Montenegro at about 750 m above sea. It is the largest producer of this energy resource in the Montenegro. Coal production in this area began at the end of XIX and early XX century, and this company was founded in 1952. Production was done on two open pits: "Potrlica" and "Borovića". So far, a total of 70 million tons of coal have been produced, with the largest annual production of 2.7 million tons. Coal is used for the production of electricity (TE- Pljevlja 210MW), of industrial and wide consumption. Only in the area of some locality in field of Pljevlja existing exploitation reserves are only about 70 million tons. The average coefficient of detection is 4.15m^3, and DTE 10.720 kJ/kg.

Geology of the pljevlja's basin consist the three packages: underlying (predominantly clays with coal seams), main coal seam and immediate roof (marls and marl limestone). The applied exploitation technology is discontinuous, except in the part of the haulage of overburden, the combined transport is there. This technology is conditioned by the high strength of the layers of overburden and it is necessary the drilling and mining operations.

Excavation is carried out by the shovel excavators and draglines with excavator bucket of volume up to 15 m^3, and haulage by dampers loads up to 100 t.

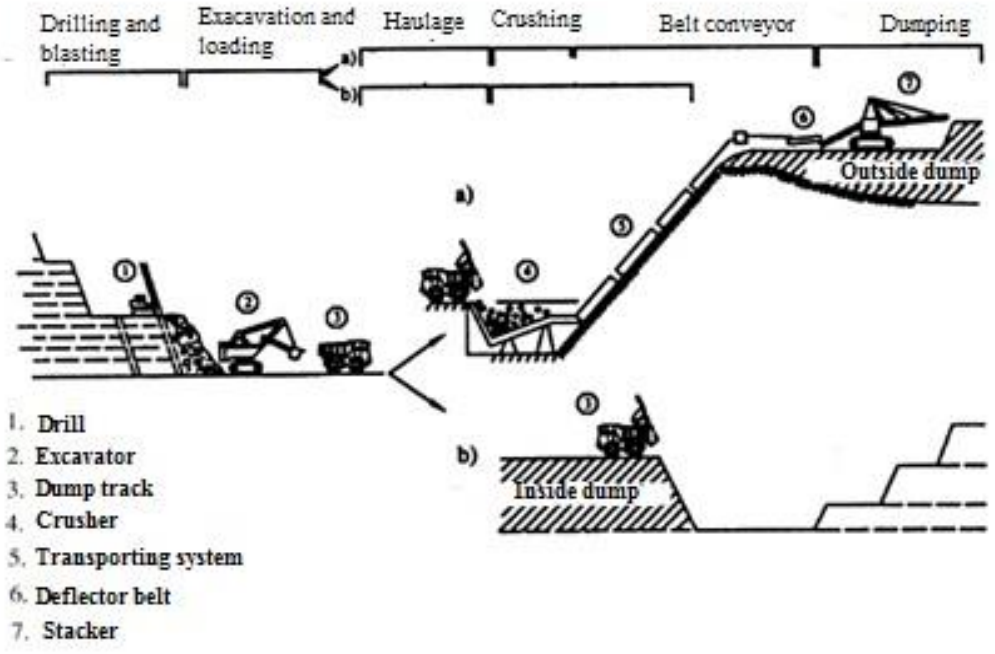


Figure 1. A technological scheme on overburden of open pit

2. OVERBURDEN

Overburden or roof of the main coal seam on the open pit "Potrlica" is made of the natural cement marls and marl limestones. They are developed in the entire spatial distribution area of the main coal seam. Their thickness rises from the periphery to the center of deposit, where it reaches thickness of 128 m. Within this sediment there are the four litostratigraphic units, primarily based on the content of $CaCO_3$ component. There are: lower laporic limestones 1) lying directly on coal seam and containing about 90% of $CaCO_3$, 26 m thick, lower natural cement marl 2) with 75% of $CaCO_3$, 45 m thick, upper marly limestones 3) with 85-90% of $CaCO_3$, 13 m thick and upper natural cement marls 4) with 70-75% of $CaCO_3$, 16 m thick.

From the aspects of applicability of the roof marls in Portland cement production the three upper sections (2, 3 and 4) are the most favorable. The current overburden thickness- the cemented marls in the surface of the "Potrlica" open pit ranges from 10 m to 115 m and average is about 80 m. It has a 10 benches, the first is 15 m and the other 10 m.

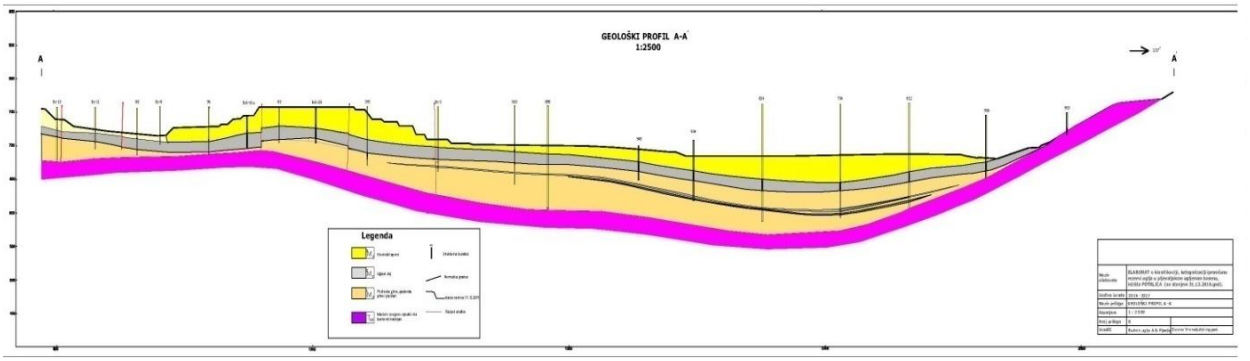


Figure 2. Geological layout of the open pit

The reserves of cement marls on this open pit in this period amount to about 145 million tone of m^3 of solid mass. Reserves of A+B+C_1 category are about 60%, and C_2 are about 40%. From this surface till now about 39 million $m^3$of solid mass of overburden-marls there have been deposited to the outside waste dump "Janjilo", to outside waste dump "Grevo" about 8 million m^3 of solid mass, when in the excavated area, inside waste dump loading about 100-110 m^3 of solid mass.

It should be emphasized that in the area of the coal field Pljevlja at other sites-deposits also overburden of marls of high quality. The reserves of cement marls in locality "Kalušići" are about 13 million tons, locality "Rabitlje" 12,5 million tons and locality "Grevo" about 1 million tons. These reserves are categorized as C_2 reserves.

The average quality of the balancing reserves of cement marls on the open pit "Potlica" is as follows:

- *SiO2 …… 13,48 %*
- *Al2O3 …… 3,11 %*
- *Fe2O3 …… 2,06 %*
- *CaO …… 41,99 %*
- *Loss on ignition…… 37,39 %*
- *MgO2 …… 0,69 %*
- *SO3 …… 0,19 %*
- *Na2O …… 0,33 %*
- *K2O …… 0,56 %*
- *P205 …… 0,08 %*
- *MnO …… 0,25 %*
- *Degree of clinker saturation (DS) …… 96,41 %*
- *Hydraulic module (HM) …… 2,28 %*
- *Silicate module (SM) ……2,73 %*
- *Aluminate module (AM) …… 1,51 %.*
- *Specific gravity of pure marls……from 1,85 to 1,95 t/m^3.*

Apart of marls from the existing open pit for the production of cement, in considerable quantities deposited marl on the "Jagnjilo" landfill can be used, while the inside dump waste and "Grevo" landfill, due to the worse parties of clayed coal and clays, as well as technology of excavation, haulage and loading, usable in a smaller extent. This has become practically the potentially technical deposits.

3. THE VALIDITY OF CEMENT PRODUCTION

Reserves, quality and other comparative advantages have influence that Rudnik uglja access to the realization of an investment program for the cement factory since 1976. The plant had a capacity of 327 thousand tons of cement annually, with the applied technology of wet process. Unfortunately, the factory worked until 1988 with 53% of the capacity for production of clinkers and 50% for cement production. The reasons for production termination were economic and ecological. The factory had inadequate and expensive technology with high specific energy consumption per unit of the products, exceptionally small capacity, and location near the city with inoperative of the electro filters.

From today's professional and economic view, the approach to the construction of cement factory is quite different. A significant contribution to the explanation of cement industry development in Pljevlja is given in the Preliminary Study of cement factory construction in Pljevlja, II phase of 3.000.000 tons per year, which was done by the Mašinoprojekt- Kopring, Belgrade 2005 for the Rudnik uglja- Pljevlja.

The interests of cement factory construction are multiple and can be defined, in addition to the general social, and the interests of direct users as follows:

a) The interests of Montenegro

- Emigration and reduction of the number of inhabitants from the north of Montenegro is one of the major social problems primarily because of fail of formerly developed industry in these areas;
- Production or haulage of cement increases the use of rail from Bijelo Polje to Bar and Bijelo Polje to border of Serbia.
- Use of red sludge as technological waste in Podgorica.
- Significantly improving the trade balance to abroad.

b) Interest of Rudnik uglja Pljevlja

- Current overburden as a cost of coal production would be the basic raw material for cement production;
- The global negative attitude toward to production of electricity from thermal power plant on coal is causing the price drop of coal, so it is necessary that this company to search the other areas of coal usage where conditions exist;
- An eventually developed of cement industry lead to an increase in the value of coal, while the use of marl lead to a reduction of productional price of coal.
- The production of electricity and cement production in this area and under these condition are not excluded, on the contrary have a number of comparative advantages.

c) The interests of population of Pljevlja Municipality

- To develop a profitable industry based on coal industry, which will with the provision of well paid jobs to increase the purchasing power of the population, from which the population of this municipality in the wider area will benefit;
- Cement production enables the development of a number of labor intensive capcities on the basis of this product and employment of the workers.

4. CEMENT PRODUCTION TECHNOLOGY

Mineral processing for cement production, the clinkers is carried out by dry process, while the wet process is no longer in use. Clinker production, as the most important process in cement production is done in rotary kilns. The modern rotary kilns have new types of burners and the systems for the air preheating for combustion. They enable that for the clinker production can to use the various types of fuel such as: fuel oil, petroleum coke, various types of coal in pulverized state, used oils, used car tires, and various types of waste. According to the fact that the clinker production is about 80% of the total costs of cement production, the energy costs are around 35-405, and therefore the importance of reduction of energy consumption, and consequently, reducing the amount of gas as the products of combustion process.

Many offers, informative interviews and documents suggest that the cement industry in Pljevlja by application of *BAT (Best Available Technology)* can be achieved by consumption of 300 GJ/t of clinker and 100 KWh/t of cement. It is therefore realistic to expect that cement production in Pljevlja will have a consumption of 0.3 tons of coal per ton of clinker and about 100 kWh per ton of cement. Norm of coal consumption would decrease for the energy that would come from of municipal waste, old oils and other energy waste.

The listed characteristics of the production parameters refer to the Portland cement production, with the note that other types of cement can be produced in the same equipment.

The most important alternative types are aluminum cements, for which is availability of different types of red and white bauxite in Montenegro. The bauxites of Montenegro were produced in the factories of Europe and Yugoslavia, and particular in the cement factory in Pula. In the world, more and more red sludge is used in cement, in quantities of 10-30%. A special advantage is the fact that in Podgorica the red sludge is an environmental problem and in the case of construction of the cement it becomes a significant input. The red sludge would be previous calcined in an existing rotary kiln that is not being used because a new kiln for calcination of alumina has been built in a case that the alumina factory continues for now stopped production.

5. INVESTMENT AND BASIC ECONOMIC INDICATORS

Cement production capacities of 3 000 000 tons per year, according to the completed study, is 355 000 000 €. Cost structure for this capacity is given in the following table.

Table 1. Cost structure of the cement production

	Production of cement 3.000.000 tons per year			
No.	*Type of costs*	*Costs €/per tons*	*Total costs 000 €*	*In % in ratio of total costs*
	1	2	3	4
1	*Interest*	4.47	13401330	8.67
2	*Rate*	12.33	36978500	23.92
3	***Annuities***	**16.79**	**50379830**	32.59
4	*Energy*	**12.88**	38630000	24.99
5	***Bruto salary***	**3.36**	10087000	6.52
6	*Investment maintenance*	**1.83**	5500000	3.56
7	***Raw material***	**8.97**	26920000	17.41
8	*Other material costs*	**1.87**	5607000	3.63
9	***Immaterial costs***	**2.49**	7476000	4.84
10	*Unexpected*	**3.33**	10000000	6.47
11	***Total***	**51.53**	**154599830**	100.00

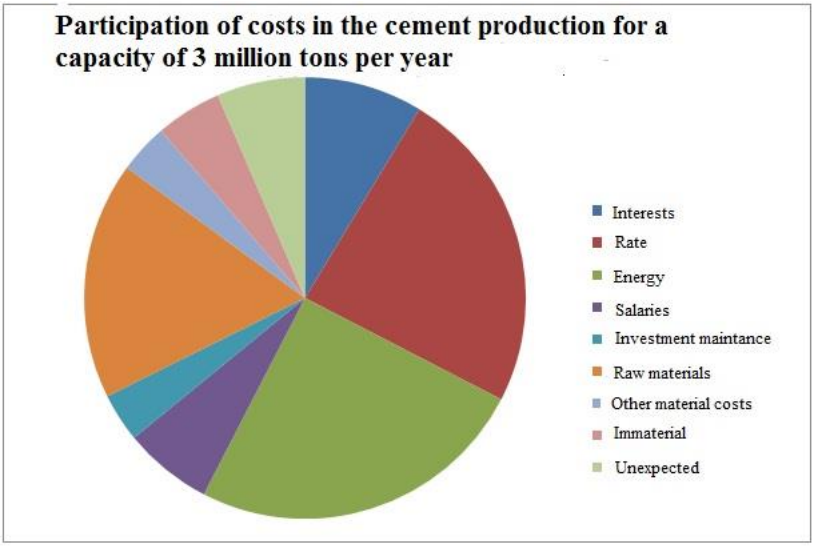


Figure 3.Structure of the costs

The following important facts should be noted from the table of production costs and graphic layout:

1. Production costs, including the capital costs, where 12-year the repayment are planned and average annual interest of about 4.5% is 57% of the sales price of cement.
2. In addition to the capital costs, key costs are energy and raw material costs.
3. The costs of coal at a consumption of 0.3 tons per ton of cement, which is financially 7.8 € per ton of cement are introduced in the energy costs. Due to the nature of technological process in cement, 75 kg of ash from used coal, with value of about 6,7€, which means that ash from coal practically null the cost of coal in cement production.
4. In other application of coal ash is significant cost and environment load.
5. In the cost of raw material, the costs of marl and filter ash are calculated, which are usually load to environment and cost, while in cement production their consumption is part of the solution to the problem.

According to the Study, the possibility of Portland and special calcium sulfoaluminate cement in Ugljevik pg. 62, the price of cement in environment ranges from 88-92.5 €/t, without VAT. For the projected cement production of 3 million tons per year, and price of 88 €/t, without VAT, the total annual revenue is 264.000.000 €. The costs for this production, according to the previous table (column3), amount to 154.599.830 € and profit 109.011.130 €.

6. ENVIRONMENT SAFETY

From the aspect of environment safety, the critical factors are the emissions of carbon dioxide and other gases both in cement production and in the production of electricity. Another important component that endangers the environment is the dust which follows this production process. The amount of emitted carbon dioxide and other harmful gases depends on the amount of fuel which burns and from the modernity of technology. The choice of location of the production facilities has an important impact on this area. Taking into account the climatic characteristics of the valley of Pljevlja, so that to this problem needs to give a special attention.

A special advantage for the cement industry in Pljevlja is also the fact that for the thermo power plant in this municipality ash disposal is a problem which will be used as a very important raw material in the cement production.

The complex use of raw materials and the recycling of industrial and other waste are the main problem of the environment safety. The cement industry is on the line of solving both mentioned problems.

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DOI: 10.7251/BMC170701009D

CONSTRUCTING THE BORDER REINFORCED CONCRETE PILLAR FOR MINING OUT THE RESERVES IN VEIN DEPOSITS

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ABSTRACT

Border (arch and drift) pillars are often left to ensure the geomechanical stability when mining out vein deposits. These pillars remain unmined and at the same time they are re-dimensioned. The dimensioning of the border dividing pillars of this type is usually very complex because of: the uncertainty of the acting voltages (natural and induced), the properties and structural characteristics of the rock mass; the constantly changing volume of the rock mass; the dynamic and static – individual and co-acting – loads on the pillar, etc.

The article aims to present a technological approach when applying a cut-and-fill stope method of mining. A technological solution of the problem is presented by constructing an artificial reinforced concrete dividing pillar. The study is intended for implementation with regard to the mining out of the deposits in the vein section of " Varba-Batantsi" mine located in South-Eastern Rhodope Mountains in Bulgaria.

Key words: vein deposits, reinforced concrete pillar, constructing

1. INTRODUCTION

Varba-Batantsi deposit is situated in the territory of the village of Varba and the town of Madan, Madan municipality, Smolyan district in the Republic of Bulgaria, south-eastern part of the Western Rhodope Mountains. The mine is located 7 km southwest of the town of Madan and its average altitude is from 910 to 960 m. The mining of Varba deposit started in 1952 and continued until 1999 when it was conserved and the mine was closed down. The mining of Batantsi deposit started in 1962 and continued till 1999. In 1995 the two deposits were united in one mine – "Varba-Batantsi". Consequently, in 1999 the conservation of the "Varba" deposit was made, it was flooded from level 540 to level 740; as a dock to it, the "Batantsi" deposit was also flooded as the two deposits are interconnected via level 590.Only in 2007, after an eight-year interruption of the mining activities, one of the first procedures for drainage of a flooded mine in the Republic of Bulgaria - "Varba-Batantsi" mine started.

The "Varba-Batantsi" deposit is opened up with three vertical shafts: the "Varba-1" shaft opens up the reserves from level 940 to level 540, and it is equipped with a cage winding for the wagons; the "Varba-2" shaft is driven from level 920 to elevation 175, it is not reinforced

and there is a project for the use of a two-storey wagon cage; "Batantsi" shaft is driven and opens up the reserves from level 850 to level 490 and it is equipped with a cage winding and a one-storey wagon cage. The elevation view of the mine is presented in Figure 1.

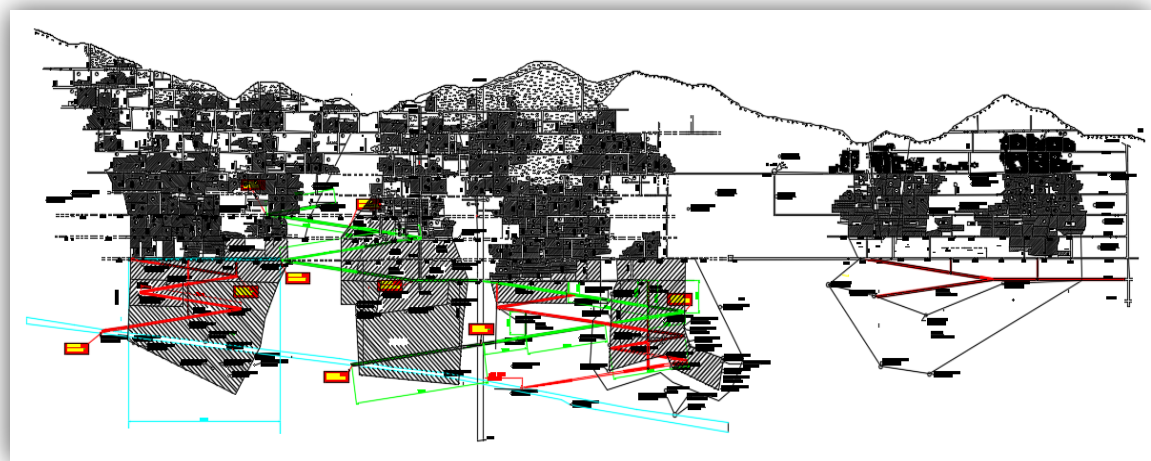


Figure 1. Elevation view of "Varba-Batantsi" mine

The methods of mining which are currently implemented in "Varba-Batantsi" mine are: sublevel stoping method of mining and flat back method of mining with ascending fill with rock material. The first one is characterized by low productivity due to the great preparation for the cutting of the block and the processes related to the draining and supplying the ore. For the latter it is necessary that the rock material - the dry rock fill - has high compression properties in order to be hydraulically compacted and uniformly fed. The deficiency of this method of mining is the waste of time to build a chimney in the filled space that is 50 m high.

The mining systems applied at the "Varba-Batantsi" mine are not sufficiently productive and can not guarantee maximum exploitation of all proven reserves in the deposit. They generate different geomechanical problems in terms of scope and area, and they determine multiple interrelated risk factors.

2. PROBLEMS

The presented problems can mainly be divided into two aspects: the first one is related to the mining activities and the second one to the provision of the subsequent geomechanical stability when applying the adopted technologies and methods of mining.

The problem with the mining works is also divided into two parts:

- Problems related tothe mining out of the reserves below level 590 around one of the shaft pillars.
- Problems related tothe technologies of the cut-and-fill method of miningapplied so far - its low productivity.
- The geomechanical problems that appear are also of two types:
- The first is the subsequent (secondary) "rock pressure" that occurs around service mining workings close to the mined out blocks [3].
- The second is the deformations that occur in the main opening after the start of the mining out of the reserves close to them.

All this necessitates a unified solution of the identified and interrelated key factors, which will solve the problems that determine the future exploitation of the "Varba-Batantsi" deposit.

The aim of the present study is to propose an innovative method of mining generating a minimum geomechanical risk, which will ensure an adequate output and mining out of maximum quantities of proven reserves in the "Varba-Batantsi" deposit.

3. CONSTRUCTING A REINFORCED CONCRETE PILLAR

The two methods of mining applied at the "Varba-Batantsi" mine are characterized by low productivity and high share of the preparatory and cutting operations. The subsequent geomechanical stability in these methods of mining is low and the level of geomechanical risk is high. All this requires an innovative approach that is close to the applied method of mining.

With regard to the aforementioned circumstances, an innovative method of mining has been proposed and developed, with the separation of the mining block in two sublevels by means of a reinforced concrete pillar. The construction and location of the reinforced concrete pillar is presented in figure 2.

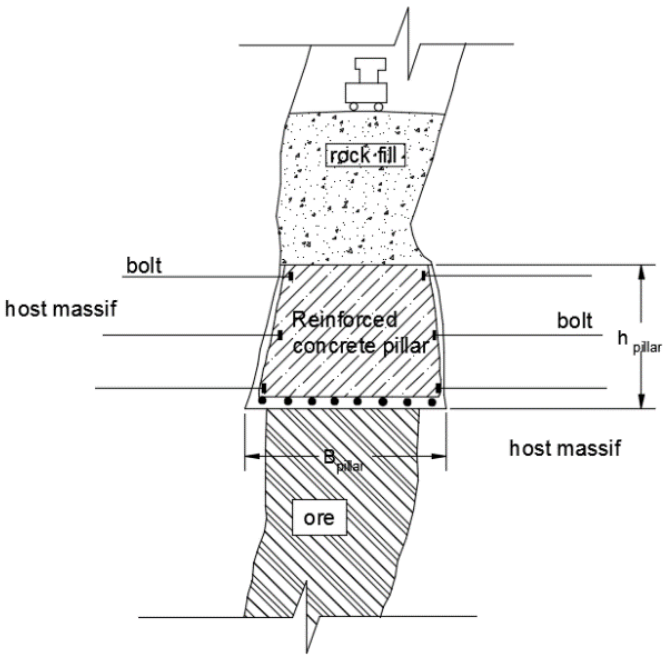


Figure 2. Construction a) and location b) of the reinforced concrete pillar during the mining out of the reserves.

The location of the reinforced concrete pillar is in the middle of the mining block, allowing the splitting of the block into two 25 m sublevels. The construction of the pillar is in three main stages: the first stage is the preparatory driving of a single mining cut; the second stage is the construction of reinforcement (longitudinal and transverse) along the entire length of the driven mining working; the third stage is accompanied by pouring of concrete throughout the pre-reinforced mining working.

The purpose of the reinforced concrete pillar is the complete mining out of the reserves in the block and guaranteeing the geomechanical stability of the mined out spaces and the host rock mass. For this purpose, it is necessary to carry out a preliminary analysis of the stress and strain state (SSS) of the reinforced concrete pillar.

4. ANALYSIS OF THE STRESS AND STRAIN STATE OF THE REINFORCED CONCRETE PILLAR

The newly constructed reinforced concrete pillar needs to be dimensioned [1,2] and for this purpose the following parameters are defined: pillar's width, pillar's height, pillar's length, pillar's bearing capacity, as well as shear strength at the "feet" of the pillar, the bending strength and the deformations occurring in the pillar. To help the analysis, part of the calculated results are summarized in Table 1.

Table 1. Geometrical and mechanical parameters of the reinforced concrete pillar.

Parameter	Calculated parameters	Measuring units
Pillar's width	4,5	m
Pillar's height	3,5	m
Pillar's length	50	m
Pillar's bearing capacity	12,00	MN/m
Shear strength at the bottom ("feet") of the pillar	2,0	MN/m
Maximum moment of bending of the pillar	2.0	MN.m
Maximum deformations in the pillar	0.3	mm

The software product of Rocscience Phase^2 v9.0 [4] was also usedto analyze the stress and strain state of the constructed reinforced concrete pillar.

The parametric analysis of the pillar's state with the softwareRocscience Phase^2 v9.0 makes it possible to determine the maximum main operating voltage - σ_1, the minimum main operating voltage - σ_3, the local stability factor FS and the absolute deformations at each stage of the mining operations.

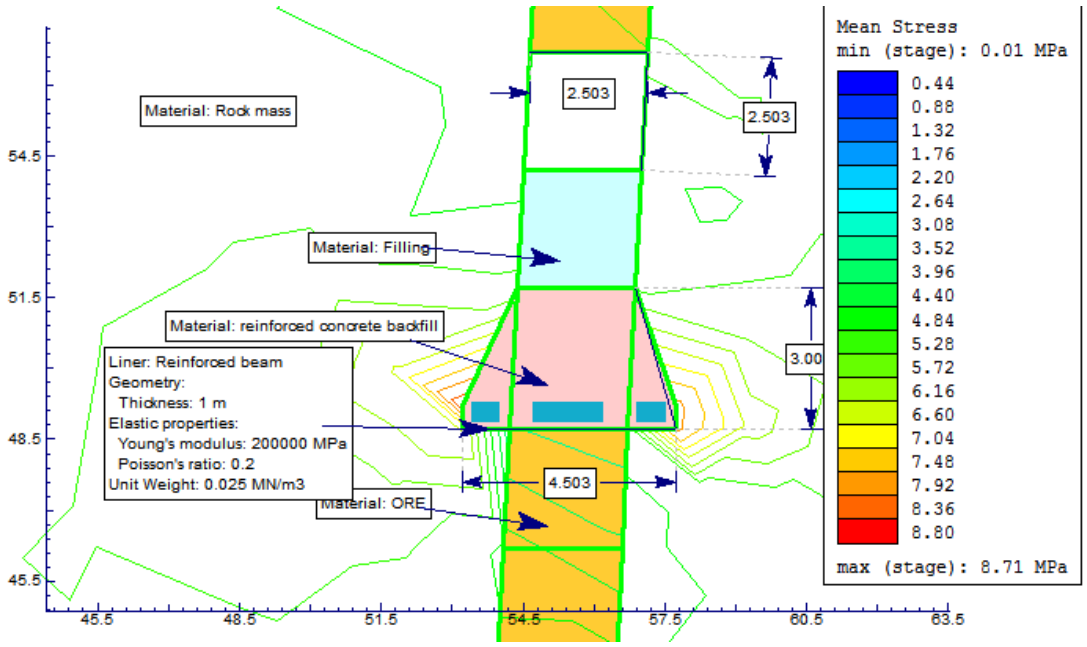


Figure 3. Analysis of the stress and strain state of the reinforced concrete pillar with the software Phase^2 v9.0

According to the conducted numerical studies [5] the stresses in the pillar (σ_1) have a magnitude varying in the range of 3.5 to 5 MPa. The study also shows that no stress concentration zones are formed around the pillar after its construction. Concentration of stresses in the pillar is formed only during the first stage, during the driving of the mining working. The study of the stress state in the host rock mass around the pillar and around the mining chambers does not reveal dangerous tension stresses. The parametric analysis of the state of the system host rock mass/open mining spaces allows to determine the deviator of stress (σ_1- σ_3). The results of these studies show that the deviator of stress in the pillar is within the range of 2.4 to 3.2 MPa. This result shows that there will be no significant increase in the stressed state of the pillar in the various stages of mining out the reserves over it, which guarantees its local stability. The latest pillar state analysis is based on the determination of the local stability factor (FS). The analysis found that the FS factor in the various stages varied from 1.26 to 1.45. In this case, the FS almost does not change, its values are above 1.0 and this is a prerequisite to ensure a steady state of the ppillar.

5. EFFICIENCY OF THE PROPOSED METHOD OF MINING

The proposed method of mining has a number of advantages over the methods implemented in the "Varba-Batantsi" mine up to now. These advantages make it possible to guarantee high productivity and minimum geomechanical risk to workers.

The mining process is carried out using mobile machinery, ensuring high output intensity.

Temporary non-active reserves are being put into operation in shaft pillars, under roads, rivers, ponds and other engineering facilities.

It provides a high level of mining out the reserves and several times lower losses and impoverishment. The construction of the reinforced concrete pillar and the laying of a rock-cementing fill of the mined out layers of the rock mass are compensated with the reduced damage caused by the loss from the impoverishment.

6. CONCLUSIONS

The proposed new method of mining allows for the mining out of uncovered and proven reserves close to main openings, blocked-undermined reserves at medium and large depths, reserves to be mined out in newly formed mining blocks, etc.

The new method of mining with division of the extraction block in two sublevels allows for higher productivity, with the simultaneous operation on four mining faces, which is a major and substantial advantage in the underground mining of deposits.

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DOI: 10.7251/BMC170701015M

AUTOMATIZATION OF COAL TRANSPORT ROUTES FROM PREPARATORY SITES

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ABSTRACT

In Velenje Coal Mine we construct annually up to 6000 m of new roadways for the production process needs. Almost all roadways are made in coal seam with a conventional method, by cutting the profile in phases and installing combined steel arched and wood support system.

The roadways are constructed with GPK PV machines, which were developed with partners and adapted to our specific conditions. We have also developed a new PL PV-08 steel arches feeder, which significantly contributes to the humanization and safety in the work process. On almost each preparatory site we have installed HTZ coal crusher, which was developed with our own knowledge. It is located on the double-chain conveyor directly behind the road header.

Crushing of coal at the preparatory site is crucial for the implementation of automatization. Crushed coal can be transported via conveyor paths with reduced risk of causing failures to the transportation lines. Coal is crushed on first or second double-chain conveyor and transported further on the conveyor system. With the introduction of crushing and automatization of coal transport routes from preparatory sites, we reduced the cost of the roadway construction process, thus less workforce is needed.

Key words**:** preparatory sites, crushing unit, automatization

1. INTRODUCTION

In Premogovnik Velenje, we make annually up to seven km of new roadways. In order to manage the economic viability of roadways, the highest level of humanization of work and the highest level of occupational safety and health, the level of technological perfection should be kept at the highest level in the field of roadways.

For high-quality and quick production of coal mine facilities we need modern machinery and suitable machinery for working in the coal mine.

In addition to cutting machines, coal crushing and material manipulation (transport), in the Velenje Coal mine during the process of coal mine production, great emphasis is placed on the development and automation of coal transportation with transport systems to the main transport system and from there to the surface.

In the development, manufacture and installation of the coal mine facilities of all machined elements, the work was done by experts within the company in cooperation with external domestic and foreign experts.

2. EQUIPMENT FOR THE PRODUCTION OF COAL ROADWAYS AND VARIOUS EQUIPMENT CONFIGURATIONS

2.1 Advance loading machine gpk pv

In addition to an experienced work team, reliable, suitable and condition-specific equipment is also required for the efficient and reliable production of coal mine roadways. The advance loading machine is one of the key elements in the production of coal mine roadways.

On the basis of many years of experience, we first approached the Premogovnik Velenje for the development of a new advance loading machine. The new GPK PV is a product of the development of Premogovnik Velenje with business partners. The electrical equipment of the GPK PV machine was developed together with Bartec. The hydraulic steering system is the fruit of the development of the experts of Premogovnik Velenje. The steel basic construction was made by the Russian business partner GPK.

The GPK PV is adapted to the conditions in the Velenje coal mine and has proved to be very useful, efficient and versatile as such. It has a higher capacity, is more durable, stronger, easier to operate and more agile than its predecessor. There are 9 machines in Velenje Coal Mine, and we sold some of them to the mines around the world, where they are very satisfied with their versatility.



Figure 1: GPK-PV

BASIC TECHNICAL DATA OF THE MACHINE GPK-PV:

The GPK-PV is an electro-hydraulically driven console machine. The machine is used for the production of roadways, cross-sections to 22 m^2 below the inclination max. 25^o. Suitable for cutting materials up to approx. 70 N / mm^2. Along with the advanced machine it is also permissible to dissembling broken steel arches when rebuilding the coal mine roadways.

Progress:	in the slope range from ± 5º to ± 12º 2.6 m/min; in the slope line up to 8.3 m/min
Production of coal:	up to 1.42 t/min
Profile shape:	round, rectangular and trapezoidal

Dimensions off profiles:

Height:	1.8 - 4.6 m
Width:	2.6 - 4.7 m
Profile cross-section:	to 22 m^2
Track slope:	max. ± 12^o

Working body:

Design: cutting arm with head, gear and electric motor, telescopic movement

Engine power - Installations:	75 kW; 500 V, 1475 RPM
Engine type:	4KTCR 280 S-4, IMB5
Telescope stroke:	500 mm
Cutting heads:	60 rpm
Cutting arm displacement:	hydraulic

Loading part:

Design:	plug with loading levers, one central conveyor chain transporter
Plunger tip stroke:	0.5 m
Chain:	Ø 16 x 64; L = 16265 mm
Power:	EM 22 kW, 500 V, 1475 rpm
Plate width:	2060 - 2600 mm

Travel mechanism:

Design:	steel track
Shifting speed:	from 2.6 m/min to 8.3 m/min
Track length:	5070 mm (complete with stand)
Hydraulics:	
Working pressure:	160 bar
Pump capacity:	158 l/min
Capacity of the tank:	~ 350 l oil VG 46

Machine data:

Name:	GPK-PV
Length:	10500 mm
Track width:	177 mm
Plug width:	max. 2600 mm
Machine height:	2150 mm (cutting head)
Machine weight:	27000 kg
Specific pressure on the ground:	9 N / cm^2
Length of chain conveyor:	7000 mm
Operating voltage:	500 V; ± 5%, 50 Hz

2.2 The pl 08 pv arch feeding machine

As part of the development of equipment for the production of coal mine roadways and the urgency to maximize the level of humanization, we approached the development and production of a manipulator – arch feeding machine, which significantly facilitated the work on the preparation site. The manipulator of the steel arch support is used for the transport, manipulation of the steel arch support and other materials that are needed for the roadway construction. This segment of work on the preparation site is considered to be one of the harder ones, but the introduction of the arc feeder significantly facilitated the transmission of heavy loads.

The PL 08 PV is the product of the development of Premogovnik Velenje in cooperation with the business partner DBSS - Sovič and Bartec. Partner DBSS - Sovič produced a steel part, and Bartec took part in the development of the steering system. Ideal technical solutions and design are the work of the experts of Premogovnik Velenje.

The basic components are a supporting structure with running units on two parallel steel rails, a hand with gripper and a hydraulic device with a control pump, proportional hand controls and hydraulic cylinders as working elements. A hydraulic shunting cat powered by a pump from the feeder is used for driving. In the case of a dual propulsion engine that is above ±12°, the second drive - shunting cat is fed from the common control valve, and the shunting cat has its own aggregate - drive. One PL 08 PV is also equipped with remote electro-hydraulic control.

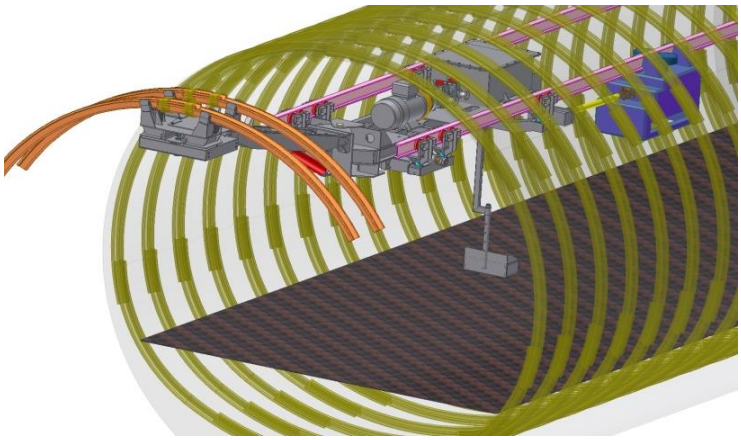


Figure 2: PL 08 PV arch feeding machine

BASIC TECHNICAL DATA OF THE MACHINE:

Rated force when lifting gripper:	F = 10 kN
Length with shunting cat:	8530 mm
Wheelbase 1500 mm between the steel rails	
Rotating the bearing head:	± 35^{o}
Lift the handle above the horizontal position:	+ 20^{o}
Raise the handle under horizontal position:	- 60 ^{o}
Maximum slope of progression:	± 15^{o}
Working pressure of the hydraulic pump:	P = 240 bar
Working pressure of the working part:	P = 140 bar
Pump capacity:	Q = 140l / min
Capacity of the tank:	~ 200 l of hydraulic oil VG 46

Machine dimensions without braking and shunting cat:

Length:	6600 mm
Width:	1838 mm
Height:	906 mm
Machine weight:	~ 4500 kg
Operating voltage:	500 V; 50 Hz
Installed power:	P = 22 kW

Brake unit:

Length along with 890 mm mounting bracket

Width:	375 mm
Height:	497 mm
Mass:	M = 205 kg
Static braking force:	F_{max} = 48 kN
Brake start time:	t = 0.3 sec

Drive unit:

Towing force:	R_v = 15 kN
Hold force:	F_d = 40 kN
Max. transport speed:	v_{max} = 0.6 m/s
Weight:	m = 740 kg
Working pressure:	P = 110 bar
Flow:	Q = 80 l/min

2.3 Ast 800 rubber conveyor belt

AST 800 rubber conveyor is part of electro-mechanic equipment used for mechanized production of coal mine roadways in Velenje coal mine. An 800 mm rubber belt conveyor is intended for the transport of coal from preparatory sites. The transporter consists of a basic steel part, a drive train and a conveyor belt. The drive consists of: two electric motors, a VOITH 422 hydrodynamic coupling, brakes and two gearboxes. In conjunction with the mobile crushing unit, the conveyor can also be equipped with a system for continuous strip lengthening.

BASIC TECHNICAL DATA AST 800:

Transporter with STAN AST 800 mm

Maximum operating length:	l = 800 m
AST 800 drive electric motors:	P= 2x45 kW
Transporting speed:	v = 2.1 m/s
Width of the rubber conveyor:	w = 800 mm
Conveyor belt angle (average):	-1.8 °

Transporter capacity:

Cross-sectional area:	S = 0,063 m^2
Volume current:	I = 0.127 m^3/s
Mass flow:	I_m= 106.3 kg/s (~ 390 t/h) alone

2.4 Coal crusher htz

In the process of development of preparatory sites we developed together with experts from HTZ the HTZ coal crusher. Crushed coal on the entire transport line is the basis for the successful execution of automation of the transport line system from roadway development. In the case that coal is not broken, they occur on the presses of causing large quantities of coal to fall and the loss of coal from the entire transport system. The crusher is mounted on the DVT 470 double-chain conveyor.

The crusher is based on the principle of direct drive with built-in asynchronous water-cooled electric motor in the crushing roller. The transfer of rotation to the crushing roller takes place through a planetary gearbox with a protective shaft. We took into account the basic operating requirements in the coal roadways and with the design solutions significantly reduced the height and weight of the crusher.

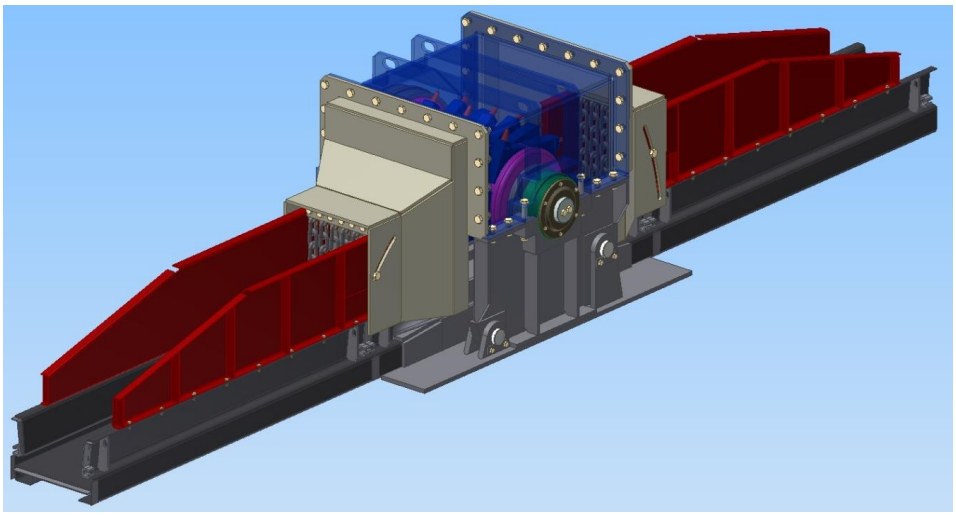


Figure 3: Coal crusher HTZ

3. EQUIPMENT CONFIGURATIONS AND VARIOUS PREPARATORY SITE CONFIGURATIONS

In the preparation site, we can have different equipment configuration, which depends on several different factors. Depending on the available equipment, we can assemble the entire equipment system for advancement, manipulation of material and transport of excavation from the preparation site, which in the given case best suits us.

3.1 Basic equipment in preparation workshops

- GPK-PV advance loading machine
- Double chain conveyor (DVT 470)
- HTZ coal crusher
- Shunting train
- 800 mm rubber belt conveyor (AST 800)

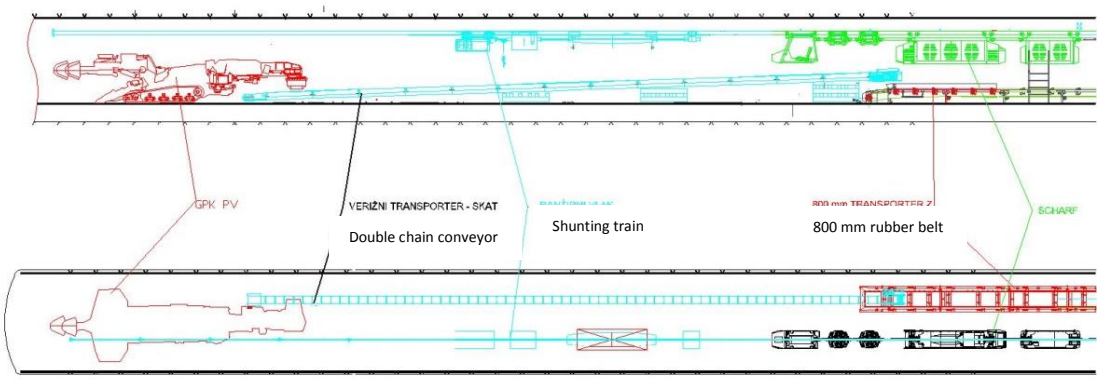


Figure 4: Diagram of equipment on the preparatory site

3.2 Basic equipment in preparatory sites with an arc feeding machine

Due to the higher level of humanization in the preparatory sites, we have developed and manifactured a new PL 08 PV steel arch feeding machine in the technological process. By doing this, we significantly facilitated the installation of steel arch support, while at the same time, with the PL 08 PV, we facilitated the delivery of the necessary material to the forehead of the preparatory site and the installation of steel arch support. A schematic representation of the described is shown in Figure 5 and is as follows:

- GPK-PV advance loading machine
- Double chain conveyor (DVT 470)
- HTZ coal crusher
- Shunting train
- 800 mm rubber belt conveyor (AST 800)
- PL 08 PV arch feeding machine

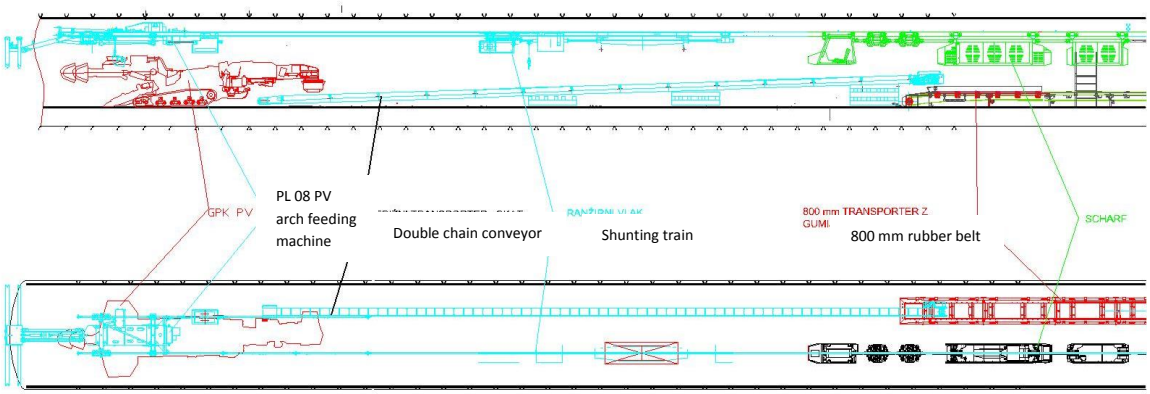


Figure 5: diagram of equipment at the preparatory site

3.3 Automatization of the transport system from roadway development

In the process of the production of coal mine roadways, we transport coal with the transport system to the main transport routes and from there to the surface.

In several cases, the shipping routes are long and broken down into individual transport blocks. When switching from one transport conveyor to another, it is necessary to provide the most continuous flow of coal. In order to ensure reliable operation, workers have to be

stationed on individual presses, which are responsible for switching on the components of the transport equipment. In cases where the breakdown of the roadway is large, we need a large number of people who would otherwise use them elsewhere in the work process to cover these prescriptive sites.

With the introduction of technical-operational solutions in the coal mine Velenje, we replace the individual switchgear and thus increase the economic optimization of the working process of the production of coal mine roadways.

We use the Becker Mining automatization system as a basis for managing and controlling the coal transportation process. The industrial computer Mincos communicates with peripheral input output units via two AST communication lines. Physically, AST is a communication line composed of communication, security and NF circuits. With AST line 0 we cover the control part of the conveyor. At the beginning there is a PP4100 adapter, to which switches for selecting the mode and turning on the tape are connected. At the plant there is a compact station KT 1002 manufactured by Becker Mining. For the compact station, the multi-adapter PE4110 is located, through which we monitor the Tuschen & Zimmerman brakes, belt tensioning position, speed control, drive gear reducer temperature, overflow protection, pre-tape lock.

To ensure safety, emergency stop switches and speaker unit are installed every 50 meters along the rubber belt conveyor. Off switches are connected to the steel rope, which is mounted on the hook side of the conveyor with a rubber band, which serves an emergency (safety) switch-off. In the middle of the conveyor and finally there are two pairs of switches for controlling the displacement of the rubber conveyor. On the last adapter there is also a switch-off switch and a connecting member, which gives the condition for the transporter to operate.

In operation of systems without the presence of servers, the tension control of the belt is also an essential safety element. In the event of slipping the belt off the conveyor belt drums, the belt speed is less than the nominal one. Due to friction between the belt and the drive drum, the heating occurs and consequently there can be a fire. The tread speed controller detects a reduction in the speed of the belt due to slip on the drive drum and sends a signal to the steering system which switches off the conveyor belt drive. Presets without the presence of servers are additionally protected with CO detector that are installed at the drive parts of conveyors with rubber band. The role of CO detector is to detect smoke (increased concentration of carbon monoxide) that would occur if slipping of the rubber band on the conveyor drum or any other rotating part of the conveyor would occur and, consequently, the conveyor operation was stopped.

On the presses of individual transport elements, oscillation probes are installed that turn off the operation of the conveyor system in the event of a blanket of one of the presses.

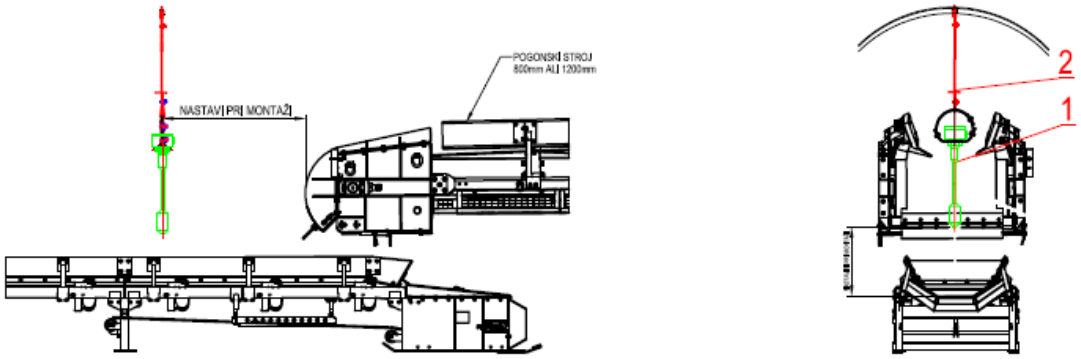


Figure 6: A swing probe mounted on a rubber belt conveyor AST 800

On conveyors with rubber belt we have installed, as an additional safety measure, also switches for rubber shift control. The task of these switches is to shut down the operation of the conveyor with a rubber band in the event of excessive deflection of the rubber belt from the center-ideal position to the left or right.

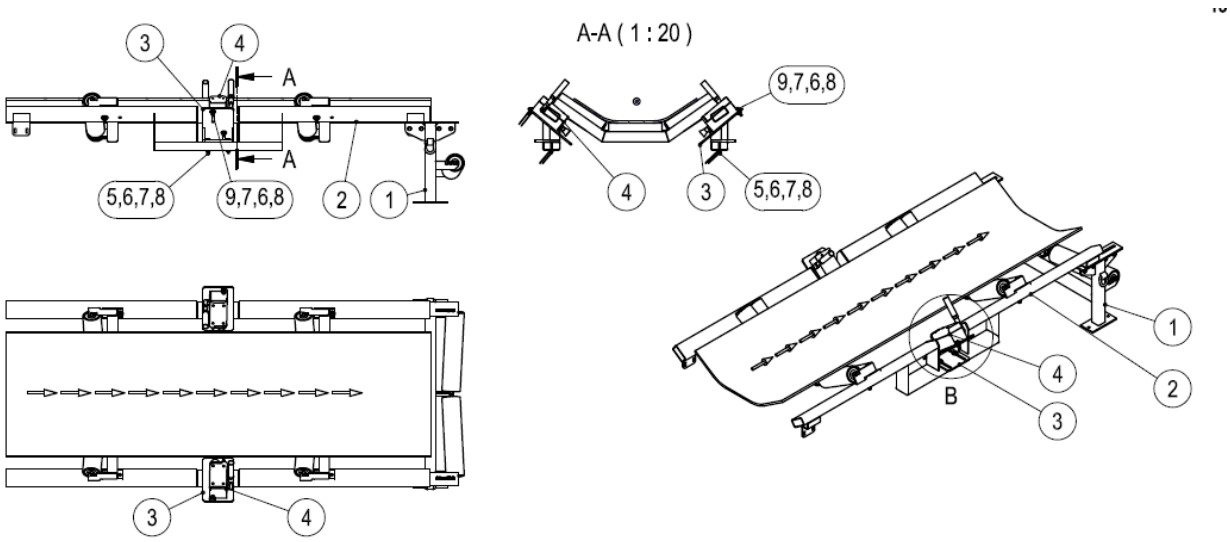


Figure 7: Tilt control switch on the rubber belt conveyor AST 800

In order to prevent additional dustiness in the coal mine roadways in the Velenje Coalmine, we use showers on all transport cross sections routes. The showers are switched on by completing the necessary logical condition for the transporter operating and by extracting the additional logical condition that the transporter is loaded with coal or excavated.

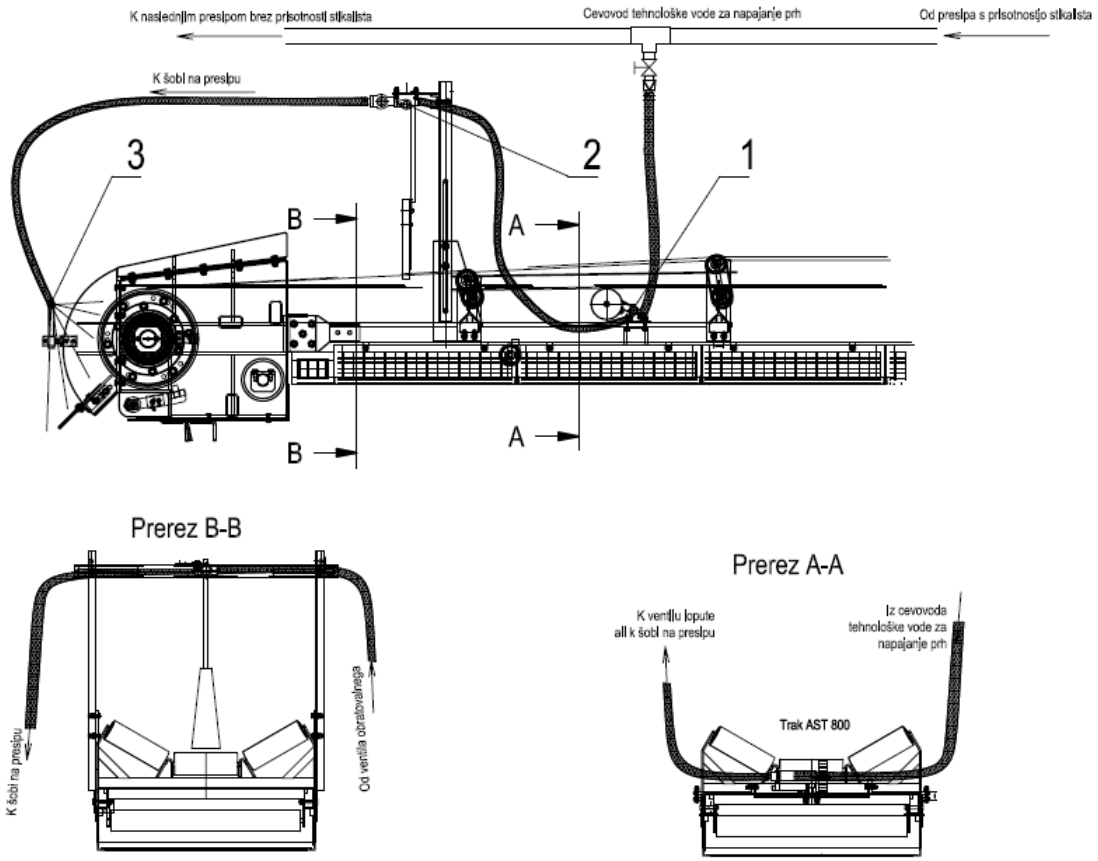


Figure 8: Automatic shower on AST 800 rubber belt conveyor

4. CONCLUSION

In Velenje Coal Mine, we pay great attention to continuous improvement and development of new equipment and processes, especially in areas that have a direct impact on safety and health at work and the humanization of work processes. An important development direction is also the tendency for a higher performance of the working process of the production of coal mine roadways, which directly affects the economic aspect of coal mining.

In the presented topic discussed and in the process of work, the introduced equipment and procedures are the result of our own development and the ideas of highly skilled personnel within the company in cooperation with foreign and domestic business partners.

Development products made in our company and fully functionality implemented in the coal mine Velenje, we offer potential buyers on the world market.

In Velenje Coal Mine, we pay great attention to development in all areas, while at the same time we strive to constantly improve work processes and equipment that influence the increase in the level of humanization and the highest possible level of safety and health at work.

DOI: 10.7251/BMC170701025G

THE BULQIZA CHROME MINES AND PROBLEMS OF ITS DEVELOPMENT IN DEPTH

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ABSTRACT

The Bulqiz chrome mine is one of the largest chromium mines in Albania. It represents one of the country's most important mining activities. The depth of its development of the mine makes it one of the deeper mines in Balkan and possibly in Europe. Last year, the inner mine shaft reached the quota -220 below sea level, in which the mine reaches a depth of 1000 m from surface. Like all deep metal mines and this mine during its development is faced with a different set of problems.

The first and most important challenge is that of improving the level of geological knowledge of this mine and assessment of geological resources and reserves according to the JORC Code 2012 Edition, because it is unique from a geological point of view Abstract.

In addition to geological problems a series of other problems related to increasing the depth of the mine will be the object of this paper such as:

- Rock mechanics, rock and ore stability, mine working support;
- Mining method and technological improvements;
- Groundwater and their pumping ;
- The temperature in the mine, the air physical parameters and the mine ventilation.

The mine has faced these problems using different calculation programs and simulations packages which will be presented in summarized in this paper

Key word : Chromium mine, inner shaft, VENTSIM, rock and ore stability, mining method, air physical parameters

1. GENERAL CONSIDERATIONS

The Bulqiza deposit itself is the most important chromium deposit in Albania and unique in its kind.

The documented beginning of chromium production in the Bulqize shown that mine started in 1948, and from this year it came continuously rising to reach the highest value in 1986 of 467,000 tons meanwhile the ore production form Bulqiza North mine has been 270 000 tons/year.

By the end of the previous year, Bulqize produced a production of 14.74 million tons. The amount of mineral resourec/reserves and the realized production shows that the Bulqize deposite is the most important source of chrome ore in Albania.

The exploitation of this source has been done with underground way by experimenting different mining methods and using, definitively, the sub-level stopping mining method.

Until 2001, the Chromium Mining Bulqizë had the status of a state-owned enterprise, while it is currently part of the concession agreement of the company "AlbCHROME sh.p.k. The chromium ore production trend of this mine was almost the same as that of in the country, representing more than 50% of it.

For ore exploitation of Bulqiza North Mine, since its inception up to 2014, a series of mining capital works have been constructed like:

1. About 2100 ml vertical shafts with diameter 4.-4.5 m;
2. More than 1120 m of declines with a cross-section over 7.2 m^2;
3. About 35 km of horizontal works for the preparation and use of minerals in the levels
4. A large number of vertical mine workings for ventilation and escape way;
5. Significant constructions and installations for the provision of infrastructure and power, compressed air and water supply have been carried out.(see fig no 1).

Currently, this mining, together with many other chromium ore and ferrochrome processing units, belongs to the local concessionary company "ALBCHROME" sh.pk, which is part of BALFIN Group.

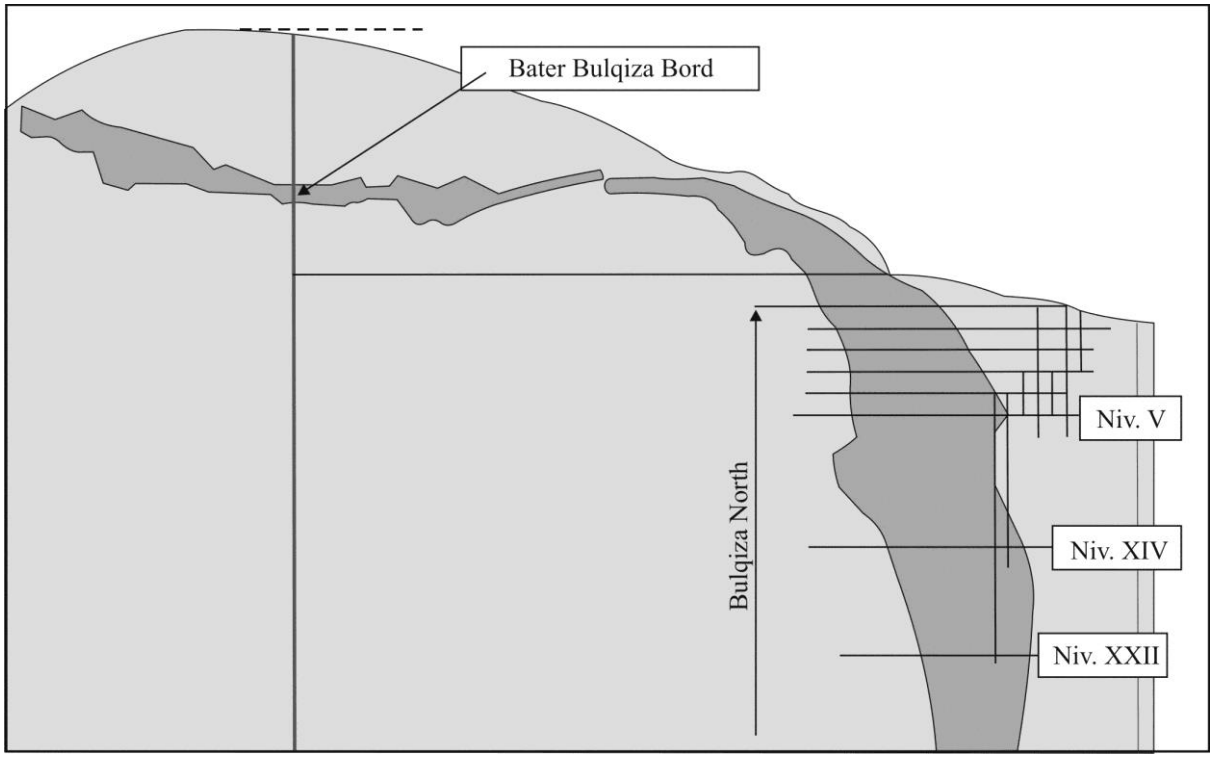


Figure 1. Bulqiza mine vertical section with main mine workings

2. GEOLOGICAL CHALLENGE AND ESTIMATION OF CHROMIUM MINERAL RESOURCES/RESERVES

The Bulqiza deposit itself, from the geological point of view, the ore body shape, the arms of which it consists, tectonics, the fracturing system, extension, deep angel, and further development is unique in its kind. For this reason it has been paid special attention by local and foreign researchers and designers.

The geological works carried out years ago could provide data regarding the geological construction of the deposit, structure and texture of the ore body, the tectonic and the estimation of the ore reserves (Qorlaze 1984, Gjoni 1988).

In market economy, banks, mining companies, the third shareholders involved in mining industry provide invest and support development of mining industry in conducting exploration survey, exploitation and processing of mineral resources. Under these conditions, the concession company and the operators are set up to carry out geological works to ensure the obtaining of the necessary information, which will enable the assessment of sources/reserves by international classifications (Goskolli 2001).

The abundant information obtained from the numerous geological drilling carried out by the underground can be done simultaneously (Nesimi R 2015).

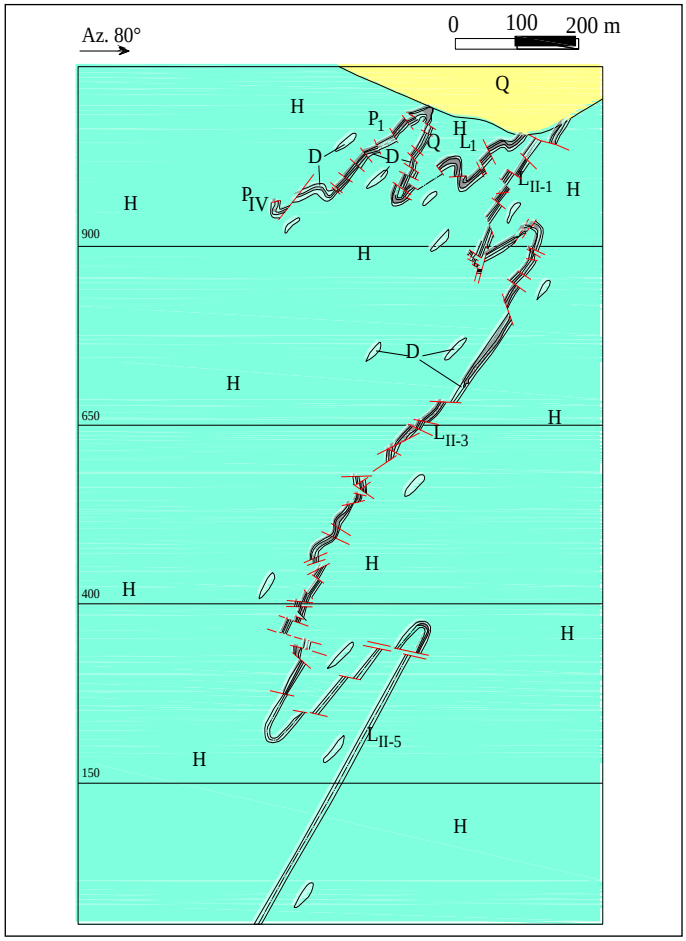


Figure 2. Vertical cros section of Bulqiza deposite

1. To make clear the visions of mineral resources based on certain classifications including parameter such as tonnage, ore grade, probability as well as factors such as geological, technological and economical aspects;
2. To collect enough data to evaluate the current mining method and to see the possibility of using another mining method;
3. To collect multiple data to make the assessment by means of the geotechnical classification of rock massifs and ore body and to judge about the types of support that can be used in the deep of the mine ;
4. To obtain samples of methane, hydrogen and other gases that are present in the mines.

Carrying out 25000 ml of a 76 mm diameter geological drilling, other geological works as galleries, traverses, raises, numerous geological documentation, measuring the zenital and azimuthal deviation of drilling wells, sampling and performing various chemical analyzes made it possible to comply with the strict rules of (JORC Code 2012) and CIM (2000);

The main data about the average thiknes of 2.6-3 m, the average content of Cr_2O_3 of 42.31 %., deep angle 60-75^o the total ore body lengh of on each level of 800 m, the intensity of techtonic and the other elements also are used in DATAMINE program to geometrise the ore body in 3D the average content of Cr_2O_3 of 42.31 %.

Finally, taking into account all the results obtained from previously made geological works and those of recent years, application experts (JORC Code 2012) and CIM (2000) are concluded that the amount of reserves is 2.12 million tonnes of mineral Chromium below the 17-th level, is classified accordingly

- JORC Cod 2012 , 50% of rezerves measured, 20% indicated and 30% infered
- CIM (2000):, 70% of rezerves proved and 20% peopable.

Appropriate valuation of chromium ore reserves provided security investors with the ability to study, project and continue investment in mining depth up to the absolute quota of -250 below sea level

In the mining perspective, the team in charge of drafting the depth project should solve the problems of the mining methode, the support that can by aply in different mine workings acording the kind of rocks in which they will be constructed, the mechanization of the works and the security problems related to the presence of different gases in the mines and mine ventilation

3. MINING METHODE

Mining methode that is used and still used in mines is the sub level stoping This mining methode has shown superiority compared to other mining methodes experimented in this mine and is likely to continue to be used.

Of course, in the cases of an importand ore body thiknes, it needs a lot of improvements and mechanisms to maintain proper performance and to use machinery with compressed air.(see fig N^o 3 and 4).

This mining methode is also favorised by the stability of the rocks of the body hunging part which are often composed by rocks with an RMR of greater than 70 while the mineral body with an RMR of less than 50. These geotechnical classification values favorise also the use of shrinkage mining methode , which is expected to be experimented soon.

4. THE GEOTECHNICAL CLASSIFICATION OF ROCK MASSIFS AND ORE BODY AND TYPES OF SUPPORT TO BE USED

As noted above, geological works carried out years ago, those carried out in recent years as well as the experience accumulated by the mining of mining works carried out during these 40 years of mining activity in this mine enabled the gathering of new data applied the geotechnical classification of different types of rock and body (Bieniawski, Z.T. 1989).



Figure 3. Pneumatic Mucking Machine LHD



Figure 4. Miniscoop with bucket capacity of 0.5m^3 type ARAMINE

The *RMR classification* is applicable in the case that the rock mass along a drift is divided into a number of structural regions, i.e. zones in which certain geological feature are more or less uniform (Bieniawski 1989) .

The following six parameters are used to classify a rock mass using the *RMR* system:

1. Uniaxial compressive strength of rock material.
2. Rock Quality Designation (*RQD*).
3. Spacing of discontinuities.
4. Condition of discontinuities.
5. Groundwater conditions.
6. Orientation of discontinuities.

The above six classification parameters are determined for each kind of rock from measurements in the field. Once the classification parameters are determined, the ratings are assigned to each parameter. In this respect the typical, rather than the worst conditions, are evaluated a considerable number of cases with the aim to take into in the consideration all the rock types.

After the evaluations carried out it results that average value of Rock Masses Rating where main mining works mine preparation will be constructed are:

- In I class, "Very good rock" with an average value of RMR 83 about 15% of the rocks are included represented by pyroxenites;
- In II class, Good rock with an average value of RMR 65 about 60% of the rocks are included represented by peridotites
- In III class, Fair rock with an average value of RMR 49 about 12% of the rocks are included represented by dunites;
- In III class, Poor rock with an average value of RMR 36 about 13% of the rocks are included represented by Breche tectonic zone.

In the figure No 5 in the *Bieniawski* diagram a considerable rock estimation are presented. According to the respective classes, considering the roof span and the service time of the mine works, it has been proposed and applied the appropriate support (si fig N^o 6 and N^o 7).

Mine ventilation is a very important challenge because it must be able to withstand a number of important problems related to the deep development of mines .

First, it is important to take into in consideration that the ranges of temperature and pressure caused by variations in elevation and heat transfer produce changes in air density that are in excess of 5 per cent, then analyses that ignore those changes will produce consistent errors that impact significantly on the accuracy of planned ventilation systems.

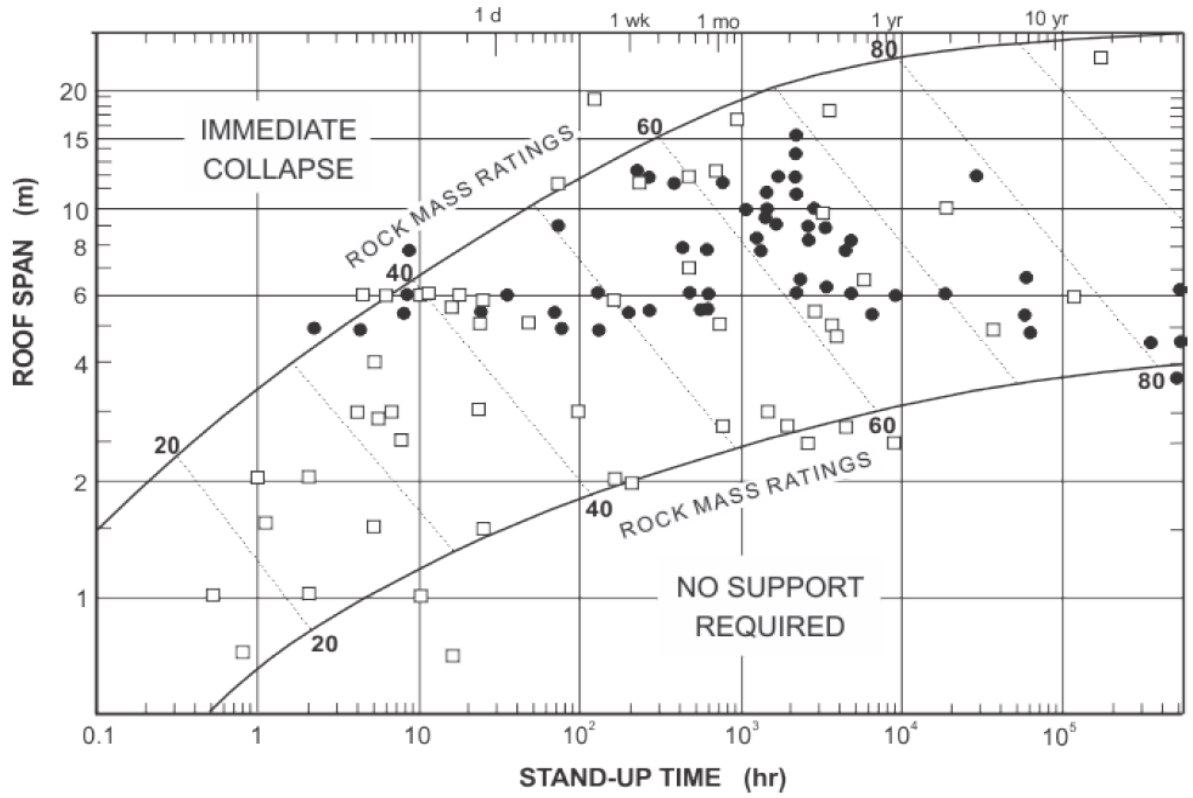


Figure 5. RMR classification of rock masses apply for Bulqiza Mine(Bieniawski1989),



Figure 6. Applying of rock suport in Class IV



Figure 7. Applying of rock suport in Class II

5. MINE VENTILATION

In practical terms, this means that for underground facilities extending more than 500 m below the highest surface connection, methods of analysis that ignore the compressibility of air may be incapable of producing results that lie within observational tolerances of accuracy (McPherson, M.J. (1988).

Different studies conducted in mines related to microclimates and various psychrometric conditions have revealed significant changes in the physical parameters of the air and working conditions in mines (Goskolli E.2006-2013).

As the mining depth increases, it is expected to grow almost linearly in barometric pressure and its value at 1200 m depth is expected to be about 16% larger than that of mining surface.

As the mining depth increases, it is expected to grow, almost linearly, the barometric pressure and its value in the depth of 1200 m is expected to be about 16% larger than that on the mining surface, as well the temperature in the dry bulb is expected to be 30.12^{o}C, that of the wet bulb of 21.4 ^{o}C and air density 1.32 kg/m^{3}(About 20% biger).

Numerous studies and monitoring in mines have shown that, in addition to the factors mentioned above, the further intensification of hydrogen and other gases is also an important problem. In this regard, a series of analyzes and hypotheses have been cast about their origins. Different geochemical interpretations have contributed to the determination of its origin as a product during the process of serpentinization or further oxidation of the fayalite into the tectonic process (Guda V, Goskolli E., 2001).

All of these factors were used for the application of VENTSIM and KLIMSIM softwares for air quantity determination of 132 kg/sec and dimensioning of the main works which serve as entry air way mainly of inner downcast shaft No.9.

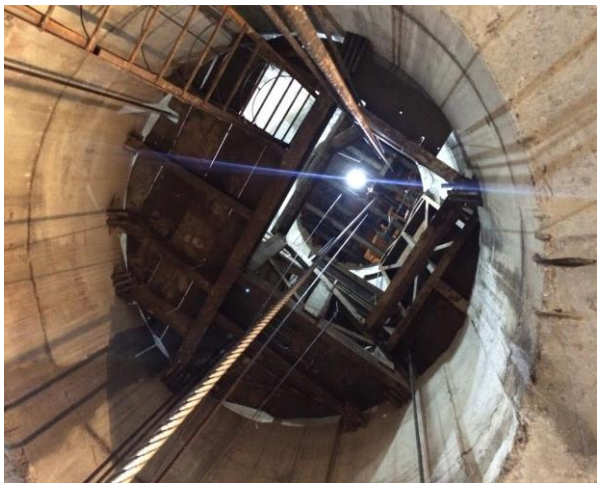


Figure 8. Inner shaft No 9 cross section



Figure 9. Burning of gases at the shaft No 9 forehead

6. CONCLUSIONS

The development of the mine at a profound depth is being faced with a number of important problems which it is impossible to elaborate in detail in this paper. The shaft No 9 is sinking.

Inner shaft No 9 is sinking and together with various mining works are being carried out at different levels. Completing them will enable more detailed studies to be carried out regarding the problems addressed in this article.

Particular attention should be paid to the problems of gas release in the mines and the determination of their various quantitative and qualitative parameters which condition air quantity for the mine ventilation.

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DOI: 10.7251/BMC170701033M

THE POSSIBILITIES OF DEVELOPMENT OF BAUXITE UNDERGROUND EXPLOITATION IN HARDER ROCKS

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ABSTRACT

The paper provides an analysis of the stability of open stopes in the conditions of hard roof during the underground exploitation of bauxite. Roof controling during the ore bodies exploitation is carried out by caving or by uncontrolled open stopes, resulting in increased losses and impoverishment of excavated ore and insufficient safety of the workers and equipment. The main goal is the stability analysis of open stopes to define the natural zone of stability, in the conditions of stability less to choose such a system of support which will be enable excavation of ore bodies in the conditions of stable open stopes.

Key words: sublevel caving method, roof controling, stabilty of open stopes, cavingof immediate roof, rockbolts

1. INTRODUCTION

The application of sublevel mining method is characterized for underground bauxite exploitation. Depending on the physical and mechanical characteristics of the ore and surrounding rocks, and the size and shape of ore bodies, the two main variants of sublevel methods of bauxite is in application [1].

1. Method of sublevel transverse excavation with ore caving, which are applies in the conditions of ore deposits with roof of poor physical and mechanical characteristics, such as marl, marly limestones and clays or in the conditions of hard roof with strong discontinuities. For this variant of mining method the high losses is typical and during the exploitation of ore deposits range from 30% to 50% (g_r=0.3 to 0.5), and increased dilution of ore with waste.
2. The method of transverse mining method with open stopes it is possible in the conditions of the solid and stable roof rocks and ore body of the limited sizes. In the case of larger ore bodies excavation the open stopes are significantly increased causing stresses in the roof which exceed the allowable values, whereby causing caving of the immediate

hanging wall. For this variant of mining method the low losses in ore excavation is characterized which range from 2% to 5% (g_r=0.02 to 0.05), the resulting ore is not diluted with waste rocks, and reduced security due to work in the conditions of uncontrolled ope stope.

Application of sublevel transverse mining method with open stopes are not the result of the planned application with calculated and elaborated technological parameteres, but can be defined as a necessary consequence of the natural characteristics of the deposit. In such circumstances, the safety of work is closely related to solving the problems of technical security at mining method of excavation, whereby the certain working conditions must be established through the adequate technological solutions of method that would primarily enable the excavation with safe roof controling.

The selection of mining method and its parameters with accordance of roof controling process is one of most complex problems in the underground exploitation of deposit, primarily due to the impact of natural characteristics of the deposit, as well as possibility of their accordance with technological demands of mining method. The selected mining method must to satisfy the requirements of safety work and technical security, achieving the favorable technological and economic parameters. In current practice, the application of sublevel mining methods of bauxite has not achieved a satisfactory level of those relevant factors. Caving mining method with sufficient operational safety and high losses during the ore excavation characterize unfavorable technological and economic indicators, while the mining method with open stopes characterize satisfy ore recovery of deposit which is reflected in a favorable technological and economic effects, but the work and technical safety is not at the required level.

Research on the mining technology impovement and technical safety in conditions of the hard roof in deposit, based on detailed study: the natural characteristics of the deposit, stability of stope and manner of roof controling.

2. NATURAL CHARACTERISTICS OF DEPOSITS

Deposit of limestone type (deposits in harder rocks) are located in a spacious limestone zone of the Dinarides, and were originated by filling of karst paleo relief in limestones, rarely by dolomites. The bauxite deposits of West Bosnia (Jajce, Mrkonjic Grad and Bosanska Krupa) belong to the Cretaceous sediments and for this type is characterized that deposit is composed by number of smaller grouped bauxite ore bodies, irregularly shaped and limited reserves. The bauxite deposits occur in sinkholes depressions of Cenomanian limestones, through which tresgressive lie the flysch sediments of the upper Cretaceous [2]. Floor limestone is pale reddish and yellowish, well bedded by layers with a thickness of 0.2 to 0.7 m. Hanging wall of these bauxites is represented mainly by the flat surfaces made of limestone. Immediate hanging wall usually composed by calcarenite, marls and conglomerates, red and bluish sandy marls, white limestone and breccias. In the hanging wall sporadically occur the clays, marls, marly limestone and similarly.

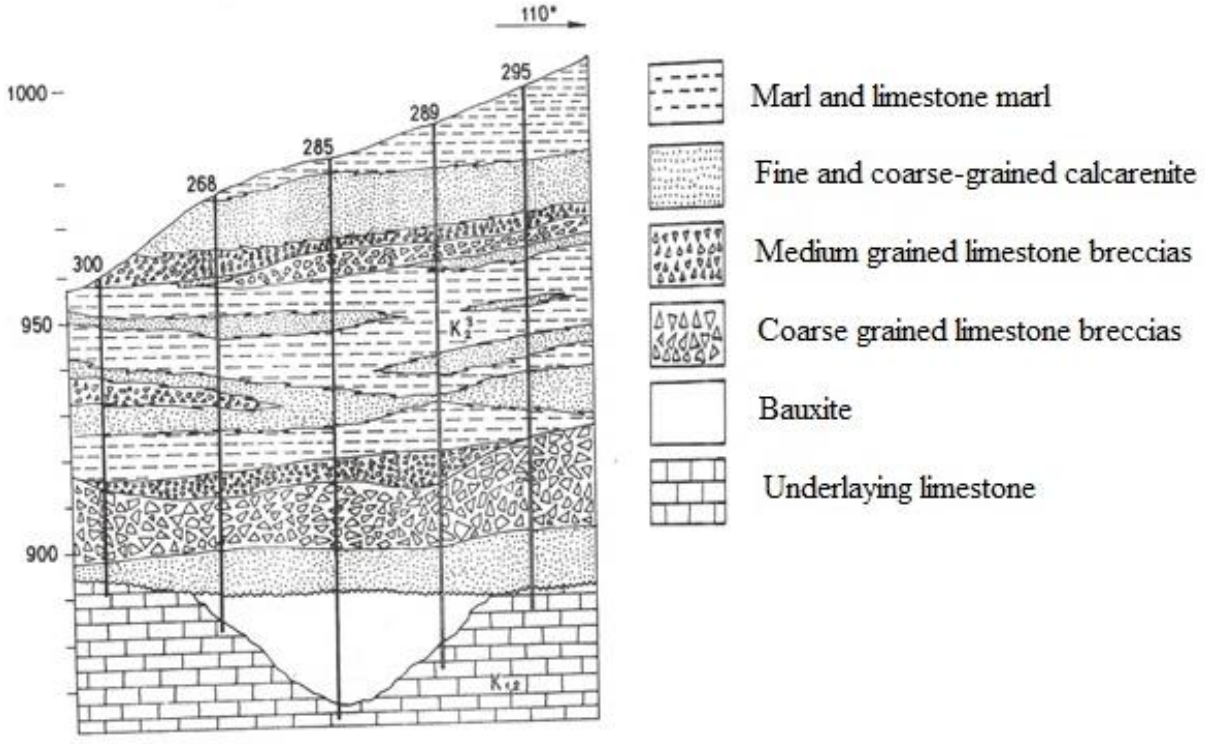


Figure 1. Characteristic geological profile of the ore body

The basic features of tectonic relationships are caused by its regional geotectonic position within the Dinarids. This area represents the parts of a large laid anticline which is broken by the numerous vertical and subvertical faults. The vertical and subvertical faults of lower range as products of post ore tectonics significanty intersected the bauxite structure, and are presented in the ore bodies which significantly cause to the process of roof controling due to the exploitation of deposit.

Determination of the geotechnical characteristics of the floor rocks, bauxite and certain lithological parts of hanging wall was carried out on the samples taken from the deposit or from drilling samples and the average values are given in Table 1.

Table 1. The average value of the geomechanical characteristics

Material	γ (kg/m^3)	E (MPa)	σ_p (MPa)	σ_z (MPa)
Floor limestone	2633	4163	34,93	3,5
Bauxite	2718	4939	44,56	5,3
Calcarenite	2630	5212	66,71	5,3
Fine grained breccia	2852	4315	62,55	6,5
Coarse grained breccia	2861	3763	51,25	5,9
Marls	2916	3576	25,32	1,4

3. ANALYSIS OF STABILITY OF THE OPEN STOPES

Research on improving the technology of excavation is carried out in two main directions: reducing the mining losses of caving excavation method and determination of stability conditions of open stopes in deposits with the harder overlaying sediments. Defining the zones of open stope stability is based on the calculations and stress trajectories and strain around of open stope for different conditions and situation that result during the deposit excavation, and is performed on the mathematical models formed according to the characteristic geological profile of deposit. The stability conditions is based through the function of structure and physical and mechanical characteristics of deposit and surrounding rock mass, and the size and spatial position of the open stopes. The research of open stope stability primarily depends on the proper interpretation of structural and geological characteristics of rock mass of immediate and basic hanging wall and identification and monitoring of the phenomenon of blocks and wedges that occur in the immediate hanging wall [2]. For this purpose is necessary to apply some of the methods for stress-strain analysis, and accordingly to that applied the finite element method and software package Phase^2[3]. Numerical approach to estimate rock mass behavior around open stope in the bauxite deposits, according to *GSI* classification and the finite element method (FEM). *GSI*, as a system of classification of rock mass for the purposes of rock mechanics, in particular can serve as a basis for input parameters in numerical analysis. This parameter with a visual assessment of the rock mass is a direct basis for the choice of parameters relevant for strength and strain rock mass estimation [4].

According to Palmström and Nilsen (2000), numerical modeling of rock mass represents a discretization of the rock mass in a large number of the individual elements. The higher the degree of disretization or more components the solutions of these methods are closer to the solutions of the analytical methods [5].

Phase^2 is a program for calculation of stresses and strain around of underground openings using 2D elastic and plastic finite element method. The program is used to solve various problems in the mining, where it is possible to monitor and analyze the change of the stress state due to stope development (subphases). The software package analyzed the problems of two-dimensional plane (2D) strain state. To describe the properties of rock material used Mohr- Coulomb model, which defines the shear strength by cohesion, normal stress and the angle of internal friction.

Missing data for Poisson coefficient (μ) and tensile strength (σ_z) were determined empirically on the basis of literature data. The values of friction angle (φ) and cohesion (c) were determined by using Hoek-Brown's failure criterion.

Table 2. The parameters of particular lithological members by Hoek- Brown criterion

Material	*GSI*	m_i	*D*	*Frictional angle* φ (^{o})	*Cohesion (kPa)*	*Poisson's ratio* μ
Floor limestone	70	12	0.7	57	5500	0.39
Bauxite	60	10	0.7	52	8130	0.28
Calcarenite	70	18	0.7	53	11250	0.34
Fine grained breccia	70	20	0.7	48	9500	0.21
Coarse grained breccia	70	18	0.7	56	6250	0.13
Marls	60	15	0.7	51	5620	0.26

Based on the characteristic geological profile of ore body (Figure 1.) formed a mathematical model, as shown in Figure 2.

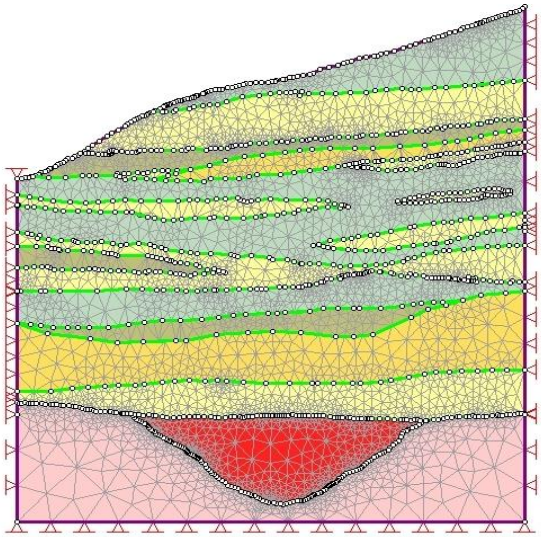


Figure 2. The mathematical model for finite element meth(

On the formed mathematical model for different sizes of open stopes (30, 50 and completely excavated first bench about 75 m) defines the state of strain and stress and is determined by the stability of the open stopes.

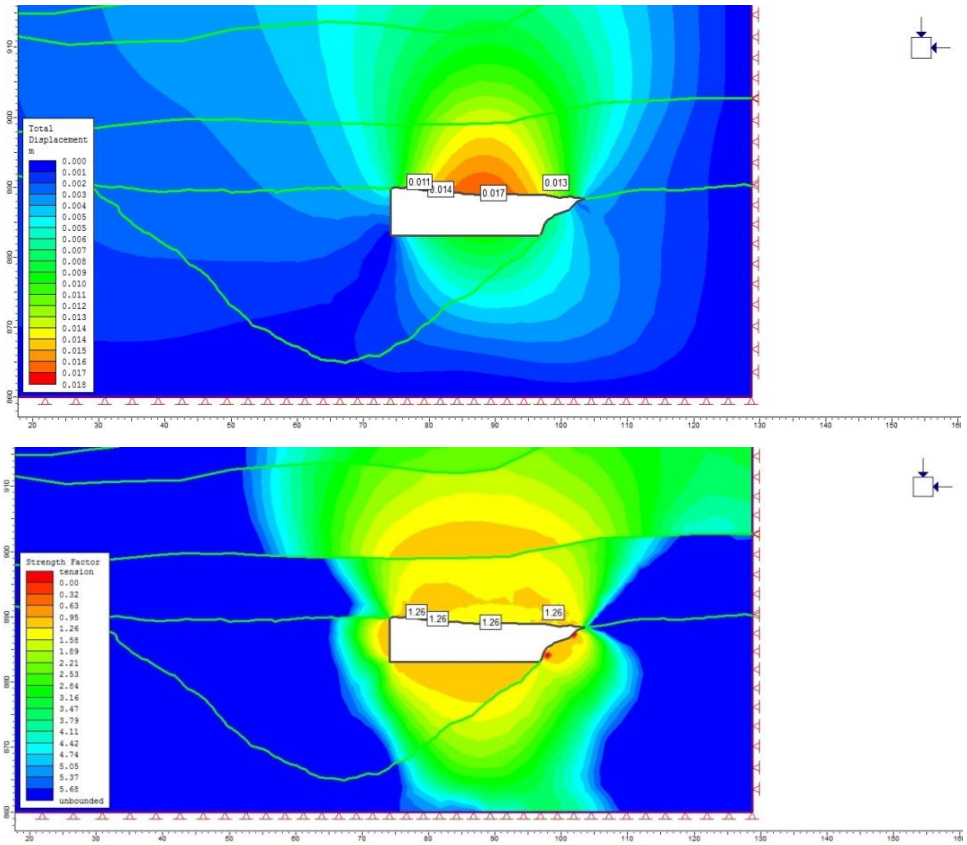


Figure 3. The results of interpretation of deformation and safety factor of rock massif around open stope length of 30 m

For the stopes of the size to 30 m is characterized the appearance of tensile stresses in the zone of immediate hanging wall, however, these stresses are not in intensity such that it can cause the caving of immediate roof rock mass, which is visible through the factor of safety, which in this case is $F_s \geq 1 (F_s = 1{,}26)$. Based on these results we can to conclude that for the analyzed ore body the open stope space is stable up to a length of 30 m.

With the increasing size of the open stope area significantly reduces the stability of immediate hanging wall, and are evident the zones with a larger deformation of rock mass and the trajectories of small values of the safety factor, which means that in case of open stope areas larger than 30 m can be expected caving of the immediate hanging wall. On Figure 4. are presented the results of strain and safety factor of rock massif around open stope length of 50 m.

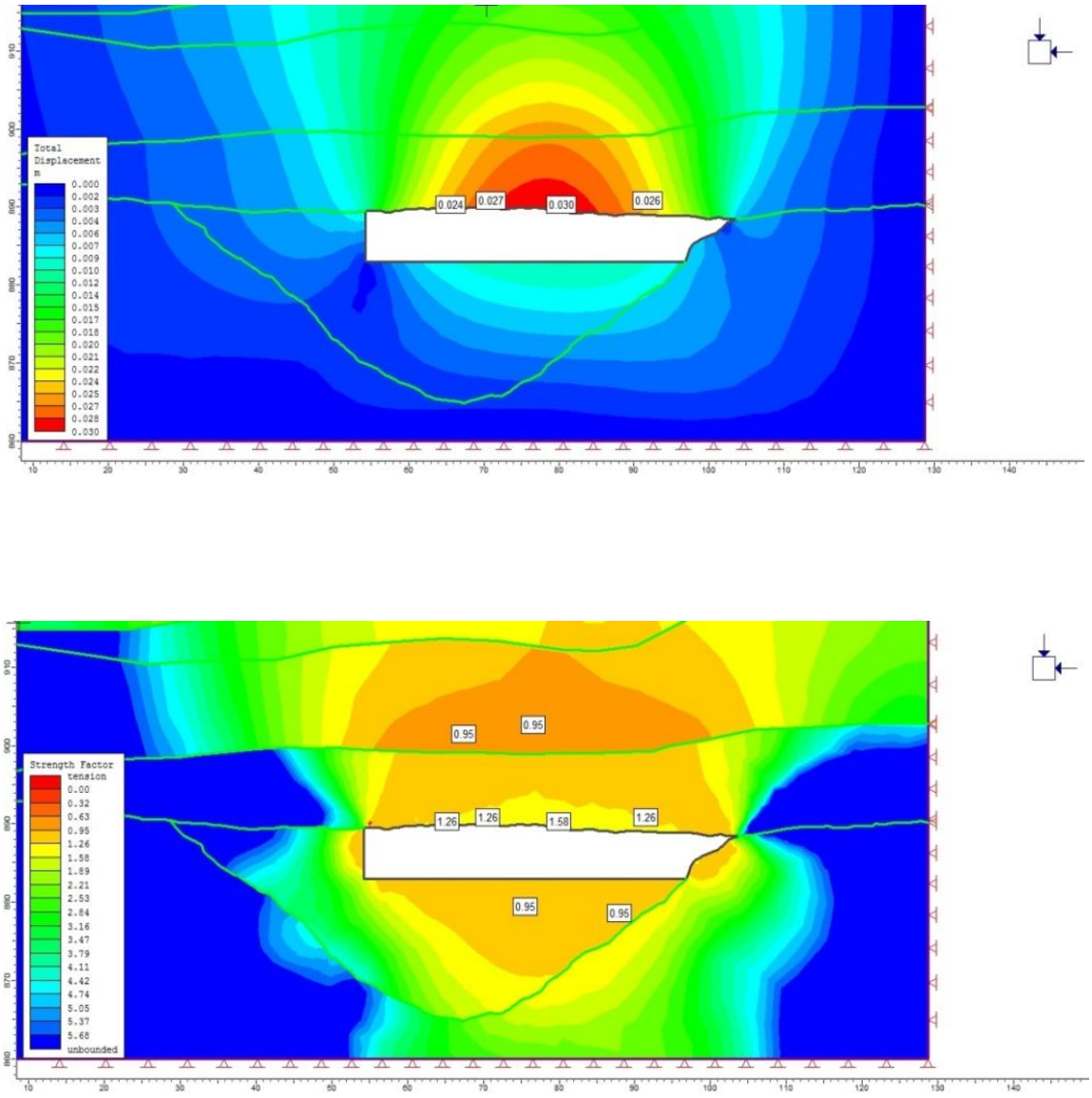


Figure 4. The results of interpretation of deformation and safety factor of rock massif around open stope length of 50m

According to the results of analysis for the case of elastic behavior of rock mass movement of rock material is present in the hanging wall of stope, and in the foot wall but in significantly less range. The strain contours are distributed in proportion that there was a gradual sliding of material, while in case of plastic behavior of rock materials there was a strong movement of rock materials into excavated stope with strong shear and tensile stress.

Analysis of stress and strain in case of excavated the first bench of the ore body, as in this case the size of the open stope is about 75 m shows evident deformation of the rock mass in the immediate hanging wall, with strong shear and tensile stress, as shown in Figure 5.

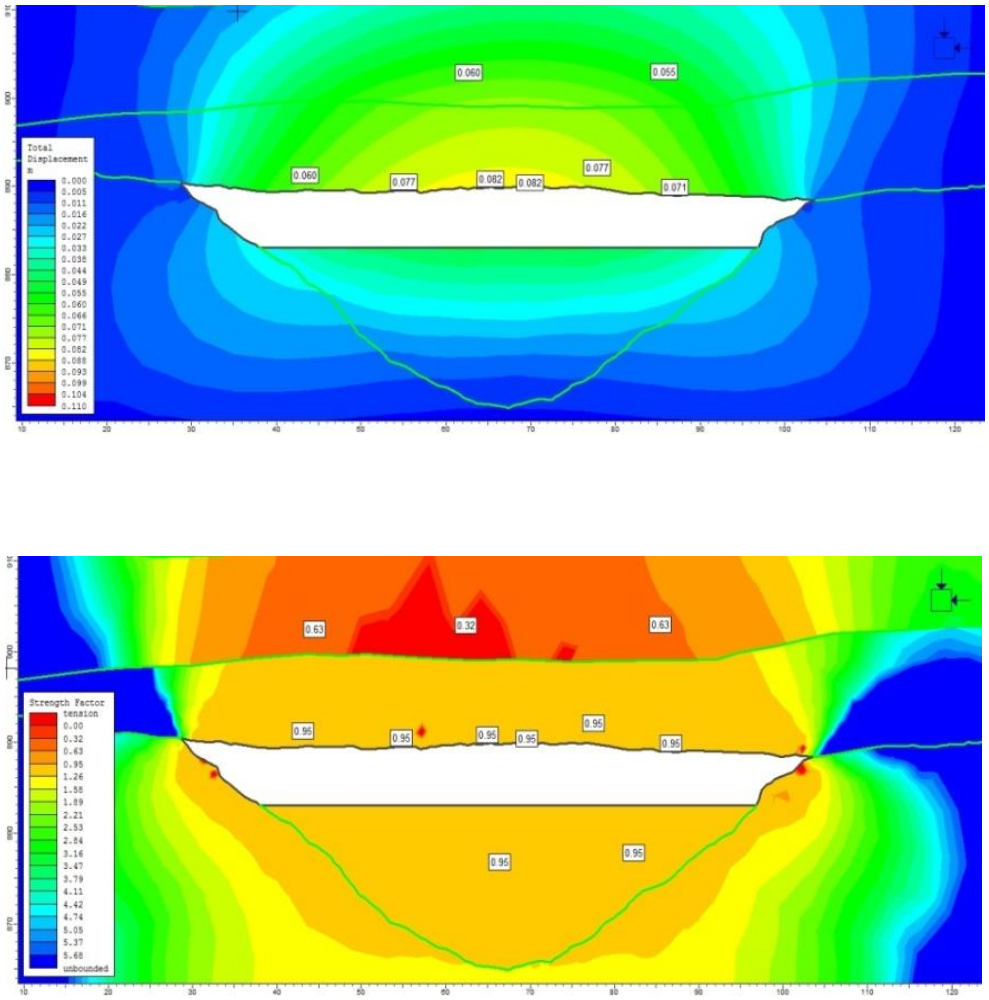


Figure 5. The results of interpretation of deformation and safety factor of rock massif around open stope length of 75m

For the size of the open stope of 75 m, that is completely excavated the first bench, characteristic is the typical occurrence of the tensile stress zone in immediate and basic hanging wall, this strain in intensity are such that will surely result the roof caving, which is visible through the safety factors, and in this case is in range of $F_s<1$($F_s=0,94$).

4. ROOF CONTROLING

The basic aim of roof controling is to balance the disturbed rock mass by excavation process and the basic task is solving the problem of underground pressure in the underground openings. According to the results of previous studies of open stope stability it can be concluded that for the analyzed ore body the open stope is stable up to a length of 30 m, after what enters into the zone of instability, or hanging wall caving above the excavated stope. On the size of tensile stress in immediate hanging wall,what are the main cause of hanging wall caving, the size of open stope and geo-mechanical characteristics of the rocks in the hanging wall of the ore body have the main influence.

To stabilize the large open stopes mostly used rockbolting of hanging wall with the rockbolts of steel cables installed in concrete [6], the so-called "cable bollting" system. The procedure of rockbolting is consisted of drilling the holes by selected diameter and length in the immediate hanging wall of ore body spaced on the calculated distance, then into the holes injected cement and water of certain water-cement ratio and steel cable. This method of rockbolting has certain advantages:

- Ability to instaling the long rockbolts, which allows flexibility of the steel cables, in the relation to the solid performance of the other rockbolts, due to limited working space in the undeground mine.
- Total space availability between the cables and the surrounding rocks with cement what contibutes to a better quality of the installed rockbolts.
- Easy procedure of rockbolting with the possibility of complete mechanization.

In the analyzed mathematical model was installed the rockbolts the average length of 17 m, at a spacing of 6.5 m and the results are shown in Figure 6.

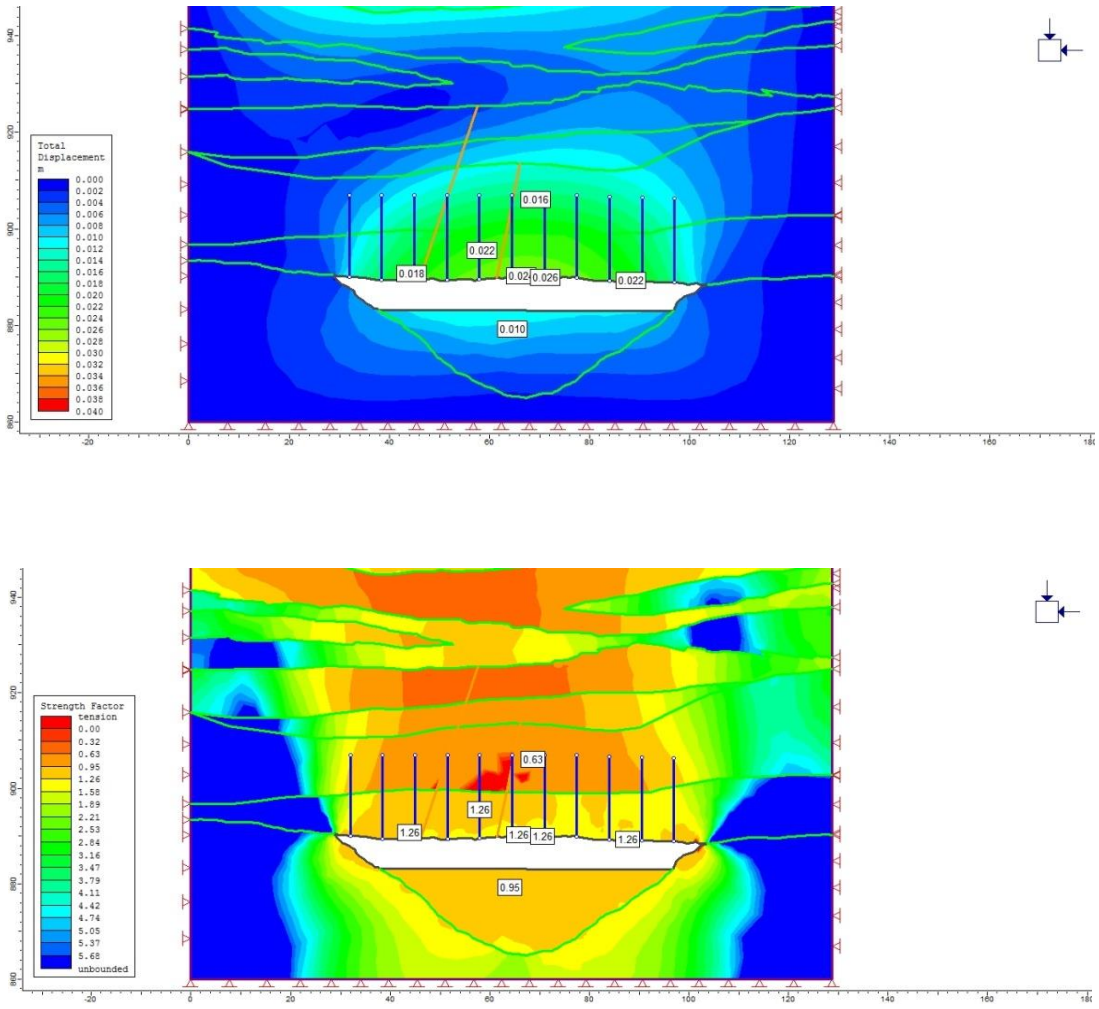


Figure 6. The results of deformation and safety factor of rock massif and installed rockbolts around the open stope length of 75m

5. CONLCUSIONS

In the conditions of harder rocks in the hanging wall, the open stopes are stable at a short distance, where is possible to successfully apply the method of excavation with the open stopes, which provides high ore recovery from deposit and negligible dilution of obtained ore. With the increasing size of open stopes tensile stress and strain of immediate hanging wall are appeared what result caving and continued work on the excavation by caving mining method, which is characterized by high ore losses in excavation process.

According to the research on the mathematical model has has been proven that by application of the system of hanging support (rockbolts) can to achieve the sufficient stability of open stopes. For the proper installation process of rockbolting for stability of open stopes is necessary to:

- Detailed research and analysis of the natural characteristics of each ore body.
- On each ore body to perform analysis of the size of stresses and strains for different sizes of the open stopes,
- On the mathematical models to analyze the rockbolts of different length and spacing, in order to achieve the optimal values of the elements of hanging support.

- Also with the development of mining works set the gauges to monitor the actual deformation of hanging wall, in order to correct the results obtained by mathematical model.

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DOI: 10.7251/BMC170701043I

NEED FOR ACTIVATION OF COAL LEDGE "POLJANA" BY UNDERGROUND EXPLOITATION

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ABSTRACT

Underground coal mining in active ledges in Serbia is carried out from 50 to 158 years and most of them are running out of reserves, which necessitates finding replacement capacity.

The opening of new mines or activation of promising ledges stems from the position of coal as a domestic energy source for electricity production, raw materials, mining traditions, the workforce and the possibility of producing most of the mining equipment in the country. The State should recognize their interest in this production activity, through the employment of their own labor, reduction of import dependence of energy, regional development of underdeveloped areas of eastern, western and central parts of Serbia, to prevent migration of the population to larger urban centers and the development of supporting industries (mechanical, electrical and chemical) to the mining industry.

Through a review of available coal reserves for underground coal mining in the Republic of Serbia, the initial elements are state of research and the level of awareness of the natural and geological conditions in specific ledges. According to available data, the potential ledges where underground exploitation could be restored are Despotovac basin (Kosa-Zabela), West-Moravian basin (Bajovac), Melnica, Poljana, Aleksinac ledge, Dragačevo basin.

Based on a versatile analysis of a set of available data, this work is made. This work has an aim to present to the professional community the need for activation of some of the new ledges with higher coal reserves and possibilities of application of modern technological solutions for underground exploitation. In this regard, this work analyzes the basic elements of the opening and underground exploitation of coal in ledge "Poljana".

Key words: coal, underground exploitation, coal mind

1. INTRODUCTION

Coal mines with underground exploitation in Serbia currently are characterized by low production, low level of mechanization and a high share of hard physical labor, which, in the total balance, gives the unfavorable financial effects of the business. In underground coal mines in the Republic of Serbia is conducted the exploitation of the stone coal, brown coal and lignite coal, which is mainly used for energy purposes.

Nowadays, all the raw potential of the Republic of Serbia belongs to underground coal mines, and according to its natural and to geological conditions is predisposed for the system of underground exploitation. In table 1 is given the condition of balance reserves of coal in active ledges – in open pits.

Table 1. The condition of balance reserves of coal in active ledges – open pits

Ledge – pit	Kategory			
	A	B	C1	A+B+C1
Mala Čuka	22.520	678.370	774.040	1.482.930
Jarando	-	875.270	-	875.270
Tadenje	-	401.560	-	401.560
Progorelica	-	1.211.280	-	1.218.280
Total stone coal:	22.520	3.166.480	774.040	3.963.040
Strmosten	41.000	2.013.500	1.706.940	3.761.540
Jelovac	123.840	1.085.440	516.600	1.723.880
Senjski Rudnik	201.890	144.340	192.480	538.710
Ravna Reka	-	998.500	926.220	1.924.720
Istočno polje	54.020	1.364.940	575.500	1.994.160
Soko	468.850	15.215.560	41.887.590	57.572.000
Jasenovac	-	761.100	-	761.100
Total brown coal:	889.600	21.583.480	45.805.030	68.278.110
Lubnica (total)	644.730	9.734.790	506.080	10.885.600
Štavalj (total)	1.677.710	104.799.080	80.149.390	186.626.180
Total lignite coal:	2.322.440	114.533.870	80.655.470	197.511.780
TOTAL:	3.234.560	139.283.830	127.234.540	269.752.930

These reserves are geological reserves and should be reduced by exploitation losses and losses during processing. Also, the situation is additionally complicated by the fact that in this Elaborate on coal reserves, reserves are counted for layers thickness of 1m and up, and more accurate counting would be if we take layers with a thickness of 2 m or 3 m and up. With the aforementioned deductions, condition of the coal reserves in the current open pits is considerably less. Taking into account the coal reserves, the development of underground coal mining in Serbia need to be directed toward mechanization and modernization of technological processes in active coal ledges with large reserves and activation of some of the new ledges. Now, it is necessary to bring to closure, step by step, active ledges which are running out of coal reserves.

The level of awareness of the natural and geological conditions in perspective ledges is quite different, but generally, still completely insufficient for making decisions on the possibilities of application of modern technology of exploitation, in the majority of ledges, except for ledge "Poljana", which is explored in detail.

According to the available documentation, currently available coal reservesin inactive ledges are in the amount of 297 million tones, shown by ledges:

Table 2. Coal reserves in the inactive ledges

Ledge	Coal types	Reserves (t)
Melnica	brown	34.780.000
Poljana	lignite	62.100.000
Despotovac	lignite	24.800.000
Zapado moravski basen	brown	93.900.000
Aleksinac	brown	27.000.000
Dragačevo (Tijanje)	lignite	55.000.000
TOTAL	-	297.500.000

*The table shows only some larger ledges where researches were carried out and where the studies about coal reserves were done.

By professional consideration is estimated that the naturally-geological conditions inthe coal ledge POLJANA, from the point of mining, are favorable and that they allow the use of the highly mechanized excavation which is proviso for achieving a high production capacity.

As for the ledge "Poljana", there, mining activities began since 1876, when, in this area, near the village of "Dubrava", started with work mine "Kamenac".

The mine itself, if looking the range of production, was small and was of local significance and the mine worked until the beginning of World War II.

In 1956, Geological Institute of Belgrade is working on a geological map of the wider area, which again stimulated interest in ledge "Poljana". Already in 1957, geological exploration drilling started and they will last until 1963. New structural, geo-mechanical and hydrogeological drilling were carried out from 1981. to 1982.A new level of geological exploration of the ledge in 1982. caused acreation of the Elaboration categorization and classification of coal reserves of the ledge "Poljana". According to the mentioned Elaborate, coal reserves have been classified into category B and C1.How the investment and technological documentation is based on the balance reserves of high-class, A + B, in accordance with the law of the Mining and geological investigations, further discussion and analysis of coal reserves of the ledge "Poljana", refers only to the proven coal reserves of category B, determined by exploration drilling.

2. BASIC CHARACTERISTICS OF THE COAL LEDGE "POLJANA"

The ledge "Poljana" belongs to coal zone of Podunavlje area, respectively, to the southern part of the Kostolac coal basin, which (in a broader sense) covers an area of about 600 km^2, in the area between the river Danube in the north, the river Great Morava in the west and the river Resava to the south, while in the east is restricted by crystalline Rama. The basin is located about 90 km from Belgrade, while the ledge "Poljana" is located on the south side of the city Pozarevac, (Figure 1). It covers a relatively narrow belt of Pozarevac Greda, from Pozarevac to the village Šljivovac in distance of about 9 km, and between villages Lucic and Poljana in the west, and Toponica and Kravlji Dol in the east. Teritorially and administratively, it belongs to the Podunavlje region, Municipality Pozarevac, respectively, it belongs to cadastral municipalities of Pozarevac, Lucica, Salakovac and Popovac.

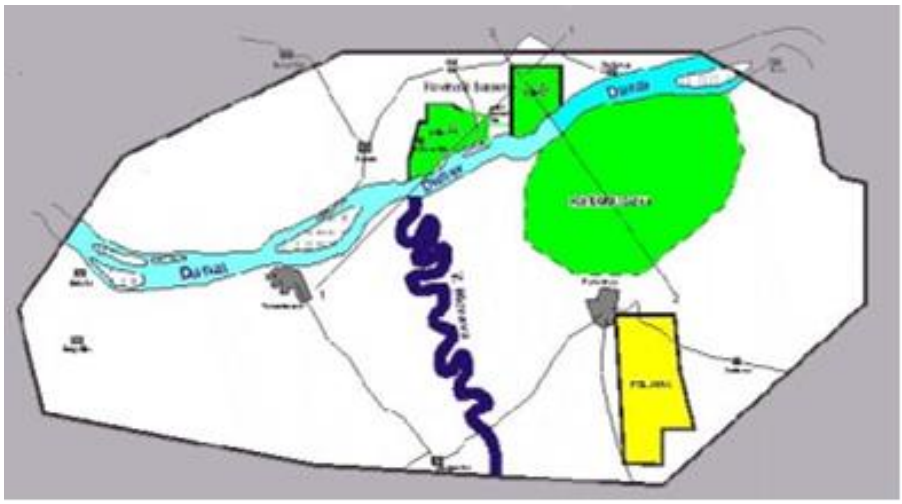


Figure 1. Geographical position of the ledge "Poljana"

The wider area of the ledge "Poljana" belongs to the Pomoravlje and to the valley of the river Mlava, with Pozarevac Greda which extends between them. It is included by the basic geological map, sheet of Pozarevac, 1: 100.000 and is located in the northern and in the central part of the sheet. According to the Basic Geological Mapand according to the interpreter of the map (Malešević, 1978, 1-2), into geological structure of the ledge "Poljana" enter the following stratigraphic members:

- Upper Miocene (Panon);
- Quaternary (Pleistocene and Holocene);
- Upper Miocene (Panon).

In lithologic view, Panonis represented by clay-sandy sediments with fauna (lower and middle Panon) and productive series (upper Panon).The lower and middle Panonic sediments in the ledge "Poljana" are not securely separated from the upper-productive Panonic sediments. Lithology productive series - Productive series is represented by the clay-sandy sediments with two coal layers, the lower A and the upper B. Picture 2 shows the lithostratigraphic position of both coal layers in the ledge "Poljana". Immediate underlying stratum of the main coal layer (A), for the most part, is composed of yellow and gray sands, which may be, to a lesser extent, clayey. In places located in the immediate underlying stratum is possibility of occurring fatty clay, thickness of up to 0,5 m.

Thickness of underlying stratum sands, fortified on the base of results of exploration drilling, is over 65 m. Coal horizon contains two coal layers with interlayer mullock, thickness from 5m to 25 m, usually from 10 m-15m. Exploitable is only the lower coal layer, while the upper coal layer is without greater economic value (unstable thickness, interlayer mullock, poor quality). Interlayer mullock is made of green and blue sandy clay and the clay with layers of coal and coal clay or green and blue sand. Thickness of the lower coal layer thickness is up to 7 m, mainly homogeneous, and rarely is containing coal clay. The upper coal layer has a complex lithological composition and is layered by clay and coal clay on two branches: the lower one (thickness up to 2.3 m) and an upper one(thickness up to 3.3 m). In the roof seam of coal horizon are laying gray and green sands, sandy clays and clays with calcareous concretions and very rarely coal clay. Roof seam sediments are reaching a thickness of up to 70 m, but in very large amount are eroded, and sometimes completely absent. They are covered by quaternary deposits, loess deposits of marsh and Eolic origin of thickness up to 35 m, and alluvial and diluvial sediments.

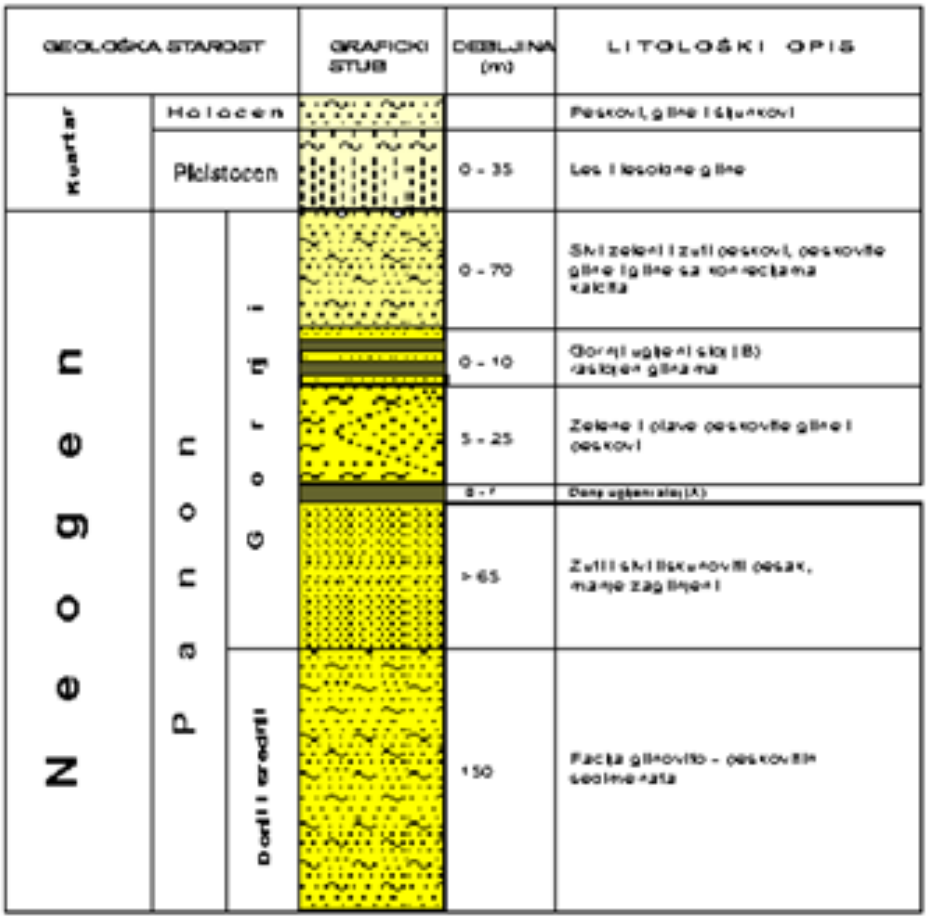


Figure 2.Lithosratigraphic pillar of the ledge "Poljana"

Gently waved surfaces of the coal layer A is generally extended in direction of NE-SW and is tilted at an angle of 5-10 ° to the north-west. The ledge spreads in the direction of fall of the layer, which means to the north-west and is sinking under the alluvium of the river Velika Morava. The northern border of the ledge, at this stage of investigation, remains open for further geological research. By current exploration activities, determined maximum width of the ledge is 3 km, and was found in the central part of the ledge. With the change in the morphology of the terrain, also changes and depth of geometry of coal layer A. The lowest is in marginal parts of the ledge and is only a couple of dozen meters, whereby, by the erosion, is affected larger part of the roof seam, with coal layer B. The maximum depth of exploitable layer is where the skerry is the highest, respectively, in the central parts of the ledge, along its longitudinal axis and is up to 120 m. In the southern part of the layer, the highest elevation is 156 m and 60 m is in the north. Thickness of the coal layer A is the largest in the central parts of the ledge and is ranging, generally, between 4.5m and 5.5 m (maximum 7 m). Slightly is increasing from south to north. The average thickness of the main coal layer for the whole ledge "Poljana" is about 4.6 m. The coal layer is a mostly homogenous structure, rarely stratified with coal clay thickness of 0.2 m, less frequently up to 0.5 m. Fugure 3 shows structural-geological profile of the ledge "Poljana".

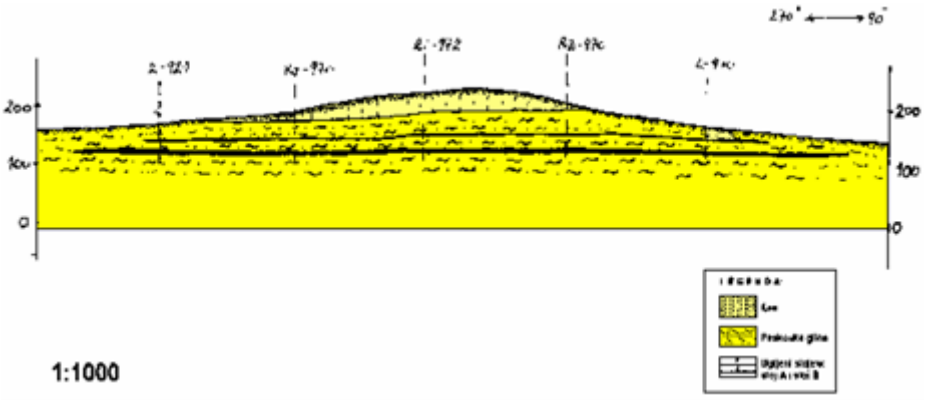


Figure 3. Structural-geological profile of the ledge "Poljana"

Structural-geological profile of the ledge "Poljana" (figure 3) shows a simple structural-tectonic structure of the ledge "Poljana".

Reserves of coal in the ledge "Poljana" were established by Elaborate of classification and categorization of reserves with balance on 31.12.1982., which the Commission for the verification of the reserves of solid mineral raw materials and groundwater of Republic Serbia adopted and confirmed reserves of category B and C1 of about 62 million tons, of which about balanced 59 million tons (Table 3), wherein prevail reserves of category B.

Table 3. Reserves of coal in the ledge "Poljana" with balance on 31.12.1982.

Category	Reserves (t)		
	Balance	Off-balance	TOTAL
B	48.467.000	2.018.322	50.485.326
C1	10.527.590	1.166.017	11.693.612
B+C1	58.994.590	3.184.339	62.178.938

Lower heat of combustion of coal with delivery humidity is between 8.14 and 12.64 MJ/kg, depending on the gangue impurities in the coal layer. Average lower heat of combustion is 10.45 MJ/kg. Upper heat of combustion of coal with delivery humidity is 8.51 to 14.23 MJ/kg, depending on the presence of gangue. Average upper heat of combustion is 11.96 MJ/kg.

3. EVALUATION OF NATURAL CONDITIONS OF THE LEDGE AND POSSIBLE EXPLOITATION SOLUTIONS

In accordance with the position and isometrics of the main exploitable coal layer and the ledge itself in general, was adopted the division of exploitable areas on two possible manufacturing pits, as follows: the pit "Vinogradi" and the pit "Bresje". According to this division, to the pit “Vinogradi” belongs the south-eastern part of the ledge with balance reserves of 21.166.800 tons of coal, while the pit "Bresje", which covers the north-western part of the ledge has 21.825.480 tons balance reserves of coal. On the choice of such opening conception of the ledge "Poljana", on two separate manufacturing pits, a crucial influence had hydrogeological structure and its relations in the ledge. Contour line of the coal seam in the floor +80 m represents a characteristic line of the hydro-geological structure of the entire

ledge. Namely, to that elevation, according to many studies, reaches the maximum level of underground water in the ledge, while the elevation +81m represents the lowest level of the ground surface above the ledge. The spatial position of the contour line+80 m approximately coincides with the already designated division of the ledge on two pit facilities and is representing their artificial border. The part of the ledge north of this contour line belongs to the pit "Bresje" so called because of the surface locality. This part of the ledge, at the same time, is below the characteristic hydrogeological line - contour line of the coal layer in the floor +80 m. To the pit "Vinogradi", also named after the locality of the surface above this part of the ledge, belongs the other half of the ledge, which is located south and is above the distinctive contour lines.

By created investment-technical documentation is envisaged that the ledge should be opened with three underground mining premises which brings to rational solvation of transportation, drainage and ventilation of the underground system. Creating premises should be done mechanized, with cutting machines and combined supporting, steel and concrete support. For excavation is envisaged the use of mechanized wide forehead methods, wherein in the work process, at the same time, will be two digging units, and depending on the height of excavation, projected production capacity is from 700.000t to 1.000.000 tons/year. In such a manner should be dug 34.5 million tons of coal in the ledge, while remaining reserves should be dug by other methods.

To achieve the expected capacity of the mine, the analysis of the required investment in the pit and on the surface is made, and in the case of commencement of excavation in the part of the ledge, the pit "Vinogradi", is amounted to 45 million EUR. Commencement of exploitation-excavation is expected within 26 months from the commencement of construction of the mine, when finishes montaging of equipment for wide forehead and starting trial operation and the running which will last 10 months, which means that the period of construction of the pit "Poljana-Vinogradi" is three years.

The mine should employ more than 600 workers, and the same would be deployed from existing mines that will be closed due to the exhaustion of coal reserves. According to the indicative budget, operational production costs would be 3 EUR/GJ, and these costs are changeable.

4. CONCLUSION

According to the abovementioned, undoubtedly fact is that in order to develop the underground coal mining industry, as an economic sector, is necessary to modernize technological processes of exploitation in active ledges with larger access to coal reserves and access to the planned activation of some, now, inactive ledges.

The level of awareness of the natural and geological conditions in these ledges is quite different, but generally still completely insufficient for making decisions on the possibilities of application of modern technologies of exploitation. Exception is the ledge "Poljana" which is satisfactory explored.

In the next 10 years, the coal reserves in most ledges which are active now will be depleted, and a longer period of exploitation can be only in the pits: "Štavalj", "Soko", "Lubnica" and "Strmosten".

Survival and development of underground coal mining in Serbia is directly dependent on the opening of a new mine, on the activation of coal ledge "Poljana", so is necessary to direct the activities of PE UCM and the line Ministry to this direction. Practically, annual production of coal from the mine "Poljana" is projected at a level that is higher than total production of all 11 pits which are in composition of PE UCM now.

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DOI: 10.7251/BMC170701051B

MINING OF GOLD IN THE APUSENI MOUNTAINS DURING THE MIDDLE AGES

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ABSTRACT

The paper presents the organization and legal framework of gold mining in the Apuseni Mountains in the Middle Ages. At that time, the miners enjoyed a series of privileges granted by the Transylvanian voievodes or Hungarian kings. Mining technology refers to tools and installations intended to transport ore and sterile in the galleries, water evacuation, ventilation of workplaces and grinding the auriferous ore.

Key words**:** *gold mining in the Middle Ages, organization of mining, mining methods.*

1. INTRODUCTION

As it is known, the Romans captured from the Dacians 165.5 ton gold and 331 ton silver, and in the 165 years of occupation extracted an additional 500 ton gold and 950 ton silver. After the Romans withdrew from Dacia, the entire social-economic course of life went back to its rural character. Economy turned back to growing sheep, to extensive agriculture, craftsmanship declined, acquiring a predominantly household character. Mining was no exception. The local population used metals to make agricultural tools, arms and household goods. Auriferous mining in the Apuseni Mountains was probably limited to exploitation of alluvium gold along rivers, and exploitation of meagre seams, close to surface, that were not worth to the Romans. It is hard to believe that the miners left behind would have abandoned their occupation and would have turned to shepherding; agriculture couldn't have been an option. Gold and silver was potent money in those unstable times as well, about which we know little.

Mining technology of the Romans was so advanced, that in the following 1500 years remained almost unchanged, not only in our country but in Europe as well. In our country, barbarian incursions stopped the Daco-Roman civilization for almost a millennium, the so-called dark millennium having been instated.

2. MINING ORGANIZATION [3, pp. 178–180]

Exploitation of precious metals in the Apuseni Mountains restarted in the 8th century, after the Slav people having been assimilated, the proof being *Zlatna* toponym, which replaced the name Ampelum, given to this ancient mining centre by the Romans, *zlato* meaning gold in Slavic. No documents are known referring to the social-economic organization of the people in the auriferous region of the Apuseni Mountains in that period, but one can assume that they lived according to the ancient customary law. Alongside with the pre-state feudal structures (princedoms and voievode's domains), the interest towards mining regions increased, they became feudal domains, and their inhabitants became serfs to the crown. Over these pre-state structures within the Carpathian Arch, with emerging rudimentary organizations of life, the Hungarians stormed in. From the 10th century on, the Romanian territories began to be enclosed in the medieval kingdom of Hungary, which adopted a policy of colonization, settling the Szeklers in the east of the province, and the Transylvanian Saxons in the south and around Bistriţa. The colonists' mission was defending the borders and putting to good use the economic resources of the autonomous voievode's domain. In the year 1210 the occupation of Transylvania by the Magyars was finished by conquering Ţara Bârsei.

The first written documents appeared in this period, establishing many aspects of the social-economic life of the period, among which mining activity. Such a document is the *Letter of Privileges* given by King Bela the Fourth of Hungary (1235–1270), in the year 1255, for the miners of Besztercebánya (Banská–Bystrica from today), granting them a series of privileges. The document is also important for the auriferous mining in the Apuseni Mountains, since German miners from Besztercebánya and Körmöczbánya were brought to Zlatna, Abrud and Bucium. They kept the rights given by Bela the Fourth and fought for a long time to maintain them. The *Letter of Privileges* of 1255 states that the miners of the area mentioned above had economic, legal and religious advantages, based on which they could run a more sustained economic activity, compared to the other inhabitants. They could prospect and exploit noble metals underground and along the rivers, but they had the obligation to pay to the regal tax administration the eighth part of the gold and the tenth part of the silver extracted, and in case metaliferous seams had been discovered, the king could take possession of the respective terrain. Similarly, the miners were exempted from customs taxes for goods of strict necessity and financial and military obligations to local nobility. The privileges were acknowledged and reconfirmed by Bela the Fourth', Laszlo the Fourth' (1287), and Andras the Second's (1291) heirs. We should also mention that Bela the Fourth had previously granted the same privileges to the Saxons in Cricău and Ighiu, by a document (February the 12th 1238), signed at, Archita (village belonging now to Vânători, Mureş County), established by the Saxons around the year 1200. The Saxons of Cricău and Ighiu received the right to exploit gold at Zlatna and Vulcoi.

Carol Robert de Anjou was the one that ensured a real development of mining in Transylvania, by developing mining regulations, by a *Letter of Privileges*, signed at Visegrád, in the year 1327. Unlike the privileges given by Bela the Fourth, referring only to certain mining communities, those granted by Carol Robert de Anjou covered all the miners in the Kingdom of Hungary. Moreover, the terrains where precious metals gad been discovered did no longer become the king's property, they stayed in the property of the owners, with the condition that 2/3 of the „urbura" to be deposited in the regal Treasury, the other one third remaining to the owner. The gold produced in the four mining centres of the Apuseni Mountains, Abrud, Baia de Arieş, Zlatna and Baia de Criş, had to be exchanged in the regal

office in Sibiu, at the official price fixed by the king. As a result of making miners co-interested, auriferous mining in the Apuseni Mountains raised up, many landowners starting a campaign of exploitation of the resources of the underground. The privileges granted to the miners by Carol Robert de Anjou were reconfirmed by Ludovic I de Anjou (1351), Sigismund de Luxemburg (1405), Matei Corvin (1486), Vladislav II Iagello (1492) and Ludovic II Iagello (1523).

Based on these regulations, functioning of mines was submitted to a process of organization. A document of September the 20th 1347, of the Hungarian Chancellor, for the miners of Baia Mare, shows very interesting data in this sense. Thus, the community of workers, together with the judge and the jury of the locality to which the mine belonged, (today's mayor and councillors), chose for a year a *knowledgeable magister of the mountain* (chief of mine), responsible for the good functioning of the mine. In his turn, he appointed the *„steigers"* (today's foremen) and *gold examiners* (today's probers), with the approval of the judge and the jury. The „steigers" supervised work in the mine and collected the „urbura" for the king, and the gold researchers took samples from the faces, establishing the metal content of the ore. The regulation stipulated severe punishments for those who avoided acquitting the „urbura", but also defended the miners in case of abuses.

A report on organizing auriferous mining and production of noble metals in Transylvania in the 15th and 16th centuries of Paul Bornemisza (Catholic Bishop of Veszprém) and Georg Werner, commissioners to King Ferdinand the First of Bohemia and Hungary (1526–1564), sent to analyze the worrying situation of the mines of that area, shows interesting data. In the first half of the 16th century, gold production in Transylvania decreased significantly. Mines became deeper and the methods of that time could hardly handle the evacuation of underground water, and in many cases the producers were too poor to cope with the situation. The report shows that Transylvania was richer in gold than in silver, but from the four important mines only the one of Baia de Arieş belonged to the king, and this preponderantly extracted silver. The mines in Zlatna and Abrud, very rich in gold, belonged to Alba Iulia, Baia de Criş mine was submitted to Şiria, belonging to Matei Corvin. The decline in gold production was caused by the lack of cash of the Exchange office in Sibiu. The clandestine money changers, called *gozars* by the population of the Apuseni Mountains, dealt heavily with gold, eluding the regal monopoly and making huge profit. This is the reason why the small gold producers are scarce on the exchange tables, although they made the greatest gold production. Usually the gold exchange was leased to the Saxons in Sibiu, for 4000–5000 Florin per year, sometimes even more. By the end of the 15th century, the gold exchange was approximately 350 kg/year, decreasing to 300 kg in 1552. In order to correct the situation, the Transylvanian princes took a series of measures, among which reconfirmation of the old privileges of the mining towns, secularization of bishop domains of Alba Iulia and introduction of foreign capital in auriferous mining.

3. MINING TECHNOLOGY [1,3]

Interesting data referring to the technology applied in the gold mines near Zlatna and Abrud in the second half of the 16th century are presented by foreign travellers visiting these places. Thus, the French Pierre Lescalopier, who visited Transylvania in 1574 visited the Zlatna gold mine, where *„one goes far in under a high mountain,* [Vulcoi – n.o.]. *The miners take out some rock from here, then they burn it first as it were gypsum, then they grind it in a water mill that throws the burnt stones in a one foot wide, two pole long wooden gutter, where the*

rock is crushed with thick pestles that rise and fall one at a time, one after the other. The gravel is slowly carried by water, and spread over thick burlaps, laid out on a slightly inclined floor over a large trough. The gold is caught up on these burlaps, and what is not caught up falls in the trough, and the water flows down. Two times a day the miners raise these burlaps, wash them in other troughs, then take out the bits and pieces on large wooden cymbals, which then are shaken little by little, until the gold gathers in one side, and the useless sand on the other side of the cymbal. When a certain amount of gravel with gold is amassed, they put a little quicksilver in an alembic where, by the action of fire, the quicksilver reduces the gold to bars, then the quicksilver evaporates" [6, pp. 140–143].

The Austrian nobleman Felician de Herberstein, who took on a lease the goldmines in Cavnic next to Baia Mare, visited the mining region of the Apuseni Mountains by the end of the year 1585, at the request of the king of Poland Ştefan Báthory (1575–1586), drawing up a thorough report regarding the mines of Bucium, Almaş, Ruda and Băiţa. The report gives interesting details referring to the technology of extraction and processing of gold ores there, completing the information in the report of the imperial commissioners sent to Transylvania in the year 1552. Herberstein's report might have been a little biased, presenting the situation more worthless than it was in reality, considering his intention to take these mines on a lease, which happened indeed. After his death, the contract of lease was prolonged by Prince Sigismund Báthory by his diploma of July the 1st 1591, given to Heberstein's sons, Sigismund and Friedrich, where the villages affected to the mines were also mentioned. The contract was dissolved in the summer of 1597.

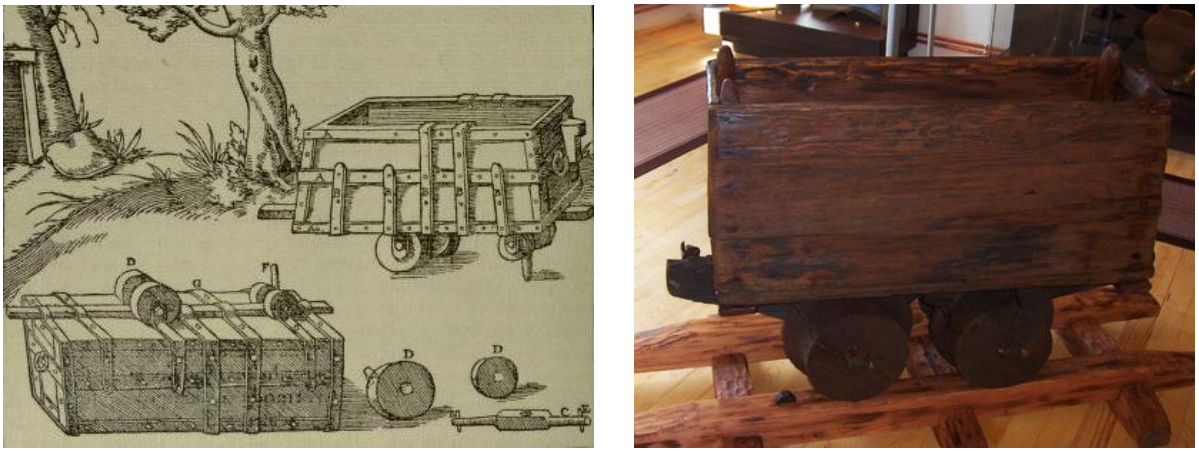
The most complete information related to ore extraction and processing in the middle of the 16th century were shown in detail by the German scientist Georg Agricola, Georg Bauer his real name(1494–1555), in his famous book *De re metallica libri XII* (*On Mining and Metallurgy*), published in 1556, shortly after the author's death. Written in Latin, the book was translated to German, with many errors in its content and language. In the year 1912 an excellent translation was made by the American mining engineer Herbert-Clark Hoover and his wife Lou Henry-Hoover; the mining engineer became later the president of the USA (1929–1933). A physician by profession, Agricola approached minerals little by little, treating his patients with mineralogical remedies, known in ancient literature; therefore he settled in Chemnitz, in the Metaliferous Mountains in Eastern Germany, very rich in nonferrous metals. There was a saying in those parts, *„If a German throws a stone or dross after a cow, it might be more precious than the cow".* It was there that Agricola deepened his knowledge on mining and metallurgy, being considered the father of mineralogy, and his book, *De re metallica*, with 273 illustrations, is a quintessence of all that had been known at that time about mining and metallurgy.

Clearly Agricola visited the gold mines in the Apuseni Mountains, he mentioned more than once the gold extraction procedures from the ore applied in the Carpathians. At that time, gold was extracted and processed only in the Western Carpathians. Besides, Agricola mentioned Zlatna, Abrud and Baia de Criş in other two works related to mountains and mineralogy (*De natura fossilium, libri III* and *De veteribus et novis metallis, libri II*). He also had good knowledge of mining technology applied in the ancient world from the writings of Plinius the Old, whom he frequently mentioned. After 1300 years, gold extraction methods remained the same, only that mines had become deeper. In hard rock the *method of fire and water* was used, progress was only made in ore transportation, water evacuation and mine ventilation. Certainly, to grind auriferous ore stamps were used.

Wooden wagons had rectangular shape, the upper part being slightly narrower, fastened by and iron strips. At the lower part there were two metal shafts, and their extremities wooden wheels were mounted. The wagons ran on wooden paths, and to avoid derailing, in the middle of one of the shafts, there was a guiding pivot that entered in the canal of the thick plank between the rails, which was also used by people to walk. The wagons were pushed from the back, full as well as empty. Since during the travel the wagons made a squeaking sound, like a puppy, the German miners called them *hunt* (*Hund* = dog), becoming *hont* in the language of the people living in the area.

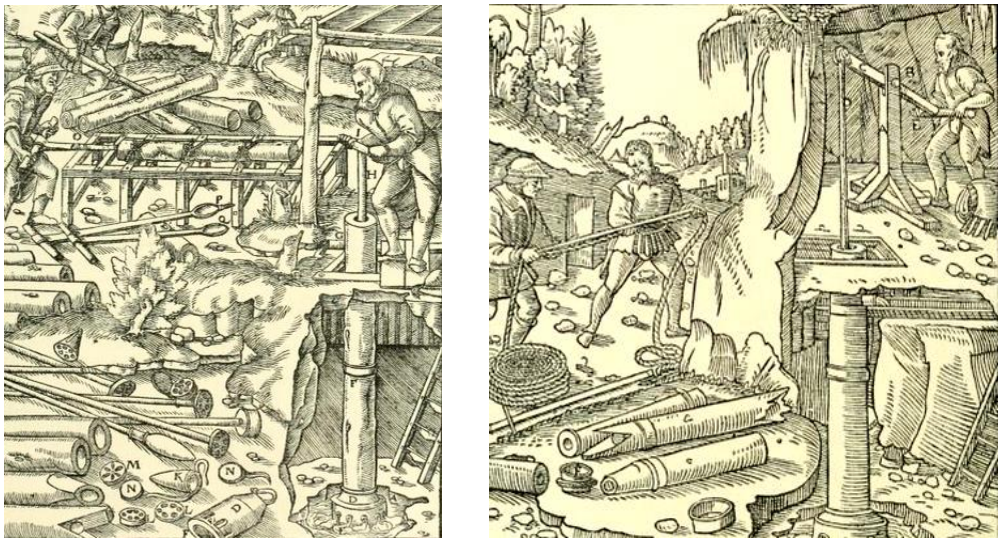
The oldest wooden wagon in the world goes back to the 14th century, and had been discovered in the old mining workings in Ruda–Brad, together with the transport line and a switch, also made of wood. The original wagon is found at the Museum of Mining in Bochum (Deutches Bergbau-Museum), and the switch at the Museum of Transport in Berlin (Museum fur Verkehr und Technik). The line switch was invented around the year 1600 by the miners in Brad. Similar wagons were used in the mines in England, but only at the beginning of the 17th century, that is almost 300 years later.

Water evacuation was made by pumps with piston, manually driven by a handle or a balancing mechanism. In principle, such a pump was made up of a wooden cylinder, wherein a piston with several holes in it moved, and on its upper part there was a leather choke, attached to the piston in only one point. The lower part of the wooden cylinder had several radial holes, by which water collected in a sump entered. At the descending stroke of the piston, manually driven by a rod with a handle or a balancer, water entered above the piston, through its holes. At the upstroke, the water weight pressed on the leather choke, which sealed the piston, so that water was lifted up to a flow gutter, being then discharged in a canal or in another intermediary sump. This principle is used nowadays to take out water from deep fountains in flat areas; instead of the piston, a long, cylindrical vessel is used, with diameter slightly smaller than that of the concrete fountain. At the descending movement, water enters by its perforated bottom, and when the vessel moves up, a rubber or leather choke seals the vessel, keeping the water inside.



a ***b***

Figure. 1. Wooden wagon: ***a*** – in Agricola's work; ***b*** – reconstitution



a ***b***

Figure. 2. Water pumps with piston (Agricola): ***a*** –with handle; ***b*** – with balancer

The length of the wooden cylinders was limited to 3–3,5 m, so that special drills could be used to make the holes. Usually, for higher water evacuation, two such cylinders were joined, one lengthening the other, and for a good sealing, a funnel shaped leather gasket was used, tied with the narrower part to the piston, the larger part sliding along the cylinder. In the medieval workings at Roşia Montană a 7 m long pump was found, with 22 cm outer diameter. It is kept at the Museum of Auriferous Mining at Roşia Montană.

Mine ventilation was made by rather ingenious methods, combining forcing ventilation with induced ventilation. In the first case, an iron circle reinforced barrel was used, mounted on a wooden duct that entered the ventilation shaft. The link between the barrel and the duct was made by a wooden disk, fixed to the duct by a square hole, allowing the free rotation of the barrel in both senses. The barrel rotated around a longitudinal axis, at the slightest wind, due to certain wings, always bringing the air entering window in the direction of the wind. This construction is known today as *ventilation hat*. The clean air forced itself in the ventilation shaft, flushing the underground mine workings, the vitiated air being evacuated afterwards to the surface through a secondary shaft.

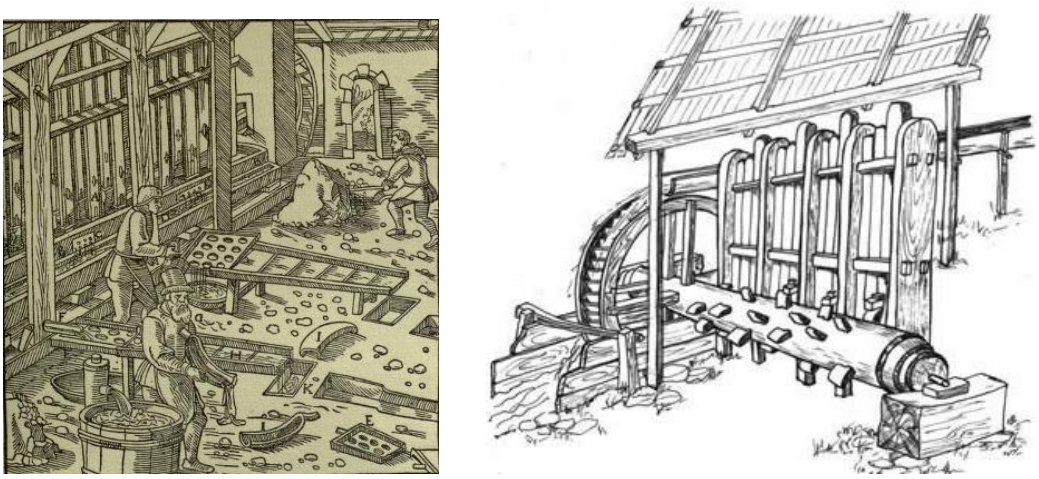


a ***b***

Figure. 3. Mine ventilation methods (Agricola): ***a*** – forcing; ***b*** – induced

Induced procedure ensured the evacuation of the vitiated air from the mine workings with only one access way, simultaneously with the unrestricted entering of the clean air, by compensating the created depression, but it involved the use of a man to manually drive the installation. It was mainly made up of a chamber with leather windbag with creased walls (bellows), driven by a system of levers. The ventilation shaft had two compartments separated by tightly joined planks, one with a smaller section, to aspire vitiated air, and the other with a larger section, to let clean air come in. The windbag was mounted on a wooden rectangular funnel, and the worker actuated the leather windbag by a system of levers. Thus, the „stinking air" (Agricola dixit) was aspired and evacuated into the atmosphere by the mouthpiece of the windbag, which folded due to the weight of a boulder mounted above. The windbag had two vents that closed and opened alternatively: the one of the mouthpiece closed when the vitiated air was aspired, and the one of the wooden funnel closed when the windbag folded, and vice versa. The induced ventilation system was applied with all types of mine workings that had no second outlet, today it is called *dead-end ventilation.*

Grinding auriferous mineral wad made in Agricola's time with stamps driven by the force of water. These installations were invented by the German miners in the 12th century, being taken over by the French miners(13th century) and the Polish (14th century). The miners in Transylvania also began using the stamps in the 16th century, first in the Apuseni Mountains, then in Baia Mare, taking over the craftsmanship from the German miners colonized there, especially after the privileges granted by Carol Robert de Anjou and reconfirmed by his heirs. The stamps revolutionized the gold extracting technology, replacing the pestle and grinders, remaining the only grinding tool until the coming about of the mills.



a ***b***

Figure. 4**.** Auriferous stamps: ***a*** –in Agricola's work; ***b*** – scheme of principle

4. CONCLUSIONS

After the Romans withdrew from Dacia, auriferous mining declined, due to the invasion of the migrant populations. Exploitation of precious metals restarted in the 8th century, after the assimilation of the Slavs and intensified in the 10th century, after the establishment of princedoms and voievode's domains. Later, these pre-state formations were enclosed in the medieval Kingdom of Hungary. The Hungarian kings colonized the conquered territories, to protect the borders, and to put to good use the underground resources.

Miners were granted economical and legal privileges for the exploitation of precious metals from underground or alluvium, on the condition that they would pay part of the winnings as taxes. The most important privileges were give in 1327 by Carol Robert de Anjou, by a mining regulation, which was reconfirmed by his heirs until the end of the Middle Ages.

As far as precious metal extraction technology from underground is concerned, it was very much like during the Roman period. Auxiliary mining activities experienced however innovations. Thus in the 16th century at Brad wooden wagons and switches were used for the first time in history, and manually driven piston pumps were used for water evacuation. Mines were ingeniously ventilated, using wind or manual actuation of a chamber with windbag. . During this period auriferous stamps were used to grind ore. All these were described in detail by the German scientist Georg Agricola in his famous book *On Mining and Metallurgy*, but tools had also been discovered in the old mines in the Apuseni Mountains, being now exposed in museums.

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DOI: 10.7251/BMC170701059M

EVALUATION AND PERSPECTIVES OF UNDERGROUND COAL MINES IN SERBIA

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ABSTRACT

This paper provides analysis of mines operating in Public Company for Underground Coal Exploitation (PCUCE), which is in restructuring process in excess of 10 years. Evaluation of these mines included analysis of performance and development potential of eight mines. Evaluation took into account a wide range of factors, considered to be most appropriate in relation to whether output and productivity could be increased and whether there are enough resources to justify the necessary investment. Based on AHP method and scores allocated for each factor, the mines were ranked according to technical potential.

Key words: Underground mining, Potential, Restructuring, Evaluation

1. INTRODUCTION

Serbia is one of the major coal producers in Europe, with over 38 Mt of coal per year. But most of this production is achieved in lignite open cast mines, which in 2016 produced 37.6 million tonnes [1]. On the other hand, combined production of all underground coal mines in Public Company for Underground Coal Exploitation (PCUCE) is just around 0.6 Mtpa, making their contribution to the energy sector quite insignificant. It should be noted that PCUCE operates 8 mines (with 11 pits) and one non-producing mine whose workers are hired as contractors in other mines. These mines are dispersed throughout Serbia, and they are producing various types of coal, as given in Table 1.

Table 1. Mines, location, type of coal produced and number of workers (as of March, 2017)

Mine	Town / Municipality	Type of coal	Number of workers
Vrška Čuka	Vrška Čuka, Zaječar	Anthracite	145
Ibarski rudnici	Baljevac, Raška	Hard coal	479
REMBAS	Resavica, Despotovac	Brown coal	1161
Soko	Soko Banja, Soko Banja	Brown coal	535
Bogovina	Bogovina, Boljevac	Brown coal	247
Jasenovac	Krepoljin, Žagubica	Brown coal	254
Štavalj	Štavalj, Sjenica	Brown coal	449
Lubnica	Lubnica, Zaječar	Lignite	329
Aleksinački rudnici	Aleksinac, Aleksinac	Contractor	4050 [1)]

[1)] Including 122 employees in management (Headquarters in Resavica).

Operation and performances of PCUCE were evaluated and analyzed on several occasions in previous years. Most notable analysis were [2], during which PCUCE was a part of JP EPS, and [3], study financed by European Agency for Reconstruction whose beneficiary was Ministry of Mining and Energy, RoS. These studies identified the need for investment in some of the mines to improve their technical performance and also limiting potential of the other mines, mainly due to lack of coal reserves, market and difficult conditions. However, restructuring process of PCUCE is in standstill, meaning that little changes have been made in previous years. Finally, it should be noted that PCUCE is has direct annual subsidies from the Budget of RoS.

The purpose of this paper is to apply more recent decision-making tools, such as Analytical Hierarchy Process (AHP), for the purpose of evaluating and ranking of the mines operated by PCUCE.

2. CRITERIA FOR EVALUATION

Initial step for evaluation of mines operated by PCUCE is to select criteria which will be used in AHP. For this purpose following criteria are selected:

- Reserves, score is given in relation to existing A and B amounts of reserves (criteria C1);
- Geological conditions, such as faulting, seam inclination, block size, seam variation, roof and floor conditions, rock pressure and geo-mechanic conditions (criteria C2);
- Mining hazards, such as gas, water, dust and potential for gas outbursts and spontaneous combustion (criteria C3);
- Potential for mechanization, such as seam inclination, seam thickness, floor and roof conditions, block size, coal clearance facilities and supply facilities (criteria C4);
- Quality of coal and products, such as coal type, sulphur content, ash content, moisture content, calorific value and amount of fines (criteria C5);
- Market and delivery possibilities, such as current markets served, the distance to the market and transport options available (criteria C6);
- Investment potential, possibility to attract an investor (criteria C7).

Score for each criterion ranged from 1 to 5 in relation to actual value of parameter or parameters. Due to restricted number of pages, scores of selected criteria required for AHP are given in Table 2. To provide more complete picture, coal productions achieved in period 2010-2015 in these mines are given in Table 3.

Table 2. Scores of criteria for AHP

	C1	C2	C3	C4	C5	C6	C7
Vrška Čuka	1	2	3	1	4	2	2
Ibarski mines							
- Jarando	1	2	2	1	4	2	1
- Tadenje	1	1	4	2	3	2	1
REMBAS							
- Strmosten	3	3	2	3	3	4	4
- Block IV	2	2	2	3	4	4	2
- Senjski rudnik	1	2	2	2	3	4	1
Soko	5	2	1	3	3	4	4
Bogovina	2	3	4	1	2	2	1
Jasenovac	2	2	3	2	3	2	2
Lubnica	4	4	2	3	2	2	2
Štavalj	5	3	3	3	3	1	1

Table 3. Production from PCUCE mines, in period 2010-2015

Year	Coal production (t)								
	Vrška Čuka	Ibarski rudnici	REMBAS	Soko	Bogovina	Jasenovac	Lubnica	Štavalj	JP PEU
2010	7,061	101,200	113,000	121,317	4,160	62,000	67,330	76,118	552,186
2011	7,034	134,474	133,001	121,317	9,684	65,230	78,150	82,665	631,555
2012	6,807	140,993	126,751	122,330	18,758	62,240	58,770	46,960	583,609
2013	5,121	155,299	138,092	119,006	1,441	40,390	60,015	82,075	601,439
2014	5,115	120,939	147,902	117,028	5,270	36,530	48,140	84,305	565,229
2015	6,016	123,318	150,521	101,200	13,490	38,960	42,966	84,180	560,651

These scores are used for evaluation of PCUCE coal mines by using the Analytic Hierarchy Process method, as described in following chapter.

3. AHP EVALUATION OF UNDERGROUND COAL MINES IN SERBIA

The Analytic Hierarchy Process (AHP) is a multi-criteria decision-making approach introduced by Saaty [4]. The AHP method enables users to determine the weights of the parameters in the solution of a multi-criteria problem. In the AHP method, a hierarchical model consisting of objectives, criteria, sub-criteria and alternatives is used for every problem [5].

Solving a problem using AHP is carried out using the weights or priorities of the criteria subjected to pairwise comparison (Table 4). Weights or priorities are determined by normalizing the pairwise comparison matrix (Table 5). While performing pairwise comparisons of criteria in the AHP method, a certain level of inconsistency may occur. Therefore, the logical consistency of pairwise comparisons must be checked [6]. To measure the consistency of pairwise comparison judgments, the consistency ratio proposed by Saaty [7] is used. A consistency ratio is calculated for the pairwise comparison matrix. The upper limit proposed by Saaty for this ratio is 0.10. In a case where the consistency ratio calculated for the judgments is below 0.10, it is considered that the judgments exhibit a sufficient degree of consistency and that the assessment can be continued. If the consistency ratio is above 0.10, then the judgments are considered inconsistent. For each level in the hierarchy it is necessary to know whether the pair-wise comparison has been consistent in order to accept the results of the weighting. The parameter that is used to check this is called the Consistency Ratio. The consistency ratio is a measure of how much variation is allowed and must be less than 10%. Finally, random index (RI) is given in Table 6.

Table 4. Pairwise comparison matrix of criteria

	C1	C2	C3	C4	C5	C6	C7
C1	1	2	3	1	1	2	4
C2	0.5	1	0.5	0.5	0.5	0.5	0.5
C3	0.33	2	1	0.33	0.33	0.5	2
C4	1	2	3	1	1	3	2
C5	1	2	3	1	1	2	2
C6	0.5	2	2	0.33	0.5	1	1
C7	0.25	2	0.5	0.5	0.5	1	1

Table 5. Normalized pairwise comparison matrix

	C1	C2	C3	C4	C5	C6	C7	
C1	0.2182	0.1538	0.2308	0.2143	0.2069	0.2000	0.3200	0.2206
C2	0.1091	0.0769	0.0385	0.1071	0.1034	0.0500	0.0400	0.0750
C3	0.0727	0.1538	0.0769	0.0714	0.0690	0.0500	0.1600	0.0934
C4	0.2182	0.1538	0.2308	0.2143	0.2069	0.3000	0.1600	0.2120
C5	0.2182	0.1538	0.2308	0.2143	0.2069	0.2000	0.1600	0.1977
C6	0.1091	0.1538	0.1538	0.0714	0.1034	0.1000	0.0800	0.1102
C7	0.0545	0.1538	0.0385	0.1071	0.1034	0.1000	0.0800	0.0911
Sum	1	1	1	1	1	1	1	1

Lambda (λ) = 7.355393; CI = 0.059232; RI = 1.32; CR = 0.044873;

Table 6. Random index

n	3	4	5	6	7	8	9	10
RI	0.58	0.9	1.12	1.24	1.32	1.41	1.49	1.51

Rank of the underground coal mines operated by PCUCE obtained after completion of the calculation according to the AHP method is given in the Table 7. It can be seen that Soko has the best ranking, while the Jarando pit of Ibarski mines has the worst ranking. This result is mainly because Soko has significant coal reserves, stable market and potential for improvement (mechanization and investment), while these criteria are completely opposite in case of Jarando pit in Ibarski mines.

However, it is more important to highlight perspective or potential of all the mines. In this context, ranking of the mines provides reasoning for supporting some of the mines which are having potential for improving their operation (Soko, Strmosten pit in REMBAS mine,

Lubnica and Štavalj) and for closure of those mines without potential (Tadenje and Jarando pits of Ibarski mines, Senjski rudnik pit of REMBAS, Bogovina and Vrška Čuka).

This approach leaves Jasenovac mine and Block IV pit of REMBAS mine in between these two groups, meaning that these two mines should probably continue operation until depletion of reserves or in relation to other non-technical criteria (such as social importance).

Table 7. Rank of the PCUCE mines

Rank	Mine	Value
1	Soko	0.1296
2	Strmosten	0.1203
3	Lubnica	0.1191
4	Štavalj	0.1179
5	Block IV	0.0987
6	Jasenovac	0.0738
7	Tadenje	0.0715
8	Senjski rudnik	0.0699
9	Bogovina	0.0687
10	Vrška Čuka	0.0682
11	Jarando	0.0624

4. CONCLUSION

Public Company for Underground Coal Exploitation operates 8 mines and one company working as a contractor at other mines. AHP analysis, based on technical criteria, identified two groups of mines, one with potential for further development (Soko, Strmosten pit-REMBAS, Lubnica and Štavalj) and the other group of mines which should be closed in near future (Tadenje and Jarando pits-Ibarski mines, Senjski rudnik-REMBAS, Bogovina and Vrška Čuka).

Results of this analysis are in accordance to findings of study [3], completed some 10 ago, meaning that any changes are not likely and decision on closure of above mentioned mines should be made soon. This approach would enable proper support to mines with potential for development. Finally, two mines (Jasenovac and Block IV-REMBAS) are in between these two groups, and decision on their future should include other aspects, such as social management.

Acknowledgment

Research described in this paper was performed during development of the project "Research on possibility for AT (Advanced Technology) rockbolting application in mines for the purpose of increasing work safety and production efficiency" (TR33025). Development of this project is financed by Ministry of Science and Technological Development, Republic of Serbia.

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DOI: 10.7251/BMC170701065M

OPENING AND DEVELOPMENT OF THE "ĆELIJE" DEPOSIT NEAR GADŽIN HAN, REPUBLIC OF SERBIA

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ABSTRACT

This paper presents underground opening and development works in the "Ćelije" deposit, which will, in the shortest time possible, enable starting excavation works in the "Ćelije" mine.

Opening and development, according to the mining and geological conditions in the "Ćelije" deposit, strictly respecting the safety and enviromental protection measurements, are the most important for starting and, therefore, sustainability of the planned calcite production.

In order to achieve planned underground rooms production, diesel-powered LHD machines will be engaged, and the whole production process will be completely compliant with modern trends in mining industry .

Key words: opening and development, calcite, Ćelije

1. INTRODUCTION

Head office of the Belkal Han Ltd. company is situated in Gadžin Han, near the town Niš, Serbia. Based on the corresponding technical documentation (*Calcite Mineral Reserves Study of the "Ćelije" Deposit Near Gadžin Han as Carbonate Raw Material* [1]) and *Major Mining Project for Opening and Exploitation of the "Ćelije" Mine* (GRP), [2]) According to Study [1], balanced calcite ore reserves are 1.238.620t.

Quality of calcite from the "Ćelije" deposit is certified by several relevant institutions (Institute for cellulose in Ljubljana, Slovenia, Institute for technology of nuclear and other mineral raw materials (ITNMS) Belgrade, Serbia, etc.) It is suitable for industrial applications due to high whiteness, excellent chemical composition and purity. In the previously mentioned GRP, it is calculated that the annual ore production is 25,000 t.

Within this GRP, a chamber-pillar method for excavation with open stopes was selected and defined, with all the appropriate calculations and defined parameters.

On the request of the Investor, and according to the needs stated, the decision for realizing the *Technical Mining Project for the excavation of the "Ćelije" deposit - Gadžin Han from level k+865 m to level k+924 m* (TRP) [3], in which development and the method of excavation is processed in order to modify the aforementioned GRP, was made, with one essential change - the application of diesel-powered equipment for loading and transport, instead of compressed air-powered equipment.

The purpose of this project is to start the excavation phase in the shortest possible time, starting with the exploitation of calcite, in order to process it further for the needs of the Investor, whose industrial plant should start operating in 2017. It is expected that the annual capacity is about 30,000 t of ore, after opening and excavation of the part of the deposit from level k+865 m to k+924 m; more accurately - 303,246 t of pure ore, after the elimination of losses by leaving the safety pillars and boards, which ensures the primary life of exploitation of 10 years, while the part of the deposit below the level k+865 to the level k+835 m will be excavated later. The project-designing of excavation of this part of the deposit is progress. This paper presents the opening and development the part of deposit between levels k+865 m and k+924 m, which is shown in the TRP.

Mode of opening and development of the "Ćelije" deposit is shown in the plan in Figure 1.

Technical description of the opening and development of underground rooms

The opening and development rooms that will be covered in this paper include 5 types of underground rooms:

Type 1: Export adit -1 (IP-1), Service-transport ramp (STR) and Access corridors (PrH), dimensions 3.3x3.0 m, without support, light and excavation cross-sections F_s= 8,51 m^2, F_{isk}= 8,93 m^2,

Type 2: IP-1, STR and PrH, dimensions 3,3x3,0m, with steel arc support, bright surface of cross-section F_s= 8,69 m^2, F_{isk}= 9,6 m^2,

Type 3: Ventilation drift - 1 (VP-1), dimensions 2,8x2,9 m, without support, F_s = 7,28 m^2, Fi_{sk}= 8,12 m^2,

Type 4: VP-1, dimensions 2,8x2,9m, with anchoring support, cross section F_s = 7,28 m^2, F_{isk} = 8,12 m^2,

Type 5: Passing and ventilation raise - 1 (PVU-1), bright dimensions 2.0x2.0 m, with supporting, if necessary, with woodden frames one on the other ("cribbing" method), if necessary, in the very top of the passage-ventilation raise. Cross section is to be F_s= 4,0 m^2, F_{isk}=4,2 m^2.

Cross-sections of the underground rooms that are intended for the opening and development of the "Ćelije" deposit according to [3], (types 1 and 2) are shown in Fig. 2. The cross-sections of rooms of type 3, 4 and 5 are shown in [2 and 3] and will not be displayed here.

In the following text, due to the large number of rooms, only the most important opening rooms will be described - IP-1, VP and STR .

1.1 Haulage adit-1

On the request of the Investor, the opening of the mine is carried out in the premises that are located on the parcels that are its property and the location and azimuths of the main opening rooms (IP-1 and VP-1) are defined and determined as shown in the GRP [2].

An Haulage adit-1 (IP-1), which starts with k+862 m, is projected as the main room for the opening of the pit, under an azimuth of ν=45°. The Haulage adit IP-1 is a type 1 room, or, if necessary, type 2.. It is made through limestone, with a rise of about 4%, and saves the altitude difference of 3.0 m.. Endpoint of the IP-1 is at level k+865 m. The length of the IP-1 is L=74.49 m, of which the last 14m passes through the ore - calcite. The IP-1 position is shown in Figure 1, and the cross-section with the supporting method in Figure 2.

1.2 Service-transport ramp - 865-921 (STR-865-921)

Service and transport ramp STR-865-921 (STR) is a low-gradient slope underground room whose construction starts from the end of the Haulage adit-1 (IP-1) at level k+865m and is made up to the highest point of the part of the deposit intended for exploitation, or up to level k+921 m. The main purpose of this room is to transport the excavated calcite and other material (limestone) obtained by the construction of underground rooms. STR is also used as the passage of employees, as well as for the delivery of necessary working materials and energy. At the same time, through this underground room the work sites and this part of the pit will be ventilated.

STR is to be constructed through limestone in full length, in the floor, along the calcite ore vein. Overall length of the STR is 384.36 m. Generally, it is designed with a slope of 18% and saves the altitude difference of Δh=56.0m. The beginning of the STR is made in azimuth of ν=46°, and after the curve the general azimuth is ν=91°. STR is type 1 room, or, if it is necessary, type 2. This cross-section size is dimensioned for the purpose of using an underground truck. The position STR-865-921 is shown in Fig. 1, and the cross-section with the supporting method is shown in Fig. 2.

1.3 Ventilation adit -1 (VP-1)

Ventilation adit-1 (VP-1) is defined by GRP as the main room for ventilation of the pit. Construction of the ventilation adit-1 VP-1 begins at level k+879,0 m, at the azimuth of ν=68°. VP-1 is constructed at the length of 76.27m with a slope of 9.50% and saves the altitude difference from Δh=7.20 m. Ventilation adit-1 is made up to level k+71,80 m. Dimensions of the VP-1 are defined by GRP, type 3. The supporting of the ventilation adit-1 is not planned, and if necessary, as defined in the GRP, the support will be carried out with anchors and, exceptionally, anchors and a steel wire. The position of VP-1 is shown in Fig. 1.

1.4 Ventilation drift-1 (VH-1)

The ventilation drift-1 (VH-1) is also defined by GRP as one of the main pit ventilation room. It connects the end of the VP-1 and the beginning of the STR-865-921. Construction of the VH-1 ventilation drift begins at level k+871.80 m and is made at the azimuth of ν=113°. VH-1 is made in the length of 63.87m with slope of 9.35% and it saves the altitude difference from Δh=5.90 m. The VH-1 is made up to level k+865,90m. VH-1 dimensions are defined by GRP (type 3 or 4). The position of VH-1 is shown in Fig. 1.

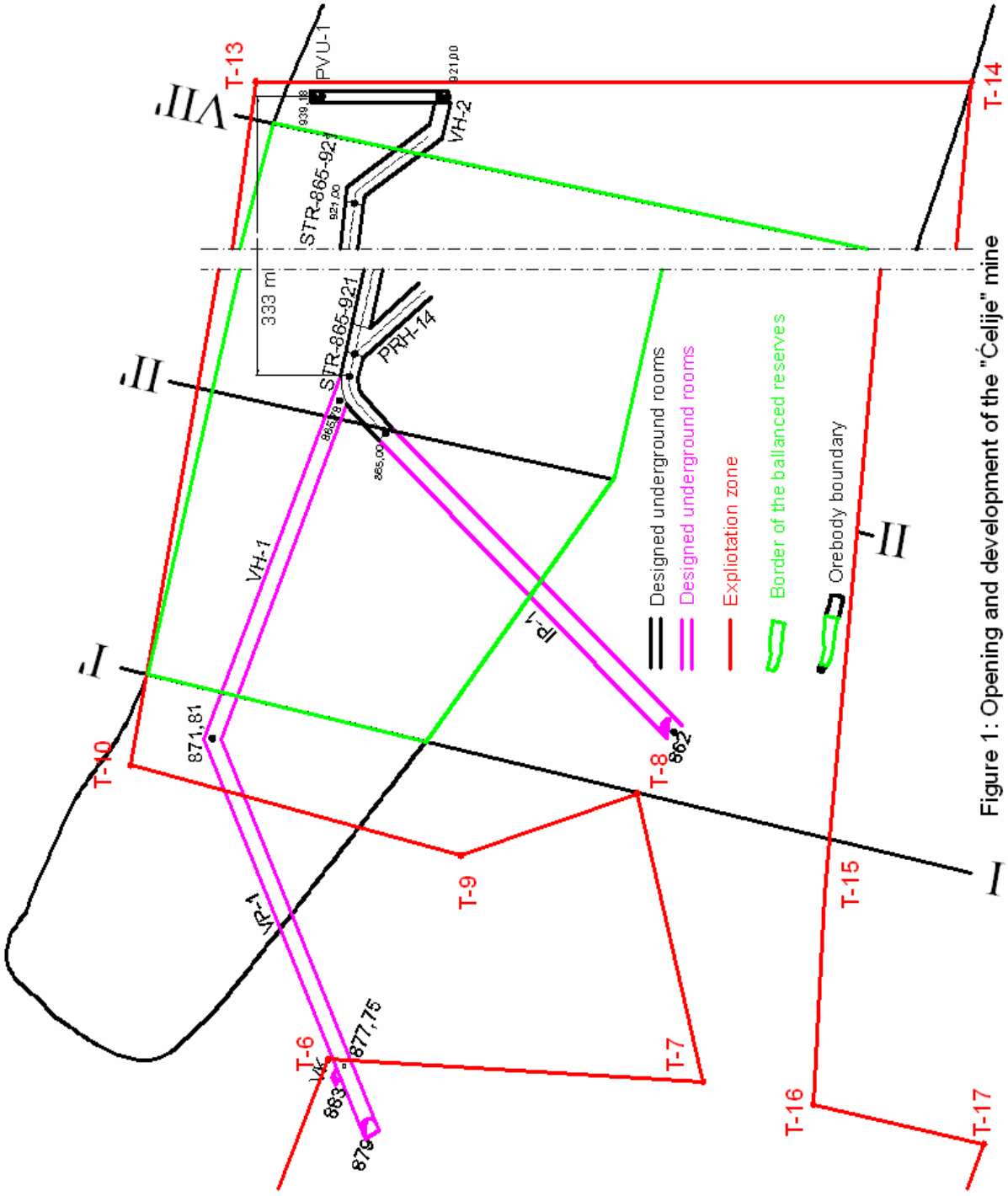


Figure 1: Opening and development of the "Ćelije" mine

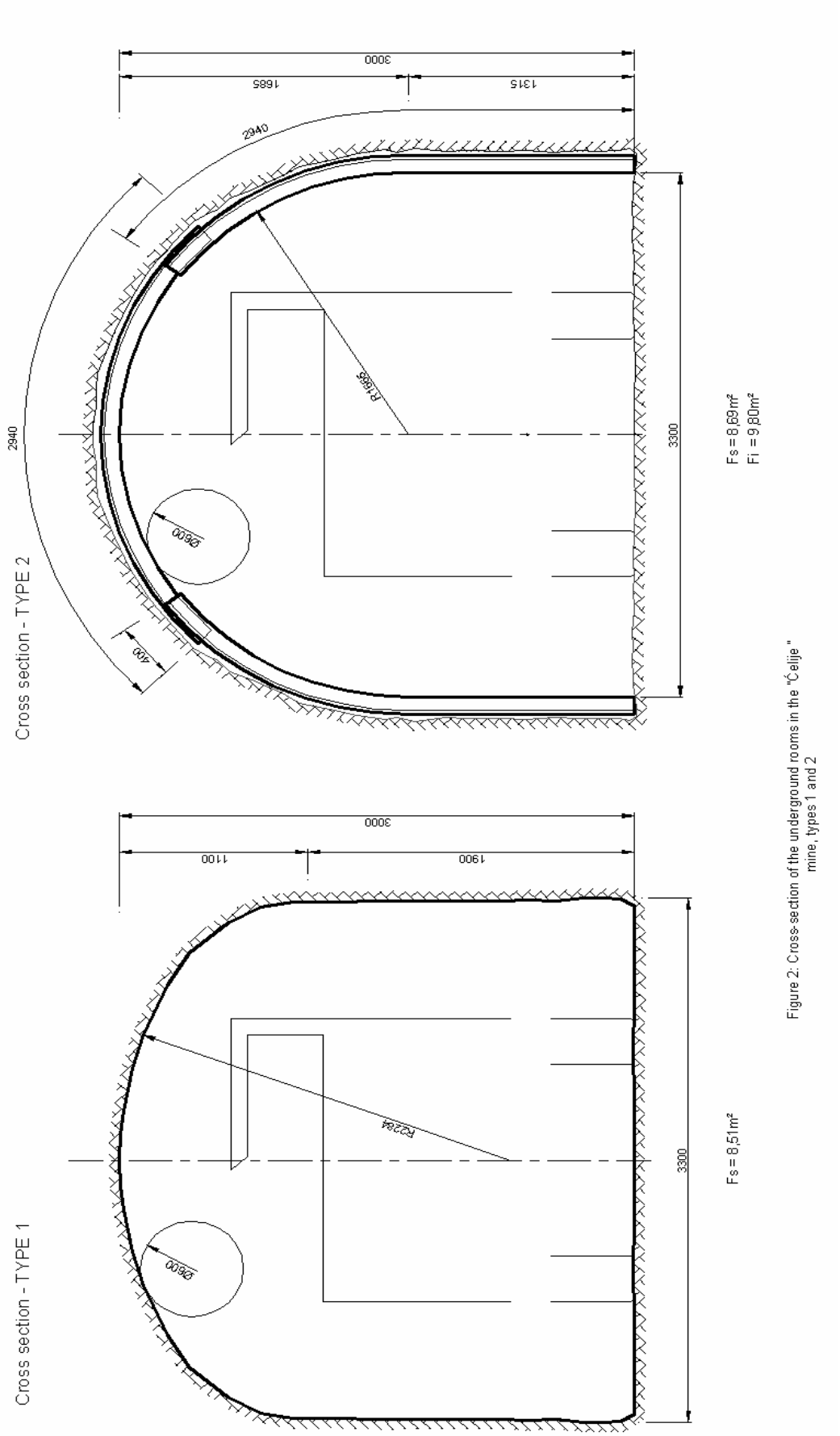


Figure 2: Cross-section of the underground rooms in the "Čelije" mine, types 1 and 2

Table 1: Suimmary - opening and development undergroundrooms

Underground room	Designed length	Δh	Real length
	[m]	[m]	[m]
		(a-b)	
STR 865-921	379.35	56.00	384.36
PrH	163.23	0,00	163.23
Other	242.07	-	249.29
Overall	**784.65**	**-**	**796.88**

As shown in Table no. 1, for the opening and development of the "Ćelije" deposit from level k+865 to level k+924 m it is necessary to construct the total length of L=796.88 m, or about total length of L=800 m of underground rooms.

2. TECHNOLOGY OF CONSTRUCTION OF THE UNDERGROUND ROOMS

Classical technology will be used to construct undreground rooms for the opening and development of the "Ćelije" mine. Hole drilling will be carried out by drilling hammers with support leg and monoblock chisels, length 1,6 m. For blasting, patronized explosive type "Amonex", which will be activated by electric detonators, will be used.

Ventilation of underground rooms during construction will be performed separately, with appropriate compression fans with diemeter of Ø=600 mm. After constructing PVU-1 all throught to the surface, the ventilation of the rooms and the entire pit will be done in combination, both flow and separation way.

The loading and transport of the blasted material is carried out by a diesel-powered loader (LHD), with a bucket of about 2 m^3, and later, during the excavation, the trucks for underground mining will be used for transportation of ore from stopes to the surface.

The supporting of underground rooms is not planned, and, if needed, the anchors with steel wire or the steel arc support, depending on the condition of the working environment.

2.1 Manpower, dynamics and cost of production

For the realization of the projected technology for the construction of underground rooms for opening and development in the "Ćelije" mine, it is necessary to hire 6 workers by shift: 1VKV (highly qualified worker - shift controller), drilling and blasting 1KV (qualified worker) and 1PK (half-qualified worker), for additional operations (supporting, etc.) - 1KV +1PK, and for loading and transport 1KV (operator of LHD machine). The summary of working effects on the construction of horizontal and low-gradient slope rooms is U=5.88t/per single shift. According to the dynamic construction plan, in case the rooms are made with one

attack point, it takes about 13 months for the realization of the mentioned works. If the Investor decides otherwise, works may also be carried out in simultaneously, with two points of construction (working heads), where possible. The estimated cost of construction of these underground rooms is about 356,000 €, or less than 450 € /per m' of excavation.

3. CONCLUSION

In the underdeveloped municipality of Gadžin Han preparations for the opening of the "Ćelije" calcite mine are being made. All necessary documentation has been prepared and works should begin in September 2017, based on the Major Mining Project (GRP). However, in order to obtain production of ore more quickly, by the explicit request of the Investor, a new Technical Mining Project (TRP) was designed, which envisages the excavation phase earlier, respecting the technical solutions defined in the GRP. In this paper the technical solutions for opening and development according to the new TRP are presented, with the description of the underground rooms and the technology of their construction

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DOI: 10.7251/BMC170701073R

VERTICAL SHAFT SINKING NOP II FOR A NEW VENTILATION PURPOSES IN VELENJE COAL MINE

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ABSTRACT

Vertical shaft sinking for underground transportation purposes is a very demanding technological process in mining and geotechnology, which requires specific and specially designed technological equipment. At the same time it is a technological process that humanity deals with from its beginning, since mining is one of the oldest industries in the world. The aim of the article "Vertical shaft sinking NOP II for a new ventilation purposes in Velenje Coal Mine is to show the technical challenge of this project.

Velenje Coal Mine has made up the decision for the sinking of a new ventilation shaft, depth of 393 meters. We started with the preparation works in 2011 and with a shaft sinking in 2012. The aims of the paper is to show the preparatory work before the start of the actual work, describes the equipment required for the vertical shaft sinking, it shows the primary and secondary support measurements and monitoring of mining work at such facility. With this article, we would like to represent the mining field of shaft sinking in Slovenia.

Key words: Vertical shafts sinking, new ventilation shaft

1. NEW VENTILATION SHAFT »NOP II«

Premogovnik Velenje – Velenje Coal Mine is an underground coal mine with average annual production around 4 million tons of lignite coal. In 2009 we started technical and economic studies for new ventilation shaft. New shaft "NOP II" is located closer to active production panels underground, closer to stock deposits and closer to power plant on the surface. After exhaustive study and revision, at the end of 2010, we got allowance for investment and approval to start operative activities on our project. You can find some basic technical characteristic for our new shaft in table 1.

Table 1: Technical characteristics for new ventilation shaft

Shaft inner diameter	6,15 m
Diameter of excavation	8,0 m
Shaft depth	393 m

Preliminary-works have started in January 2011. In first and second phase of pre - works we prepared infrastructures for construction site, we constructed shaft collar and we sank shaft to depth of 37 m using crane at open shaft collar. In April 2012 we started with assembly and installation of hanging platforms, top cover, temporary headframe and winch hall with nine winches. Complete set of shaft sinking equipment was ready in September 2012 when we started with shaft sinking. At this moment (September 2017) we are at depth 393 meters and we are in the end of shaft sinking.

2. EQUIPMENT IN THE SHAFT SINKING TECHNOLOGY

On location of new shaft detailed geologic exploration was done and we had some experience information from shafts which were sunk in the same area years ago. Therefore, we knew exactly what would be geological and hydrological conditions for shaft sinking. We decided to use conventional mining shaft sinking method; using hanging platforms in the shaft, drum hoists, drum winches and temporary headframe on the surface and other auxiliary equipment for dewatering, ventilation, gas monitoring, etc. Hanging platforms headframe and other steel structures were designed by experts at Premogovnik Velenje and manufactured at our subsidiary companies.

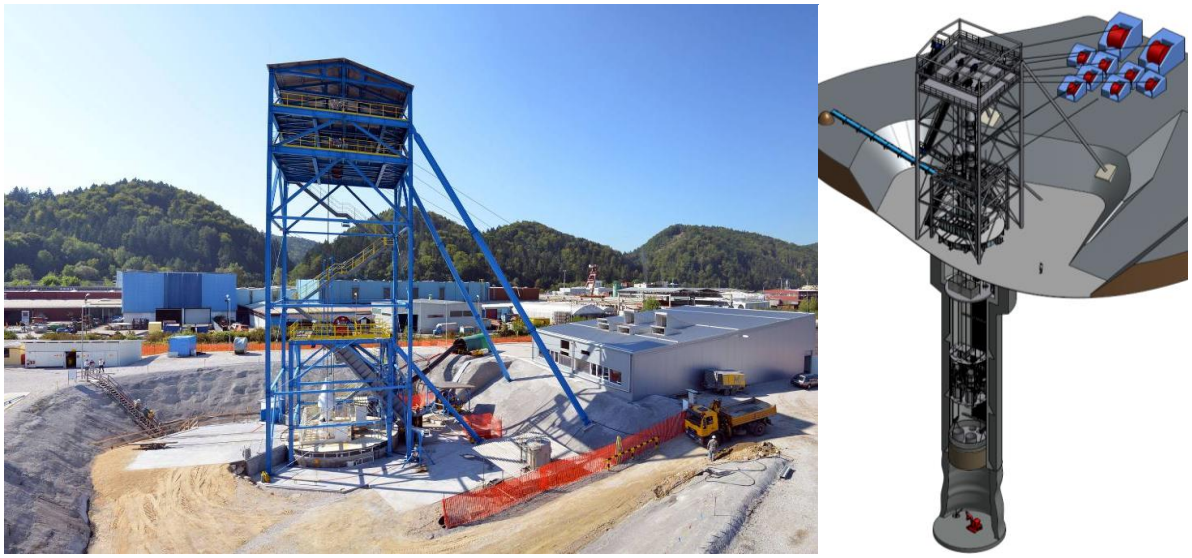


Figure 1: Shaft site – surface and cross section

Work site – bottom of the shaft

At the bottom of the shaft we installed electrohydraulic loader which can excavate profile of the shaft and load muck in the kibbles. Loader is small and movable on tracks; it can also rotate 360° around its axis. It can be lifted from the bottom by the main drum hoist and in few separate parts it can be transported through hanging platforms up to the surface. At work site you can find signalling and communication equipment, gas monitoring equipment, dewatering pump and pipelines, pneumatic tools. Fresh air is coming from surface, pushed by ventilator down the shaft, through 800 mm ventilation tube.

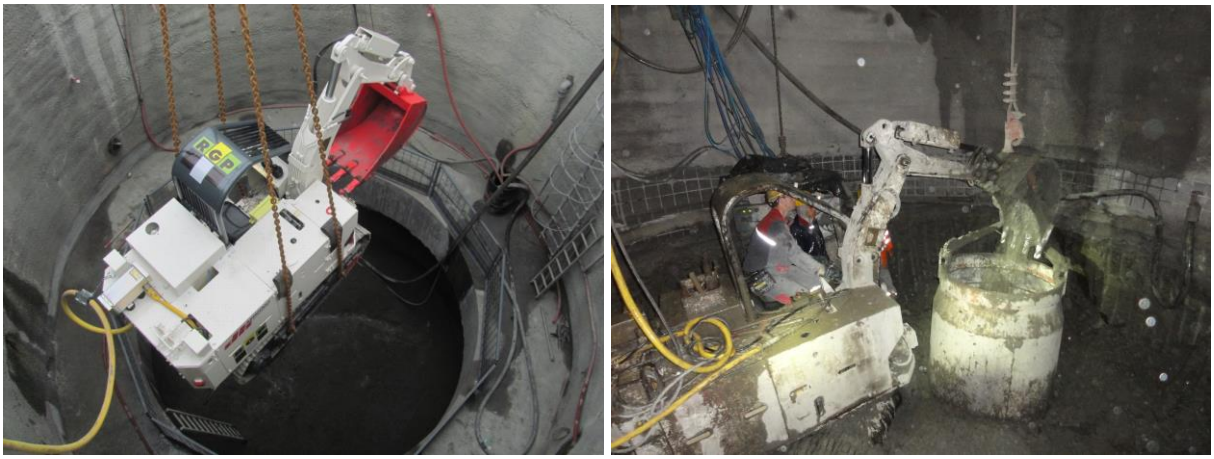


Figure 2: Shaft sinking site

Working platform

At maximum of ten meters above work site is positioned first hanging platform, so called working platform. Working platform has two levels. First level can be used as platform from which we can execute certain operations, but at the same time it is protection shield above work site at the bottom. Second level of working platform is construction formwork for final concrete lining.

Tensioning platform

Some 30 up to 50 m above work site is positioned second hanging platform, so called tensioning platform. Its weight is about 40 t, it has three levels, and its main purpose is tensioning of the guidance ropes. On the first level we have water container and dewatering pump and there is electric equipment for power supply to the work site. Tensioning platform is also used for installation of shaft insets including final ladder compartment. Ladder compartment is installed simultaneously as final concrete lining is done. Dewatering pipeline, pipelines for compressed air and shotcrete are installed at the same time. Beneath working platform and tensioning platforms we are using rope ladders as emergency escape way. Both hanging platforms have openings to enable kibbles to pass through on their way to the bottom/surface. Kibbles are guided by guiding frame - yoke which slides along guidance ropes to the tensioning platform; beneath tensioning platform kibbles are driven unguided.

Sounding line platform

Sounding line platform is fixed level about 10 m beneath surface. There we have installed bridge platform which carries the central axis sounding line. This is the most important sounding line used for shaft cantering. Additional on this level we have installed six more sounding lines. Three of them are already used for accurate installation of main holders of ladder compartment, and the rest of them will be used later at installation of guidance rails.

Top cover

On the surface, on shaft collar, we have top cover with 1,5 m high parapet wall, which protects shaft entry, allows used air exhaust and enables all infrastructure to enter in to the shaft. Top cover is massive steel construction which also carries half of load on tensioning platform. Tail ends of guidance ropes are connected to holders of top cover with special hydraulic system, which enables us to adjust tensioning force in each rope separately. On top cover we have two openings with pneumatic flaps, entry to inspection platform and to ladder compartment.

Headframe

On surface we have 37 m high headframe with deflection sheaves for all ropes. Headframe has smaller inner frame with manger slide for muck which is used for emptying kibbles. Particular pneumatic tool in headframe also enables us to separate kibble from guiding yoke to exchange kibble. We are using three different type of kibbles for man-riding and transportation of different material.

Winch hall and winches

We are using two independent drum hoists for man-riding and transportation of material. Then we have four winches to manoeuvre tensioning platform, two winches to move working platform and one more winch for manoeuvring main energetic cable. All together nine winches, which are installed on massive foundation in winch hall. In winch hall we have a special room for electric equipment and separate control room for winch operator.



Figure 3: Winches OLKO and SALUS

Safety signalling system and communication system

Modern safety signalling system was designed and installed by our personnel. System also provides digital and analog signals to safety circuit of hoisting winches and to visualization display on operators control panel.

For communication we are using three systems; conventional telephone line, open line communication units and portable wireless communication units. All three systems are designed to operate in explosive atmosphere (ATEX M1, Ex ia).

3. SHAFT SINKING TECHNOLOGY

Electrohydraulic loader on the bottom is used to excavate profile of the shaft and load muck in the kibbles. Profile is excavated in steps of 0,75 up to 1,5 m, depending on geologic conditions. For primary support we are using steel arches and shotcrete reinforced with steel mesh, which is installed in two layers. In difficult geologic conditions we were also using piles, anchors and some draining materials (mesh, geotextile, draining tubes). Achieved average daily advancing at excavation and primary support is 1,0 m/day. Every 20 up to 25 m we increase excavation cross section to make intermediate ring foundation with final inner diameter 6,15 m. Then we continue with shaft sinking and we excavate next 20 up to 25 m. While we excavate next section, we are surveying upper section (pre-installed extensometers, convergences). If there are no movements or deformations we can make final concrete lining.
For concreting we must move working platform with construction formwork for final concrete lining to last intermediate ring foundation and fix it in position for concreting. Final monolithic concrete lining is 500 mm thick and also reinforced with steel mesh; of course steel reinforcement must be installed in correct position before formwork is moved and fixed. Concrete with special characteristics is prepared in local concrete-plant and transported to shaft site. Further, down shaft, it is transported in kibbles designed for concrete transportation. One step of concreting final lining is up to 3 m high (which is determined by formwork height) and we are doing it in 24 hour cycle; so next day we move up working platform with formwork for next 3 m and execute concreting in next section. Depending on distance between intermediate ring foundations, and distance between openings for main holder for ladder compartment, sometimes we must adjust/reduce height of concreting step. Average efficacy at concreting final lining is 2,7 m/day.

When final concrete lining is finished, we move working platform in lower position. Then we make manoeuvre with tensioning platform and simultaneously we install final ladder compartment and all pipelines. Average efficacy at this phase is 7,5 m/day.



Figure 4: Shaft, ladder compartment

One shift crew is ten up to eleven workers including electrician and mechanic for basic maintenance. Work is running in three shifts, seven days a week. Average efficacy at complete shaft sinking is 0,6 m/day.

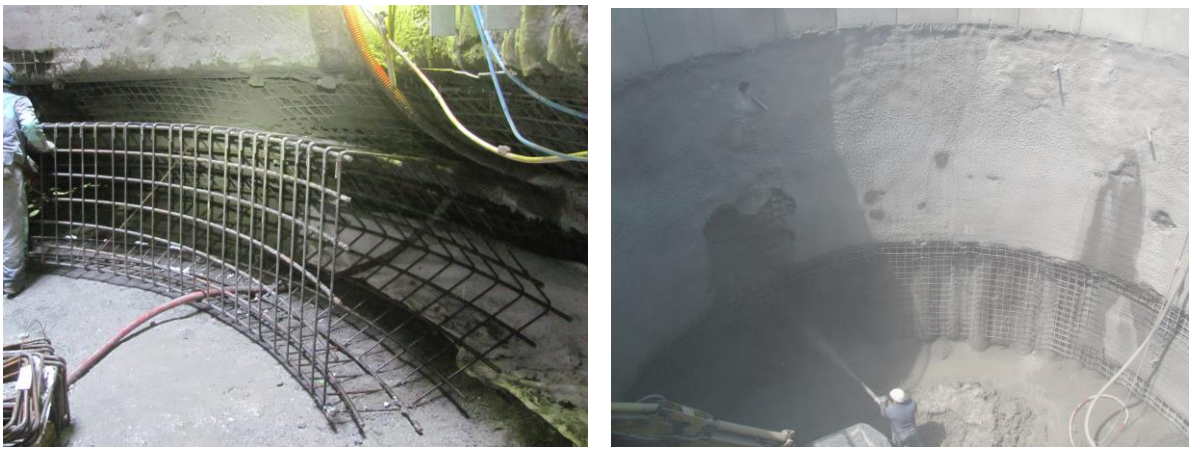


Figure 5: Support measures in shaft

4. GEOLOGY

As already mentioned, geology on location for new shaft was well known even before we started with the project. Despite this fact we execute additional exploration drilling. Borehole JUG – 48/09 (depth 521 m) was drilled in the profile of new shaft. Core from this borehole was used for exact geologic prospection.

From geologic prospection we knew that in first 100 m we will have to cross three packages of geologic layers which are containing water (sand, gravel). For more precise hydro-geologic survey another borehole, JUG – 49/09 (depth 175 m), was drilled in distance of 35 m from shaft centre. Water affluence was estimated in all three packages, and pressure sensors were installed in separate layers for latter monitoring. According to this hydro-geologic survey anticipation for water afflux during shaft sinking was 150 up to 320 l/min, from all three packages. Actual water afflux at excavation in first package (depth 28 - 39 m) was 80 l/min at the beginning and later dropped to 40 l/min. We were prepared for dewatering and this afflux cause no problem. In second package (depth 49 – 60 m) water afflux was much bigger. At depth of 52 m we had water and fine sands irruption which stopped us for several days. Water afflux was more than 200 l/min at the beginning, later reduced and stabilized at 80 l/min. Presence of gases was detected. At the time of first irruption concentration of methane - CH_4 rises up to 7 vol. %. We were pumping water and removing irrupted sand for several days before water pressure has dropped. We were constantly monitoring gas and water pressure by pressure sensors which were installed in borehole JUG – 49/09. With experience from second package, we were very careful when we were approaching third package (depth 85 – 95 m). We execute some additional exploration drilling, but there was almost no water afflux from this layer. Permeability to gas and water in this package was much lower and we crossed it with no problems. According to geologic prospection geologic conditions was good. We had several meters of more solid and hard layers but we didn’t have significant problems.

5. GEOTECHNICAL SURVEILLANCE

For such pretentious underground construction as mine shaft, it is very important to have adequate geotechnical surveillance. Certain technical parameters about construction itself and its influence to surrounding must be constantly monitored. To achieve demanded accuracy at shaft sinking geodetic surveillance is very important. Beside sounding and guiding at excavation and concreting the final lining geodetic surveillance also includes ground movement surveillance on the surface around the shaft. Additionally we have three vertical borehole inclinometers, which are located around the shaft, to observe possible horizontal movements in top 100 m strata.

Geologic and hydro-geologic observation is done on daily basis. Geotechnical profiles (extensometers, convergence profiles) are installed in primary support when needed (depending on geologic conditions). Of course, there are several more technical parameters which are measured and monitored. For instance technical parameters about concrete quality control, or technical parameters about ventilation in the shaft.

6. CONCLUSION

Shaft sinking project for new ventilation shaft at Premogovnik Velenje is running well. Shaft sinking is finished at depth of the shaft of 393 meters. We have successfully sunk the shaft through first 100 m, which was geologically most difficult part of whole 393 m. In this moment, we are in the last phase of the project.

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DOI: 10.7251/BMC170701081V

DRILLING EQUIPMENT FOR AQUIFERS DRAINAGE FROM COAL MINES WITH UNDERGROUND EXPLOITATION

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ABSTRACT

To ensure safe mining under aquifer layers is necessary as far as possible with drainage processes to reduce the water pressure in the aquifer above the coal layer. The water pressure is also one of the main input data for the calculation of the permitted height of excavation, therefore, to obtain realistic data necessary to carry out the appropriate monitoring of changes in water pressure, which is indicator of the effects of drainage. The solution results in making alternative underground facilities - impress filters. For the drilling of impress filters must be used properly drilling equipment. A method of making impress filter is the standard manufacturing process of the drilling impress filter in aquifers. The article dealt with the technological processes of drilling of impress filters and equipment necessary for the successful implementation of the projects of drainage aquifers. This equipment has been tested in Coalmine Velenje. Although it can be used anywhere with underground exploitation of coal.

Key words: underground exploitation, aquifer, impress filter, drilling,

1. INTRODUCTION

To ensure safe excavation under aquifers it is necessary to employ drainage in order to decrease water pressure in the aquifers above the coal layer. In calculating the permissible height values of excavation, water pressure is one of the most important input data [(1,2,3)].to obtain real data it is necessary to provide suitable monitoring of the water pressure, which is an indicator of water drainage efficiency. To improve the effects of drainage by using jetted dewatering boreholes, a possible solution was either to install more impressed filters on a shorter distance, or to encompass aquifers simultaneously through o borehole. Drilling with screens is possible if an inner tube is installed in the screen which will direct the flow of the drilling fluid directly through the drilling crown (Figure 1).

2. INSTALLATION PROCEDURE OF THE JETTED DEWATERING BOREHOLE INTO AQUIFERS

2.1. Construction of the conductor pipe

The conductor pipe is made in a standard way. A borehole for the conductor pipe is drilled with a drilling crown which is then cased with the conductor pipe. The length and the diameter of the conductor pipe will depend on the length of the jetted dewatering borehole and water pressure in the aquifers lying above the coal. Next, a tightness test of the conductor pipe needs to be carried out: water is pushed into the borehole via the liquidation flange with 50% greater pressure than the maximum anticipated water pressure in the aquifers. If the borehole does not provide the desired tightness, cement slurry needs to be injected, or the edges of the borehole need to be sealed down with cement. After completing the tightness test, and before proceeding with drilling, it is necessary to fit the mouth of the borehole with tube brakes, and a valve, as shown in Figure 2.

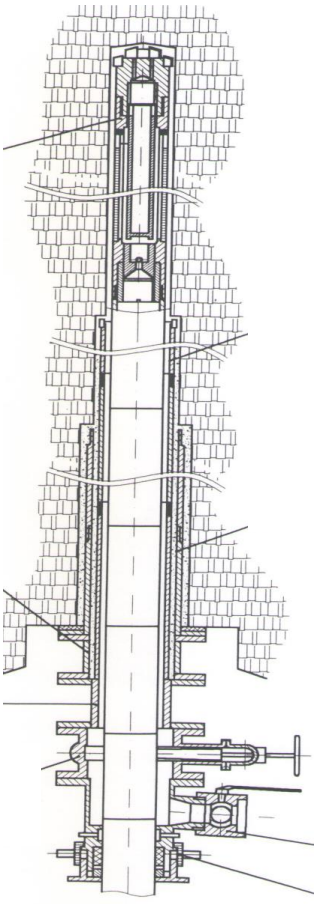


Figure 1: Impress filter



Figure 2: A flange of the conductor pipe, brakes, a valve, and a swivel pipe

2.1.2. Drilling and installation of the filter into the aquifer

Drilling through the conductor pipes is carried out in a standard way, using drilling pipes with diameter of ø128 mm and a double-layer screen with ø 128/113 mm diameter. The drilling crown of the screen is composed of two parts: a fixed crown with cutting diameter of ø148 mm, which remains on the screen, and a removable crown fixed to the seal pipe. After fixing the swivel pipe behind the valve (the swivel needs to correspond with the diameter of the pipe of ø 128 mm), drilling can be continued to a desired depth using a suitable polymer drilling fluid. In the Velenje coal mine we used Modipol 600, a polymer additive (powder), or Argipol, which is a liquid polymer additive. The first layer of the aquifer is activated by loosening the seal head and the removable crown. Figure 3 shows an adapted transportable unit K1 for preparing the drilling fluid, while Figure 4 shows the first filter with diameter of 128 mm with the drilling crown (cutting diameter 148 mm), and the removable crown.



Figure 3: Preparation of the drilling fluid



Figure 4: Filter with ϕ128 mm diameter, a drilling crown with cutting diameter of ϕ148 mm, fixed to the screen, and removable crown.

3. EQUIPMENT FOR THE CONSTRUCTION OF JETTED DEWATERING BOREHOLE INTO AQUIFER LAYERS

The following equipment is needed for the construction of jetted dewatering borehole:

• a RVS-Max drill	45 kW,
• ČGZ 250 Pump, or TRIDO 220 pump	22 kW,
• Submersible pump [3 m^3/min]	37 kW,
• Flygt Submersible pump	2.9 kW,
• suitable pipe armature	
• supply of technological water and wastewater drainage	

RVS-Maxdrill

The RVS-Max drill is used for drilling in mine workings and in methane conditions, and particularly for drilling of boreholes for the drainage from water-bearing layers. The RVS-Max drill is manufactured by HTZ Velenje.

Basic components of the RVS –Maxdrill

1. Hydraulic drive unit (PICTURE 5), which is an independent transportable unit, driven by a 45 kW (500 V) electric engine. Secondary drive is provided by two hydraulic pumps.



Figure 5: Hydraulic drive unit

2. A mast with integrated extension and double clamping jaws (PICTURE 6). The mast facilitates the movement of the carriage with rotation head and provides movement of the drilling pipes forward and backward during drilling. The stroke-length of the head allows for the use of pipes with 1500 mm in length. The pull-down and the pull-back force is transmitted to the pipes via a hydraulic cylinder and a stud-link chain Wipermann ¾". The extension of the mast is integrated, with double clamping jaws attached to it.
3. A system for fixing or supporting the mast (hydraulic - mechanical support).
4. Control panel with driving and control tools (PICTURE 7).
5. The rotation head on the carriage is a multi-stage reduction gear, driven by two hydro engines, which can be started in three different modes.



Figure 6: A mast with the rotation head, clamping jaws and the system for fixing the mast



Figure 7: Control desk

Technical data:

Rotating head stroke	1500 mm
Mast length	2500 mm
Max. speed	0.3 m/s
Operating oil-pressure	180 bar
Max. thrust force	4500 N
Machine weight	1700 kg

Level	Moment [Nm]	Rotation [rev/min]
I (M1+M2)	3150	100
II (M2)	1240	254
III (M1)	1930	163

Clamping jaws: technical data

	Clamping diameter [mm]	Clamping force [N]
Front jaws	42 – 73	6900
Rear jaws	89 - 163	6900

4. CONCLUSION

To ensure safe mining under water-bearing layers it is necessary to provide drainage to minimise water pressure in the aquifers lying above the coal layer. Water pressure is one of the main entry data in calculating the permissible height of excavation. This means that it is necessary to introduce suitable monitoring methods to control changes in water pressure which is the indicator of the effects of drainage. Since by excavation works in the north-western part of the Preloge (Coal mine Velenje) coal field the system of surface observation wells and drift filters has been damaged to a great extent, it was necessary to find a suitable replacement [(4)]. This was done by installation of jetted dewatering boreholes. From the existing jetted dewatering boreholes, which dewater just one aquifer, we have developed a completely new technology for jetted dewatering boreholes installation which would drain aquifers simultaneously, while each of the aquifers is drained separately. In this way we have significantly shortened the time needed for the construction of a jetted dewatering boreholes drainage system and thus made the construction of drainage facilities more cost-effective [(5)].

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DOI: 10.7251/BMC170701087B

THE SELECTION OF COAL EXCAVATION AND TRANSPORT SYSTEM FOR THE UGLJEVIK EAST 2 OPEN PIT

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ABSTRACT

Surface mining implies the integration of a significant set of activities. Based on the project goals and numerous relevant factors every activity is individually optimized. Moreover, in order to ensure the functioning of a whole system and success of the mining project, each of individual optimization must be an integral part of global optimization. This is particularly true for the selection and definition of excavation and transport system. The reason for this lays in the fact that a significant share of the total production cost is associated with these two operation.

This paper presents a coal excavation and transport system for Ugljevik East 2 open pit. Results of the analysis suggest that in the conditions of open pit Ugljevik East 2, which are characterized by complex geological structure, as well as significant distances between coal excavation sites and stockyard, the selection of a combined system provides the best results.

Key words: excavation, transport, conveyor, dump truck, coal

1. INTRODUCTION

During the mine planning process the one of the most important activity is the selection of the proper loading and hauling system, a topic that many authors have dealt with[1] [2] [3].This is especially true for the futureUgljevikEast 2open pit, because it has specific mining conditions with the structurally complex deposit, difficult terrain morphology and long transportation distances.

2. LOCATION AND REGIONAL GEOLOGY

The UgljevikEast 2 coal property is located near the town of Ugljevik, situated in north-east part of the territory of RepublikaSrpska, part of Bosnia and Herzegovina. Exploration in this area had begun in 1977, lasted until the end of 1990, and was reestablished in 2010 and 2011. In first two phases 157 holes were drilled, and after the year 1985 aiming to re-categories

resources from C1 to A and B category, 260 holes were drilled. During 2010 and 2011, a new exploration phase began. There were 62 boreholes drilled. Total length of all boreholes is almost 30000 m. [4]

The geology of the deposit is complex and extremely tectonically controlled with several generations of faults. There is two coal seams upper and lower. The upper coal seam is layered up to 4 plies and lower coal seam is layered up to 5 plies.

3. LOADING AND HAULING COALTO THE CRASHING PLANT

Geological characteristics, terrain morphology and open pit capacity are the most significant influence factors on the choice of loading and hauling systems. In addition, in this case, it is very important the fact that the distance from the open pit mine to the power plant stock yard is great. The route for the transportation of coal can be divided into two parts: within the open pit mine and outside of open pit mine, but it should be noted that the part of the route outside the mine is extremely complex.

The overburdenwill be removed by crawler dozers exposing the top of coal. When the waste mining operations has advanced far enough, along strike the coal, will then be loaded out either from below, with the hydraulic shovels in front shovel configuration or possibly from above with a shovel in backhoe configuration. The 8m^3 hydraulic shovels will load the majority of the coal. The coal will be loaded into the 90-tonne trucks and hauled to the crasher location.

The design considerations for the haul roads should be based on an overall traffic management plan. The purpose of this plan is to separate heavy mining equipment and light vehicles/personnel wherever possible [5]. For the purposes of this work, haul roads were assumed to be 25m, in width. The maximum gradient used for the haul roads was 10% [6]. Gradient will be a critical design parameter as the haul trucks will be required to traverse the maximum down gradients in both an empty and loaded state.

In Figure 1a mining fleet for coalis presented.

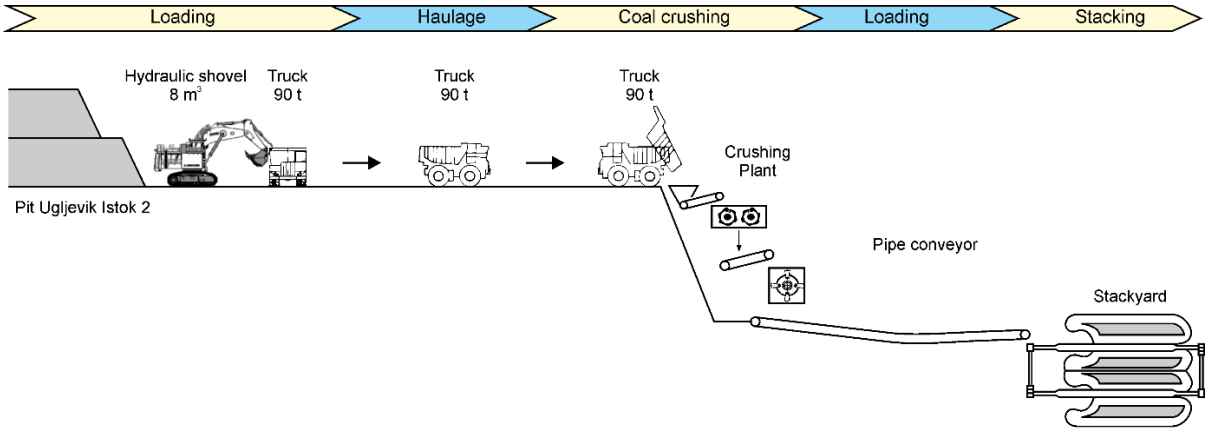


Figure 1. Coal mining fleet

Shovel Selection

The design basis assumes 5 to 9 shovels (depending on the year of production) as an optimum fleet size of loading system for overall mining operation (waste and coal). This number of shovels ensures that planned capacities will be provided with minimal capital costs.Two to four 8 m^3hydraulic shovels or equivalent have been selected to excavate raw coal mining requirements to meet the 2.06 million tonnes production target at a strip ratio of 6:1 m^3/t. The shovel 8 m^3 units are matched to load the 90-tonne haul trucks, in eight passes.

Truck Selection

The coal will be transported along roads cut through the mining area to give a suitable gradient path back to the coal crashing plant. The terrain in parts of the project area is very steep and will require goodconstructed and designed roads to achieve the haulage targets.The design and construction of the haul roads is seen as a critical component of the ability of the project to achieve the required production in a safe and cost sustainable manner.

Haulage equipment was selected based upon production needs, and Talpac [7][8] was used to calculate fleet production. Multiple haul cycles were measured for different stages in the mine life. These haul routes, as well as rock loose and blank density, were put into Talpac. All load and dump times are based off the Cat performance handbook, as well as the Talpac database.Truck productivity is based on truck-cycle times from the pit to predetermined destinations – crusher. Each truck selected for the coal haulage has a 90-tonne payload.

Optimisation of Truck/Shovel Fleet

The term optimum reflects that the selected equipment must comply with the mining conditions/limitations and meet the basic requirements and preferences of the mine[9]. The model of the equipment presented is according to the analysis of cost and operating parameters, based on the experience in operating such equipment in similar working conditions and manufacturers of mining equipment.

RUNGE Talpac, a truck and loader analysis software package was used to optimize the haul cycle and loader selection, as well as the estimating haulage costs. Multiple haul cycles were simulated in Talpac for different stages in the mine life.Table 1 shows the required mining fleet for the coal mine plan through the life of the mine.

Table 1. Truck-shovel Mining Fleet

Truck Fleet																													
Year	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**
Trucks (90t)	0	1	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	5	8	8	8	8	8	8	8	8	8

Shovel Fleet																													
Year	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**
Shovels (8 m^3)	0	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	4	4	4	4	4	4	4	4	4

Over the mine lifetime, which is approximately 28 years, each shovel will operate approximately 50,000 h, which is considered to be a useful life cycle,compared to industry averages. Major maintenance overhauls will take place after 17,000 hours in each unit's life cycle. The estimated average life cycle of the haul trucks is 60,000 operating hours, each unit will be rebuilt at the 20,000 hour point, necessitating a total of three major maintenance rebuilds per truck, taking two weeks per unit throughout the mine life.

4. COAL CRUSHING

After the transportation of coal with trucks from the open pit, it is necessary to prepare coal in a crushing plant,for further transport to the stockyard of a thermal power plant. The starting parameters for defining the capacityof crushers are: annual pit capacity on coal exploitation, andannual time commitment on coal excavation.For an annual capacity of 2060000 t of coal and expected average commitment on coal excavation of 4000 hours per year, the average capacity of a crushing plant should be 500 t/h. It is usual to take a little bit higher hourly capacity of crushers, so that it does not have an impact on pit's operation efficiency. Given that the transport of raw coal from location of excavation to the crusher will be done by trucks, it is desirable to dimension crusher in order to take full quantity of coal from one truck, so that coal can be crushed before another truck arrives. Taking into consideration both reasons crushing plant is dimensioned on an hourly capacity of 700 t/h.

5. SELECTION OF CONVEYOR SYSTEM

Crushed coal size of -30+0 mm, which is the required coal size on input of stock yard of future power plant, may be transported in several ways, both continuous and cyclic. After a detailed analysis of possible ways of coal transport continuous transport is chosen. There can be two possible conveyor systems: one, with closed pipe conveyor and the classical belt conveyors. Out of these two, pipe conveyor was chosen.

Pipe conveyor, represents a classic rubber belt which is bent into a pipe form.Due to the suitable configuration of the roller and the appropriate belt quality, it keeps its pipe form, from the loading point to the unloading point[10]. So, the whole conveyor route, with a length of 6676 m is handled with one uncut conveyor which is capable to handle all horizontal and vertical curves on the route.In the whole route (from open pit to the stock yard) there are 11 curves with changeable radius, as shown in Table *2*.

Table 2. Radius of curves

Horizontalcurve	Min. radius[m]	Horizontalcurve	Min. radius [m]
1	280	7	300
2	205	8	285
3	205	9	275
4	215	10	270
5	230	11	600
6	300		

In order to make the transport operational, a bridge support structure, with an approximate length of 600 m will be required. For the capacity of 700 t/h the belt width should be 1200 mm, following the diameter of the pipe conveyor of around 310 mm, at a conveyor speed of 5.0 m/s. For its drive it is required to install 4 × 560 kW motors.

The area between the designed position of crushing plant and the coal stockyard at power plant is characterized by a terrain of complex configuration with the presence of important infrastructure. All of these have significantly affected the designed route of the coal transport system. It was sought that the designed routes of the conveyors provide an efficient and safe coal transport, from crusher to the coal stockyard. In Figure 2, the conveyor route is shown.

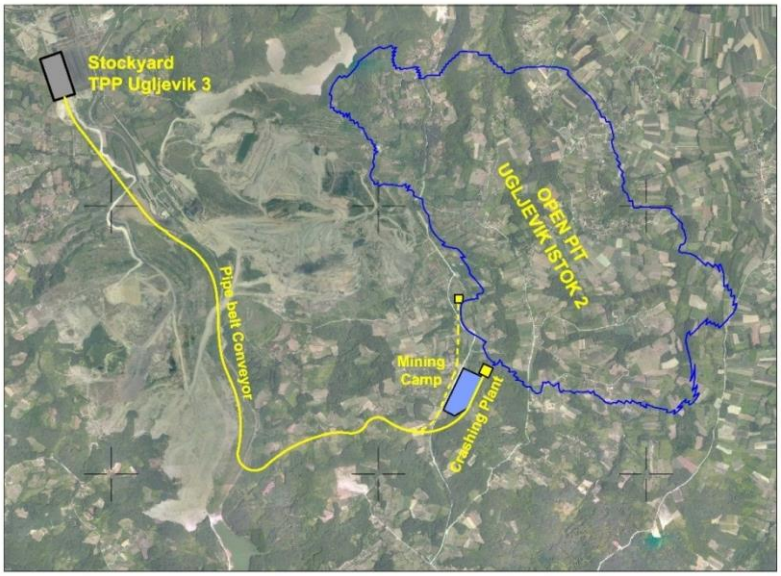


Figure 2 Disposition of designed system route for coal transport to power plant

The location of the processing plant is South-West of the pit, close to a mine camp. Nearby elevation, where the pipe conveyor starts equals to 272 RL. Throughout the whole route conveyor is declining, because the output elevation on the stock yard is 180 RL.

6. CONCLUSION

Based on the needs of a new thermal power plant, it is necessary to open a surface coal mine Ugljevik East 2 and deliver coal to the stockyard.

Coal excavation will be carried out with up to four 8 m^3 hydraulic shovels or equivalent. Considering the complexity of the terrain route for the transport of coal, it is divided into two parts. From the open pit to the crushing plant coal will be transported with 90-tonne payload trucks. Further, with closed pipe conveyor. Although the use of pipelines in our conditions is unusual, it should be noted that with the usage of a pipe conveyor, a secure transport is achieved in all realistically expected climate and weather conditions, with complete fulfilment of all ecological requests, especially the ones linked with dust and spilling. Possible dust on the load is solved by de-dusting system in closed area, as well as during unloading. Before conveyer leaves the closed part, it is reduced in a closed pipe form, in the way that air pollution is being prevented.

In order to further optimize this system, using stockpiles at the crashing plant should be considered. This will provide possibility of coal blending [11] if necessary and the mine with additional flexibility to schedule the coal haul around coal releases and shovel maintenance. When the coal haul is not scheduled the haul trucks will be redirected to the waste hauls. This approach allows the fleet capital to be minimized and the utilization of the trucks to be maximized.

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DOI: 10.7251/BMC170701093N

DRILL HOLE DEVIATION IN SURFACE MINE BLASTING

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ABSTRACT

The paper deals with the issue relating to the deviation of drill holes during the rock blast operations in surface mines. Accurate drilling is of primary importance in every rock drilling process. It results in improved rock fragmentation and lower overall cost. Each hole should be drilled in the correct direction to the proper depth according to the designed drill pattern grid. However, during the drilling process, the actual drill hole path often deviates from the planned one, more or less. Many factors influence the drill hole deviation such as the rock mass characteristics, the technical parameters of drilling equipment, the experience of drill operator, the angle drilled, geological conditions etc. Drill hole deviation can be measured. Nowadays, there are many precise deviation measurement systems. Many problems may occure if blast operators proceed to charge blast holes on the basis of the designed drill pattern grid on the assumption that holes are straight, rather than the actual drilling results. Drill hole deviation may cause bad blasting results and increase blast and operating costs. Deviation in drilling has also an impact on flyrock accidents, injury to workers and property damage. The factors that lead to deviation during drilling, the consequences of deviation and measures to be taken to minimize drill hole deviation will be presented in this paper.

Key words: deviation, drill hole, rock blasting, cost, surface mine

1. INTRODUCTION

In surface operations, drilling is performed for one of the following reasons: to construct space in the rock, to exploit the material being excavated and to use the drilled holes for special purposes other than blasting (water well drilling, prospect drilling, anchor hole drilling, pumping hole drilling etc.)[1]. Rock drilling in surface mine blasting is the first operation where the holes are drilled in order to enable the placement of the explosive charges. Mechanical drilling systems are widely used in surface mine blasting especially rotary-percussive methods (the top hammer as well as the down-the-hole-hammer) and rotary

methods (subdivided into two groups: rotary drilling with rolling tricone bits in medium to hard rocks and rotary drilling with cutting action in soft rocks) [2]. Accurate drilling is very important issue in every drilling operation. Taking into account that the drilling operation is the first step in surface mine blasting, the quality of execution of the drilling process directly affects on the quality of blasting operations as well as futher operations: loading, hauling, crushing etc.

2. DRILL HOLE DEVIATION

2.1 The factors that lead to deviation during drilling

Hole deviation is the unintentional departure of the drill bit from a designed path. There are many factors influencing hole deviation in bench drilling operations: the geology of rock mass, the drilling method, the design of drilling equipment, the level of training of drill operator, the proper setting of drilling parameters according to working conditions etc.

Figure 1a illustrates various causes of hole deviation [3]. The magnitude of in-hole deviation is exponential to the hole depth and according to Atlas Copco, the deviation is proportional to the depth to the power of two [3]. Geological conditions are a major cause of in-hole deviation during drilling [3]. Figure 1b illustrates the influence of bedding and foliation on hole deviation. The drill hole tends to deviate to a direction perpendicular to the jointing, and the more structured, foliated and faulted the rock, the greater will be the deviation [3].

In order to increase blasting efficiency, it is necessary to adjust the blasting parameters which are mostly depending on the quality of execution of the drilling process. If there is an extensive deviation between the actual and planned drill pattern grid, the values of blasting parameters will also vary from the planned. Therefore, the blasting results will not be as expected.

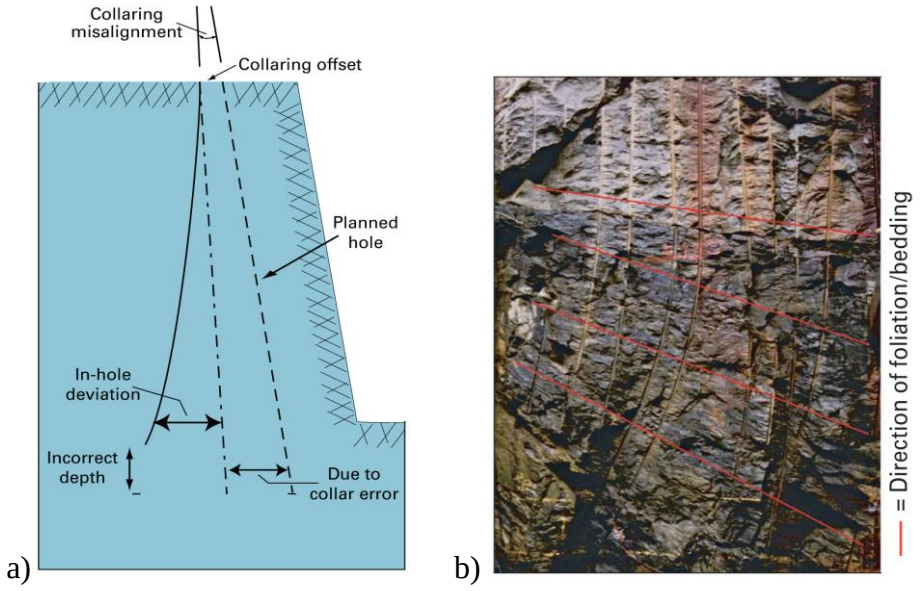


Figure 1: a) Various causes of hole deviation, b) Deviation caused by the direction of foliation/beding [3]

During the drilling process, it is very important that the start-up or so called collaring is performed in a correct manner. Collaring should be carried out carefully by adjusting the drilling parameters until the drill bit reaches the proper hole depth to ensure a correct direction, before full feed force is used.

The level of training of drill operator is also very important factor influencing the hole deviation. In earlier period, the adjustment of drilling parameters according to working conditions was mostly depended on the experience of drill operators. Nowadays, there are many types of modern drill rigs equipped with sophisticated tools for surveying all drilling parameters both during drilling operations and after ending drilling process in order to increase the precision of drilling operations.

2.2 Consequences of deviation

The main consequences of hole deviation are: poor fragmentation of blasted material, possible misfires and flyrock, higher intensity of ground vibration induced by blasting, excessive or too small burden and spacing between adjacent blastholes, secondary breakage of blasted material, uneven bench floors, shorter equipment and tool life, lost of drill string components, higher overall costs (drilling, blasting, loading, crushing, hauling, maintenance costs etc.).

Every blasting operation is designed according to designed drill pattern grid. When the deviation of drilling is high, the actual drilling result can differ from the designed one. In case when blast operators proceed to charge blast holes on the basis of the assumption that holes are straight, many problems may occur. In-hole deviation leads to significant increase or reduce the burden as well as the toe spacing of adjacent holes. A reduced burden results in flyrock - the uncontrolled ejection of fragments of blasted rock material from the blasting site. Flyrock from surface blasting operations may cause the damage of equipment and serious injury to employees and other persons. Excessive burden causes blast induced ground vibrations. According to Jimeno et al. [2], if burden is excessive, the energy from the explosion has too much resistance to effectively fracture and displace the rock, therefore part of the energy becomes seismic. Drill hole deviation causes the poor fragmentation of blasted material leading to further secondary breakage of boulders by mechanical means or other special methods.

2.3 Measures to be taken to minimize drill hole deviation

Although the geology of rock mass is a factor that can not be influenced, certain measures can be taken to minimize hole deviation. According to Atlas Copco [3], small clearance between the hole and the drill string components as well as a stiff drill string increase the straightness of the holes. Better control of deviation can be achieved in low benches i.e. less hole depth [3]. Also, a combination of reduced feed force and bit load with increased rotation speed gives less deviation [3].

According to Atlas Copco [3], straighter holes can be produced by COPROD with a combination of drill rods and large diameter drill tubes and by DTH (down-the-hole) drilling with large diameter drill tubes than any type of tophammer drill string due to stiffer string and better hole guidance through the rock (Figure 2).

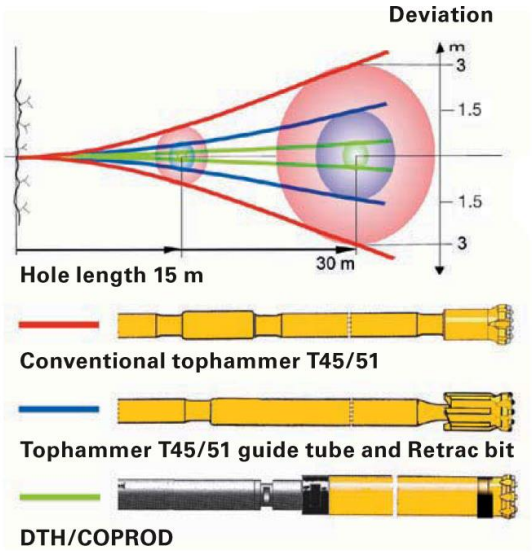


Figure 2 The influence of different types of drilling equipment on hole straightness [3]

The COPROD system is the hybrid system which combines the speed of tophammer drilling with the precision and long service life of DTH [3].

An acceptable level of drill hole straightness can be reached by the proper selection of drilling tools. According to Atlas Copco [3], crossbits generally result in straighter holes compared to button bits while the ballistic button bits in most cases result in straighter holes than spherical button bits. A concave bit front also results in straighter holes than a flat front [3]. Retrac bits are self-guiding bits and act as a short guide rod directly behind the bit front to reduce deviation. The combination of a guide rod or tube installed behind a retrac bit can eliminate more than half of the in-hole deviation, compared to a conventional tophammer drill string [3].

According to SANDVIK [4] starting off with Sandvik MF extension rods and – as the conditions and situation demand – fitting the drill string with Sandvik Retrac bits, Drop-Center bits, Guide bits and Guide/Pilot tubes, the bench drilling operations step by step can be improved to decrease hole deflection to and even below 2 %
Nowadays, the precision in drilling is enabled by modern drill rigs equipped with sophisticated tools for surveying all drilling parameters both during drilling operations and by transferring logged data from the rig after ending drilling process. ROC Manager [5] produced by Atlas Copco is a stand-alone PC-based tool for making drill plans, measuring hole deviation, and logging, presenting and reporting drilling data graphically. This information can be presented individually or in combination with other parameters. A number of graphical tools of ROC Manager can be customized at site for getting better insight in local rock conditions such as discontinuities, faults, and boundaries. According to Atlas Copco, the benefits of the application of ROC Manager are: accurate drill plans, better blasting results, safer blasting, clear presentation of logged drilling results, if available, graphical representation of MWD (Measure while Drilling) data that can be used to analyze the kind of rock formation [5].

The quality and straightness of the drill holes as well as the service life of the tools can be achieved by the proper use of sophisticated monitoring system in the drill rig like Tamrock's Rock Pilot System [4]. The super-fast control reacts to changing rock formations and adjusts the drilling power automatically sending the optimal amount of power to the rock [6]. The Rock Pilot drilling control system enables smooth drilling, straight holes and excellent performance even in the most difficult ground conditions [6]. According to SANDVIK [4], with such a system in operation and by carefully choosing the best suited drilling tools – hole deflection of less than 2 % is within reach.

2.3.1 Borehole deviation measurement systems

Borehole deviation measurement systems are used for surveying drill holes and measurement of borehole deviation in order to achieve more safe and efficient drilling and blasting operations according to plan. There are many systems for measuring the borehole deviation.
The Cabled Boretrak probe (Figure 3a) contains a dual axis tilt sensor and digital compass, to calculate borehole deviation from the collar position at fixed intervals [7]. It is compact, lightweight, rugged and highly portable. It is designed to be deployed downhole on its integrated, toughened cable, to depths of up to 65 metres [7].

The Rodded Boretrak (Figure 3b) is non-magnetic system for measuring borehole deviation [8]. It uses specially designed rods to guide and locate the probe in place of a compass [8]. Rodded Boretrak can be apllied to measure borehole deviation in situations where other systems will not work: in underground mines, areas of ferrous materials and uphole as well as downhole [8]. In case of measuring borehole deviation to great depths, the cabled probes are limited to the fixed cable length while the Rodded Boretrak has the possibility of simply adding additional stacks of rods [8]. The specifications of the Cabled Boretrak and Rodded Boretrak are presented in Table 1.

The Boretrak Viewer [9] is software package for the Cabled Boretrak and Rodded Boretrak borehole deviation measurement systems. This feature-packed software package [9] enables wide range of possibilities of analysis of measured data such as: process uphole and downhole from the same download, process and view multiple surveys per hole, a hole design module assists planning / importing hole layouts and design holes, combine multiple datasets from different probe types, single-click report generation on all holes within the project etc. (Figure 3c).

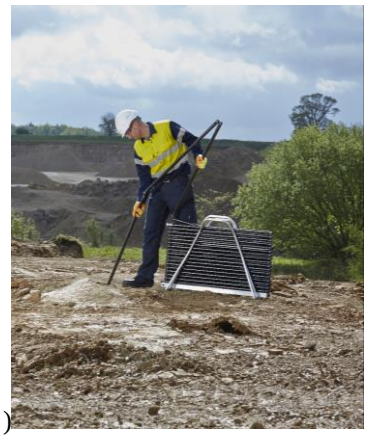
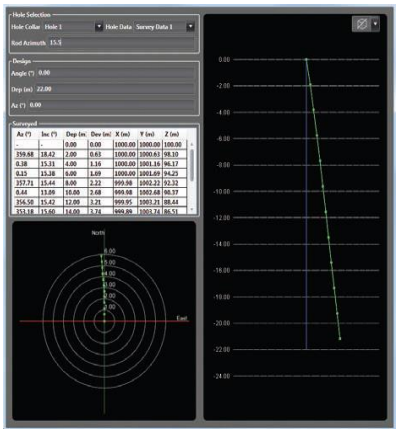
a)  b)  c) 

Figure 3 Borehole deviation measurement systems: a) Cabled Boretrak, b) Rodded Boretrak, c) Boretrak Viewer software [7,8,9]

Table 1 Specification of Cabled Boretrak and Rodded Boretrak [7,8]

Cabled Boretrak	Rodded Boretrak
Probe construction: 316 stainless steel body and brass insert	**Probe construction:** 316 stainless steel
Probe Weight : 2.5 kg	**Probe Weight** : 2.5 kg
Probe Dimensions : 480 × 38 mm (L × Dia)	**Probe Dimensions** : 660 mm × 37 mm (L × Dia)
Ballast Rod Weight : 960 g	**Rods Weight:** 400 g
Ballast Rod Dimensions:1000 × 38 mm (L × Dia)	**Rod Dimensions**: 1000 mm × 25 mm (L × Dia)
Sensor: Dual axis inclinometer: Range: +/– 40˚ from the vertical, Accuracy: 0.2˚, Resolution: 0.1˚	**Sensor:** Range: +/– 45˚ from vertical – upwards or downwards,Accuracy: 0.1˚, Resolution: 0.01˚
Compass : Range: 0 to 360˚, Accuracy: +/– 1˚ typical (vertical and calibrated), Resolution: 0.1˚	
CDU Display: 128 × 64 graphic backlit	**CDU Display:** 4 lines × 20 characters
CDU Memory: 64 KB	**CDU Memory**: 64 KB
CDU Data download : RS232 (selectable baud rate)	**CDU** RS232 (9600 baud rate)
Power system: 3 × 1.5V C (MN1400, LR14) cells	**Power system: Probe:** 3 × 1.5V D (MN1400, LR14) cells, **CDU:** Internal rechargeable batteries – 1000 recharge cycles, **CDU charger :** 100-240 V ac input
Probe: Waterproof (pressure rated to 100 m)	**Probe**: Waterproof (pressure rated to 300 m)
Operating temperature : 0 ˚C to +45 ˚C	**Operating temperature :** –10 ˚C to +60 ˚C
Storage temperature : –10 ˚C to +50 ˚C	**Storage temperature :** –20 ˚C to +70 ˚C

2.3.2 Waterproof video probe

Waterproof video probe (Figure 4) has been developed in the Blasting Center at the Faculty of Mining and Geology, Belgrade, Serbia [10]. The probe fulfils all the requirements regarding the complex conditions that exist in the drill holes. It is protected against mechanical damages and from falling of some loosened rock pieces into the hole. The probe housing has an outer diameter of 50 mm and a length of 500 mm. Video probe can be used in depths up to 100 m. The probe enables the measurement of hole deviation.

A waterproof video probe can be applied for drilling the blast holes as a control of the drilling accuracy, before and during the loading the blast holes with the explosive charges, to check the condition of the drill hole (water presence, the existence of a cavity or other discontinuities in rock) in all types of surface mines where the diameter of drill holes is greater than 50 mm. The probe has also the application for geological explorations (the examination of cracks systems, layer thickness, distances between certain discontinuity zones, etc.), repair of exploitation wells, underwater testing, speleological research to a depth of 100 m.

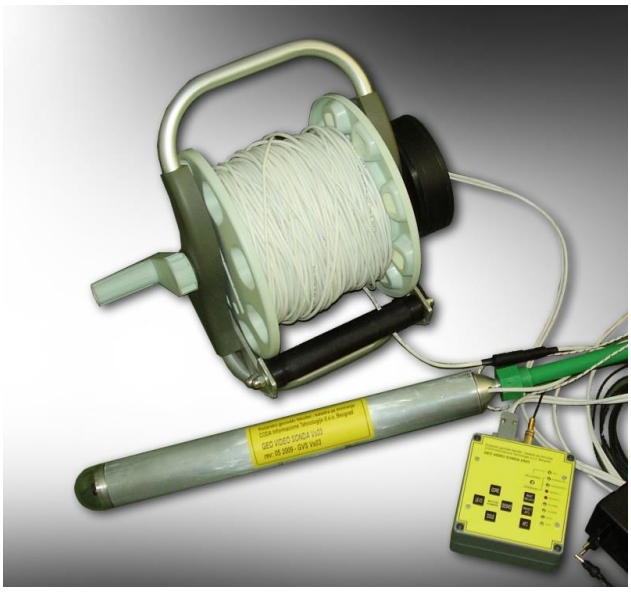


Figure 4 Waterproof video probe [10]

3. CONCLUSIONS

In order to achieve better blasting results, holes within the same blasting field should be drilled according to the designed path along the entire length. The drilling process of each hole must be started in the exact spot while holes are drilled in the accurate direction and to the proper depth. In other words, the precision of every drilling operation is very important issue.

The proper selection of drilling methods and drilling equipment as well as the adjustment of drilling parameters according to geological conditions may significantly reduce the deviation of drilling and improve the fragmentation of rock mass during blasting.

The drilling accuracy can be achieved by modern drill rigs equipped with sophisticated tools for surveying all drilling parameters both during drilling operations and by transferring logged data from the rig after ending drilling process. Straight hole drilling enables: the optimization of drilling and blasting parameters, better blasting control, good blasting results (achieving good fragmentation of blasted material without overbreak, flyrock, misfires, blast vibrations etc.), lower maintenance cost of drilling equipment as well as lower overall costs.

4. ACKNOWLEDGEMENT

The authors would like to thank the Ministry of Education, Science and Technological Development of the Republic of Serbia for supporting Projects of Technological Development TR17013 and TR 33003.

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DOI: 10.7251/BMC170701101A

NEW TECHNOLOGIES APPLIED FOR SMARTER BLASTING IN RUDNIK KREČNJAKA CARMEUSE A.D. DOBOJ

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ABSTRACT

The use of explosives remains the most efficient way to fragment rocks and therefore blasting operations are essential in all large-scale mining, quarrying or civil construction excavations. However, as only a part of the explosive energy is used for the fragmentation and the displacement of the rock-mass, improperly designed blasts may cause undesirable side effects like noise, fly rocks and vibrations or result in inappropriate muck-piles.

This paper presents the current drilling and blasting practices from the Carmeuse's limestone quarry in Doboj (Bosnia and Herzegovina) and the initiative to introduce new technologies in the field of drilling and blasting preparation. Experiences from other Carmeuse's operations will give a flavor of our future developments.

Key words: drilling, blasting, survey.

1. INTRODUCTION

For non-specialists, a blast is nothing more than drilling the holes, filling them with explosives and initiating the explosion … but, of course, it is a much more complex challenge. A rigorous analysis is required to determine the right balance between various parameters such as burden, spacing, drill hole diameter and inclination, bench height, rock-mass qualification and explosive energy. Only proper design will lead to optimal muck-pile shape and fragmentation, absence of bench toes and low side effects (vibration, noise, air impact) at an optimized cost.

Because of the complexity and importance of the drilling and blasting operations, Carmeuse invests in the continuous improvements of these operations and promotes the sharing of experiences.

This paper presents the preparation practices applied in the Hardovac quarry, near Doboj (Bosnia and Herzegovina), where are mined Eocene limestone suitable for the production of lime, chemical stone (for sinter plant, FGT, sugar industry, etc.) and aggregates for civil construction.

2. DRILLING AND BLASTING PRACTICES IN HARDOVAC QUARRY

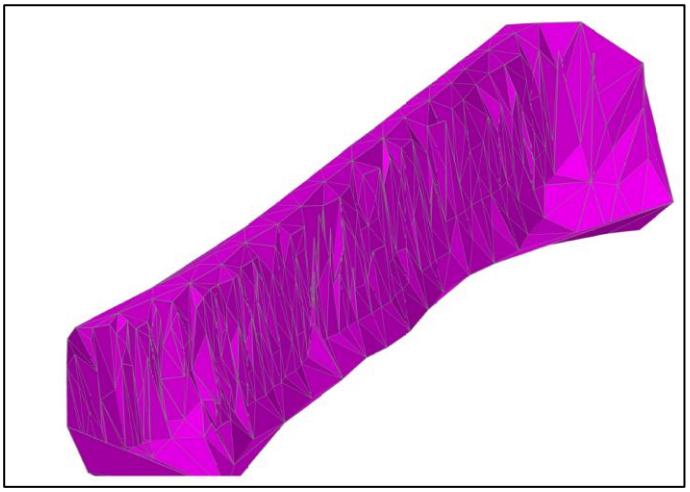
Although in Hardovac the drilling and blasting operations are currently outsourced, the Carmeuse's team keeps the control of the process and each blast is prepared and evaluated according to the company procedure.

During the initial step, based on the five years and the one year mine plans, the quarry manager decides where the blast will take place; he creates a folder with a unique identification number and prepares the drilling plan.

In the second step, with his Topcon total station, he is marking the blast field, surveying the drill hole position (X,Y,Z) and measuring the top edge of bench (crest), the bottom edge of bench (toe) and the wall (face of bench). These data are then extracted from the total station and imported in Carlson software where they are processed in order to generate surfaces (TIN Files).

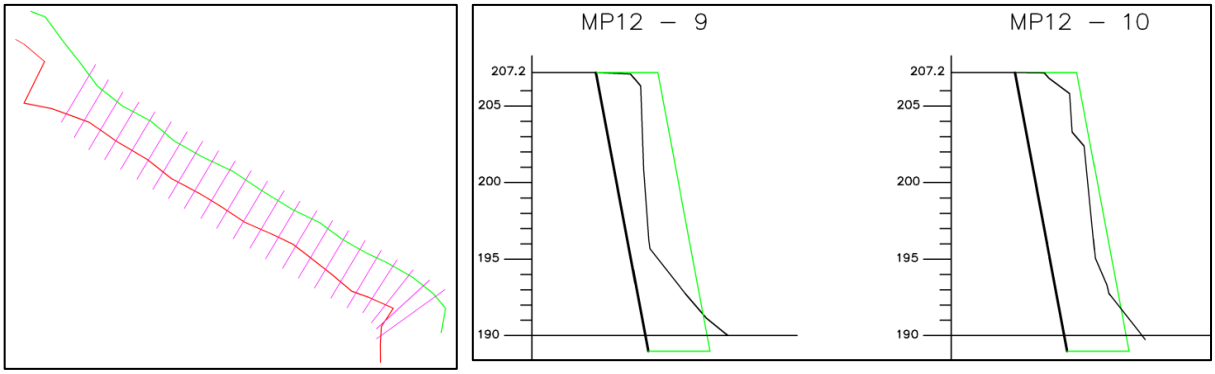


Picture 1: Topcon total station



Picture 2: Example of surface processed in Carlson software

These surfaces are used to determine the exact drilling depth of each hole and to generate the critical profiles displaying the shortest burden along them (with the theoretical assumption that each hole is drilled according to the plan and without deviation). Picture 3 gives an example of the positions of the profiles and the vertical profiles of two drill holes.



Picture 3: Example of positions of the profiles (left) and vertical profiles of drill holes (right)

From the example it is visible that the burden thickness may considerably vary from the theoretical one (green line on the vertical profiles) and ignoring this could for instance lead to a fly rock incident.

Based on this analysis, if required the position of the holes and/or the drilling angle can be adjusted. A drilling passport is issued and the drilling works can start. The rig operator fills a report where he records the depth and length of any type of weakness (fault, cavern, clay pocket, etc.) crossed while drilling.

Profiles of the drill holes are used together with the drilling diary to plan the explosive charges (type, quantity and location). It is for instance considered if split charges with inter-stemming are required. Profiles of each drill holes from the first row are printed and used in the field. By this way the blasters have a clear image about the width of the burden in front of the hole which is being filled and don't need to take the risk to check it from the top of the bench.

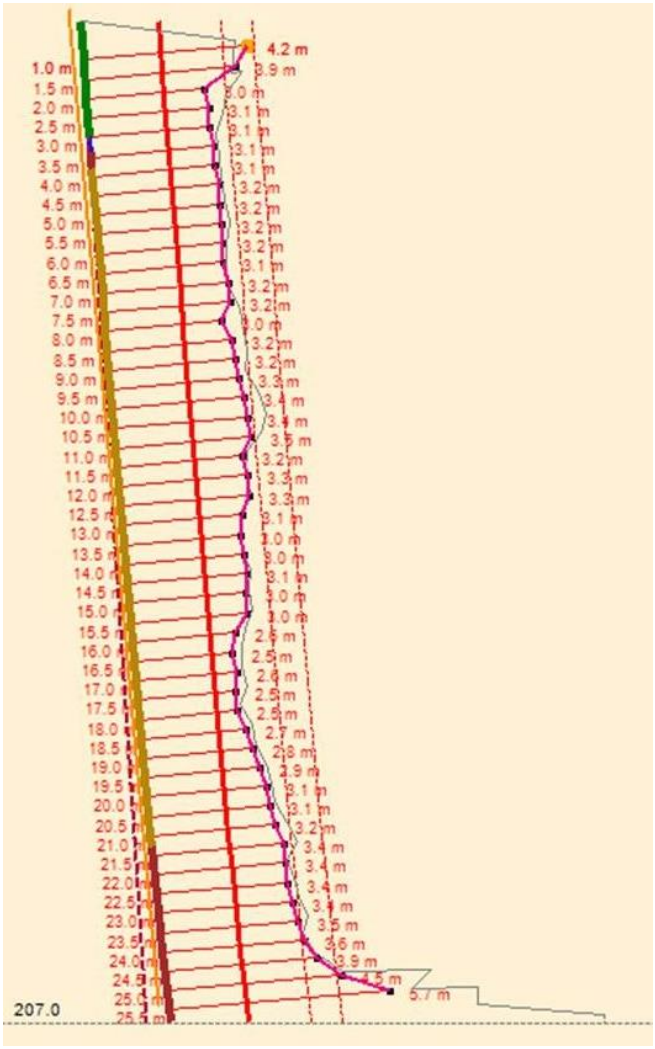
The seismic effects are being monitored for each blast. The monitoring is performed outside the quarry with two different seismographs located at the two closest objects from the blast field. Also, each blasting is recorded by video camera.

After the blasting the results are analyzed and when needed corrective actions are taken.

3. FURTHER IMPROVEMENT OF DRILLING AND BLASTING PREPARATION

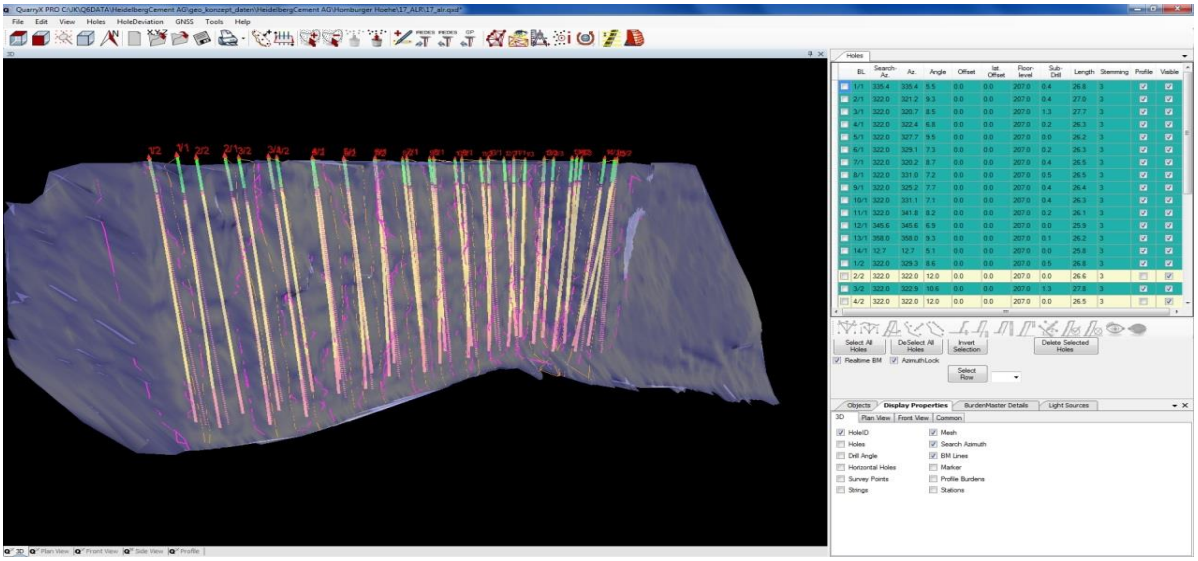
In order to keep moving with the improvements, we investigate the available tools which could make our practices more efficient. There are several brands on the market which offer full packages (hardware and software). Hardware mostly consists of GNSS (Global Navigation Satellite System), 3D scanner and hole deviation probe. Specific software are built to integrate the raw data from the field and to deliver the following outputs : actual versus planned hole positions, drill holes profiles with actual burden, quantity and position of the explosive charges, initiation sequence, €/t, etc.

For us, the usage of a deviation probe is a must if we want to get rid from the theoretical assumption that each hole is drilled according to the plan and without deviation. The figure 4 displays an example of hole deviation and actual burden.



Picture 4: Example of hole trajectory and actual burden thickness (QuarryX SW by Geokonzept GmbH)

The usage of 3D scanner would be faster and would provide us a more detailed survey of the wall. The addition of a GNSS (Global Navigation Satellite System), would ease the geo-referencing of the raw data collected in the field and the update of the quarry map. The picture 5 presents an example of combination of 3D scanning and probe measurements in QuarryX software.



Picture 5: Example of output of 3D scan and probe measurements (QuarryX SW by Geokonzept GmbH)

The usage of drones for blast preparation and monitoring (shape of the pile, fragmentation, etc.) offers very promising perspectives already under investinvestigation in other Carmeuse sites (example in the picture 6).



Picture 6: Example of 3D scan and blast preparation using a drone (Carmeuse Belgium).

4. CONCLUSION

Since the use of explosives will remain for a longtime the most efficient way to fragment rocks, and taking into account the challenges that mining companies are facing, it is necessary to make efforts to optimize these operations in order to increase their safety, reduce their environment impact and make them more efficient.

Carmeuse, as a global company, operating quarries in many countries, under different mining legislations, is determined to implement the best available practices in the field of drilling and blasting with the aim to contribute to a better world.

We hope that our initiative will inspire the Balkan mining community and help to raise the regional mining operations on higher level.

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DOI: 10.7251/BMC170701107A

MANAGEMENT OF TAILINGS DISPOSAL FROM THE PROCESS OF PREPARATION AND ENRICHMENT OF IRON ORE

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ABSTRACT

Tailings storage facilities (TSF) are facilities for disposal and storage of waste that is produced as a by-product of ore processing or other base materials in mines. TSF is an unavoidable part in mines and definition of the area for the safe disposal of tailings is a complex process that requires interaction and good cooperation between different professions. Disposal of tailings in the Omarska mine was supposed to take place in two phases. The first phase of Gradina lake TSF is provided by the construction of the Međeđa dam. Since the first phase of the Međeđa dam is finalized and space was filled with tailings, raising the height of dam, to extend the dam's operating life in the second phase, required significant investment and increased environmental risk. Therefore, a technical-economic study was carried out, and it was analyzed in details the future management of tailings disposal in the Omarska mine.

This paper summarizes the results of a multi-year study and possibility for disposal of the remaining quantity of tailings, produced by ore processing in the Omarska mine. The analyzes encompassed total investment, positive and negative aspects of each variant, differences in the price of transportation, equipment maintenance prices and TSF monitoring, then the security aspects and criteria for choosing a variant. Also, project implementation, monitoring and achieved benefits of selected and implemented variant of the TSF were presented.

Key words: Tailings disposal, TSF, dam, iron ore

1. INTRODUCTION

Main mining project of the Omarska Mine defined disposal of tailings from the technological process of washing and enrichment of ore. Tailings storage capacity was provided with construction of the Međeđa Dam and the construction of tailings storage facility was planned to perform, as per the technical project, in two stages. The first stage of the construction was planned to provide storage capacity for disposed tailings from the Jezero pit, Mamuze pit and one part from the Buvač pit, and in the second stage of the construction would provide storage capacity for the remaining amount of tailings from the Buvač pit.

Since the construction of the second phase of the Međeđa Dam represents a significant investment, provision of storage space for about 4.300.000 m^3 of tailings was necessary to be analyzed in variants. Selection of the location for tailings disposal from the gravity-magnetic

separation plant was analyzed in the Techno-Economicalstudy of mud disposal site selection-Omarska Mine (Faculty of Mining and Geology Belgrade, September 2013) and were analyzed two variants of transport and storage of tailings:

- Variant 1: to continue to use existing Gradinalake TSF with the Međeđadam upbuilding for additional 6 m, and,
- Variant 2: ending of exploitation of existing disposal site until additional space is filled and transit to new disposal site which will be formed in surface area of the former open pit Jezero.

2. PROJECT IMPLEMENTATION

ArcelorMittal Prijedor (AMP) formed a team for the implementation of the project that containedcivil construction, mechanical, electrical, mining and economical part with the constant participation of the H&S department. Their task was to review project documentation, hold regular meetings, prepare tender documents, participate in commissions and negotiate with bidders, and to monitor all activities.

Realization of the project for building of the new JezeroIn-pit TSF included the following activities:

- construction of embankment
- Installation of the pipeline for tailings
- Installation of pump aggregate and return water pipeline
- tailings deposition channel
- emergency Spillway
- gate valve T6, gate valve T30 and the base plate for pump aggregate
- power supply of the TSF and installation of the motor and frequency regulator in the tailings pump station

The project was successfully implemented during 2015 and 2016, and the new Jezero in-pit TSFhas been operational since the beginning of 2017.

3. THE COURSE OF PROJECT IMPLEMENTATION

Project implementation consisted of several phases. Below you will find an orientational chronological overview of the activities that led to the final selection of tailings disposal location.

The first phase of the tailings disposal management project is to collect the existing documentation (plant information, characteristics of tailings to be stored, defining the required storage space).

Parallel to the first phase, an analysis of the existing storage method (characteristics of disposal, filling of the dam, expansion possibilities) was performed.

Then, checking the option for disposal of tailings to the former pit Jezero, where exploitation was completed, was also carried out (storage space, hydrogeological study).

The next phase is a comparative analysis and final selection of tailings disposal location, documentation and final construction of the facility.

The final phase is the use of the TSF, monitoring and analysis of achieved results.

4. AN ILLUSTRATION OF THE TECHNOLOGICAL PROCESS OF GRAVITY-MAGNETIC SEPARATION

The gravity-magnetic separation plant consists of two identical technological lines, which can be operated separately (individually) or simultaneously, in parallel operation. The total projected capacity on both technological lines is 666 t/h of feed ore.

The amount of water consumed in the technological process is about 2500 m^3/h. Designed, current and required water balance is shown in the table below:

Table 1. Water balance

	Capacity, m³/h		
	Designed	Current	Required[1]
Return water	1230	800	2000
D – pump–water inlet onGomjenica	1230	1100	
CN – pump -water inlet onGomjenica	405	350	400
Total	**2665**	**2250**	**2400**

Tailings from the GMS plant has an average volume of 3.300 kg/m^3 and a volume of 1.320 kg/m^3, a share of 910 kg/m^3 and a hydraulic compression of 1.110 kg/m^3. By weight it is a finely crushed material in which 80-100% of the grain is smaller than 0.100 mm and more than 50% less than 0.037 mm. Per literature data, the inner friction angle is 28-35^o and the cohesion ranges from 10 to 28 kPa. Tailings is very quickly precipitated to form a recess of 15-25% (depending on the iron content and initial solid phase concentrations). Laboratory options show that it is relatively quickly cleansed (after 25 minutes).

5. ANALYSIS OF THE EXISTING STORAGE METHOD

Tailings from the process of iron ore processing in the Omarska mine gravitates to the pumping station for the tailings transport, from where it is transported by slurry pumps to the Gradina lake TSF. Tailings is hydraulically conveyed in the form of a very rare hydraulic solution (S:L ≈ 1:12) through a steel pipeline of ND 700 mm diameter using slurry pumps equipped with electric motors of 1.250 kW. Tailings precipitation occurs at the TSF, and the clean water is gravitationally returned to the pump station, located close to the GMS, from where it is transported to the process plant. Tailings transport capacity is about 4000 m^3/h and the return water is about 2.400 - 2.500 m^3/h.

To continue disposal of tailings in the existing Gradina lake TSF, it is necessary to up build existing Međeđa dam and to secure the required storage space while retaining the existing tailings transport system from GMS to the TSF and return water from the TSF to the existing pump station. In addition, it is necessary to upbuild overflow gallery for receiving and directing the return water and land buyout for dam up building downstream method.

[1]Required capacity for the production of 2Mt of concentrate per year

6. CHECK THE POSSIBILITY OF DISPOSAL AT THE LOCATION OF THE FORMER PIT JEZERO

At the time of the third phase of the project, former pit Jezrero, on which the exploitation was completed and was in function of dumping site for overburden material from the active Buvač pit. Therefore, it was necessary to define the space required for the overburden disposal and space required for the tailings disposal.

The available space within the former pit Jezero amounts to 22.940.230 m^3. In this area, a part of the Buvac pit overburden and part of the tailings from the process plant should be deposited.

Accordingly, the following distribution of available space was made:

- from the bottom of the depression to the elevation of 151 m provides a space of total volume of 19.930.967 m^3 and there will be deposited overburden from the Buvac pit
- from 151 m to 158 m, the total area of 5.467.905 m^3 will be provided for the tailings, in such a way as to:
    - from 151 m to 155 m (3.009.263 m^3) to undertake disposal into the deepest area of depression,
    - from 155 m to 157 m (1.621.055 m^3) will be deposited within the accumulated space that will be formed by the construction of the embankment, and
    - a space of 157 m up to 158 m (837.587 m^3) will be provided for freeboard.

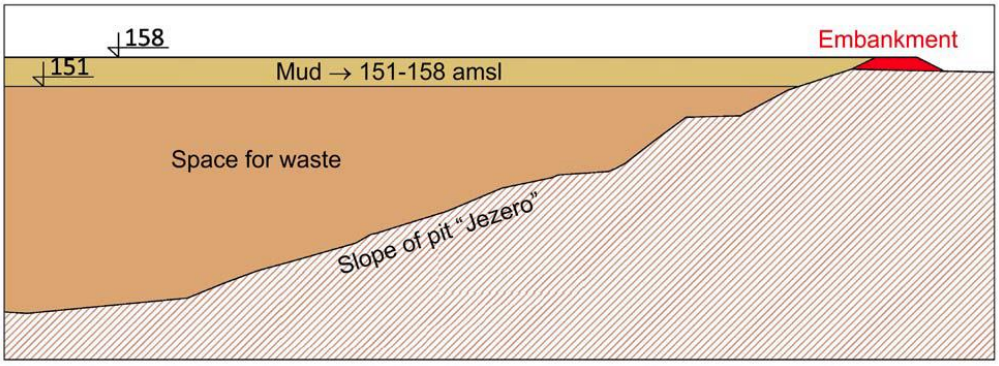


Figure 1.Characteristic cross-section throught the Jezero In-pit TSF

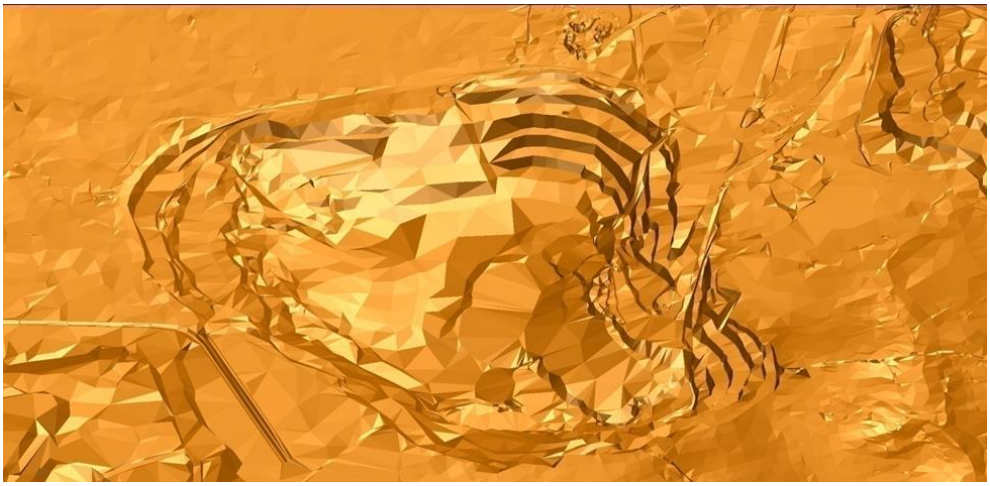


Figure 2. Total available storage space

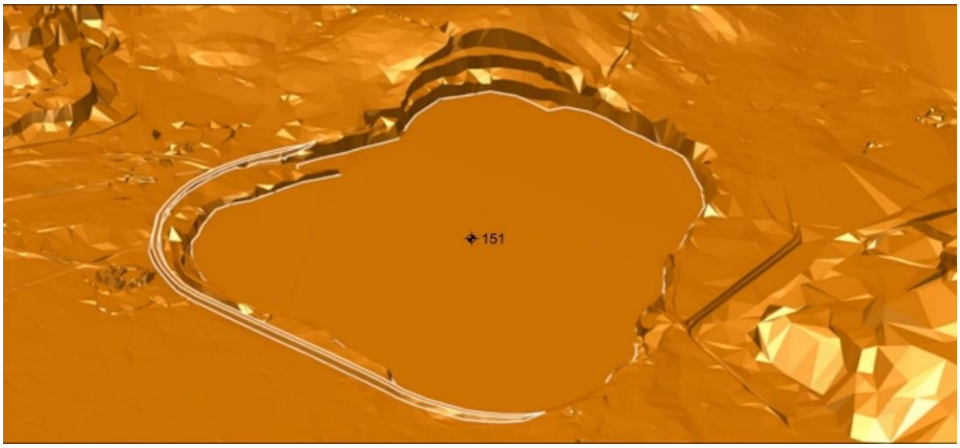


Figure 3. Available storage space for tailings disposal
(after disposal of overburden from the Buvac pit)

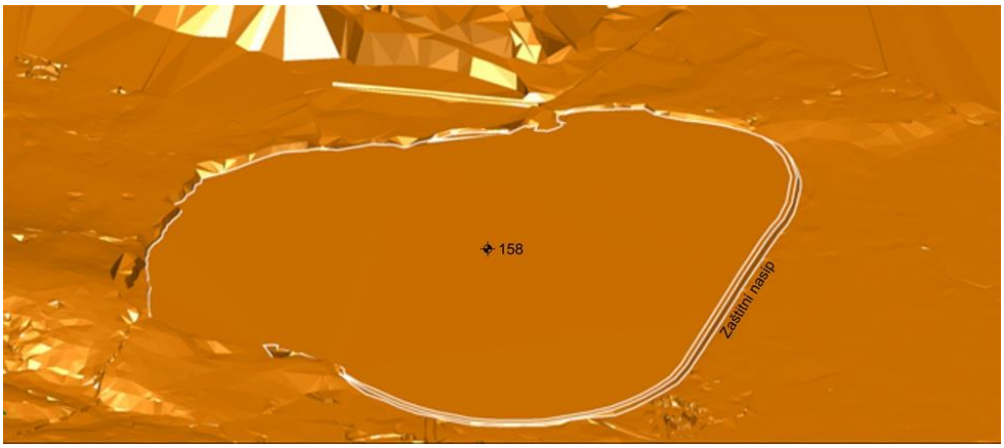


Figure4. Jezero In-pit TSF filled with tailings

Probable maximum flood (PMF) is a large water that can be expected as a result of the most unfavorable meteorological and hydrological conditions in the catchment area. The PMF is obtained on the basis of probable maximum precipitation (PMP) using the appropriate precipitation-runoff model, assuming the most unfavorable combination of other critical factors affecting runoff.

Table 2 - Summary results of the PMF assessment at the Jezeroin-pit TSF

	Total precipitation (mm)	Total runoff (mm)	Runoff volume (m^3)	Peak flow (m^3/s)
PMF	400	389,7	144.968	17.7

Considering that precipitation is also falling over the tailing pond area of 84.7 ha, additional inflow volume to the pond is equal to:

$$V_{\text{pond}} = 84.7\,\text{ha} \cdot 400\,\text{mm} = 338{,}664\,\text{m}^3$$

Maximum total volume entering the tailings pond is then:

$$V_{\max} = 144{,}968 + 338{,}664 = 483{,}632\,\text{m}^3$$

The corresponding mass volume curves are shown in Figure 5.

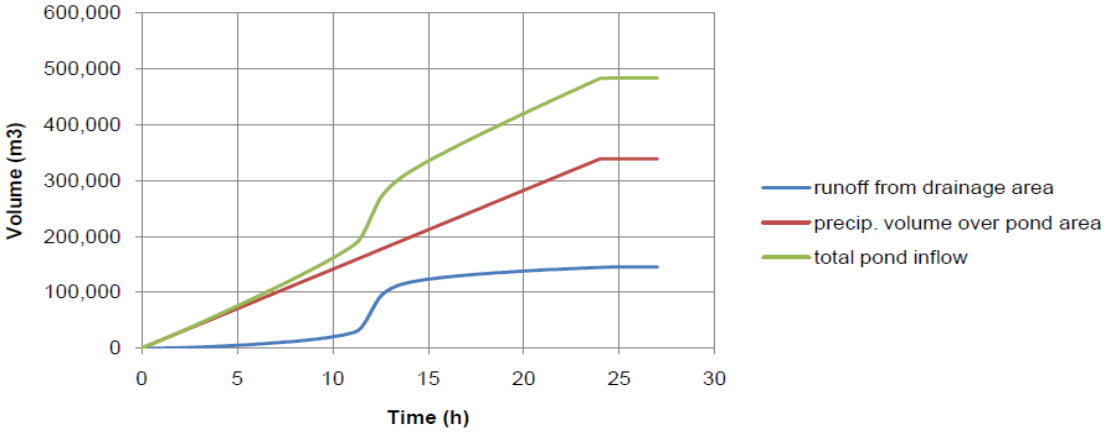


Figure 5. The mass volume curves of inflow into the tailings pond for PMP

From the above analyzes the following conclusions are:

- new tailing storage is not endangered from probably maximum precipitation (PMP) that could occur on the site area,
- storage capacity of disposal site, from elevation 151 to 157 m a.s.l., is enough to storage all expected tailings from gravity-magnetic separation plant.

7. COMPARATIVE ANALYSIS AND SELECTION OF OPTIMAL VARIANT

Below is a description of a comparative analysis of two variants.

Table 3. Comparative analysis of two variants

Variants of tailings disposal	Gradina lake TSF	Jezero in-pit TSF
GOOD FEATURES	already existing, well known working system on transport and tailings disposal would be kept	shorter tailings transport which is technically and energetically favorable and new overburden waste dump is formed in the East part which significantly shortens truck transport
	return water transport is done by gravity	easier work organization because disposal site is inside zone that is already used by Mine
BAD FEATURES	dislocated TSF in relation to GMS which complicates work organization and long distance transportation which resultsincreased energy consumption	transport and disposal system is changing which needs new adapting of staff
	the influence of the high dam on safety aspect downstream from the dam and expensive dam maintenance during and after usage	return water transport could not be organized by gravity, so the pump aggregate has to be procured in order to transfer return water from disposal site into space near existing pump station for GMS water supplying
INVESTMENT	7.178.000 EUR	3.019.000 BAM
ANNUAL OPERATIONAL COST	1,36 BAM/t	0,81 BAM/t

The results of the techno-economic analysis showed that variant 2 was more favorable and after its adoption, as part of the Supplementary Mining Project, it resulted in the creation of the Technical Project for tailings disposal from the gravity-magnetic separation into former pit Jezero (Faculty of Mining and Geology Belgrade, September 2014).

8. MONITORING AND ANALYSIS OF THE ACHIEVED RESULTS

Construction of TSF is not a definitive process and requires continuous activity through its usage, day-to-day monitoring as well as readiness to respond in emergency situations.

There is a wide range of elements that can be monitored in the TSF, which can provide indications of the work of the facility. This process involves gathering, capturing, analyzing and presenting data on a regular basis, and in many cases instruments are also included to significantly increase the monitoring methods. The use of tools enables gathering and analyzing real-time monitoring information and historically through electronic data collection and analysis. However, technology does not replace monitoring on the field.

Recommended items for monitoring the TSF in accordance with the best world practice are listed below:

- Monitoring of the TSF
- Monitoring of the environment
- Other monitors

Monitoring of the TSF includes:

- tailings properties
- profiles of phreatic surface
- pore pressure, including increased and dynamic pressure
- total pressure
- freeboard and filtered water
- tailings disposal and beach development
- tailings disposal flow
- return water flow
- vertical and horizontal movements
- cracking movements
- internal stresses.

Monitoring of the environment, ie the environment to be taken into account during the construction and operation of the TSF is:

- seepage
- groundwater
- decant water.

Other monitors during the construction, work and closing periods are:

- seismicity
- climatic data
- additional construction monitoring.

The advantage of the Jezeroin-pit TSF, in addition to the economic benefits that are realized through savings on the cost of tailings transport from the process plant, in comparison to the Gradina lake TSF is also a change of classification of the facility.

Namely, the Gradina lake TSF is classified as high dam by the International Committee on Large Dams (ICOLD). Per the same categorization, the risk posed by the construction of second stage of the Međeđa dam is marked as "high" and as such requires special supervision and frequent examinations which, in addition to routine, include review of independent auditing houses.

By contrast, the Jezeroin-pit TSF is largely formed in the depression below the ground elevation, resulting from the exploitation of ore on the mentioned location. To allow facility operation (disposal of tailings, the return of the clean water and the overflow of excess water through the spillway into the water stream), embankment of average height of 3 m was raised.

9. ANALYSIS OF THE ACHIEVED RESULTS

Operational costs of transporting one ton of tailings to Gradina lake TSF are 0,467 EUR/ton.
Operating costs include:

- energy costs spent to operate the pump for tailings transport;
- energy costs spent on the operation of the pumps for the transport of river water to process plant;
- maintenance costs of the equipment for tailings and water transport;
- labor costs;
- monitoring and intervention costs.

10. TAILINGS DISPOSAL IN THEJEZERO IN-PIT TSF

For the disposal of tailings in the Jezero in-pit TSF and the transport of cleared water to the GMS plant, the following equipment is used:

- 2M pump - nominal power 500kW with frequency regulation;
- two D320 pumps (320kW) for return water transport;
- Warman pump for return water transport (160kW).

Operating costs include:

- energy costs spent to operate the pump for tailings transport;
- energy costs spent for the operation of the pump for return water transport to the GMS;
- maintenance costs of the equipment for transport of tailings and water;
- labor costs;
- monitoring and intervention costs.

11. OPERATING COSTS IN Q1/2017 AND Q2/2017 (H1/2017)

During Q1/2017, 100.629 t of tailings was transported at a cost of EUR 0,032 million, and the actual cost per ton of transported tailings was 0,317 EUR/ton.

During Q2 /2017, 85.229 t of tailings was transported at a cost of EUR 0,026 million, and the actual cost per ton of transported tailings was 0,310 EUR/ton.

During the H1/2017, 185.858 t of tailings was transported at a cost of EUR 0,058 million, and the actual cost per ton of transported tailings was 0,314 EUR/tons.

Actual profit in that period was EUR 0,029 million or 0.15 EUR/ton of transported tailings.

12. CONCLUSION

- Management of tailings disposal is a complex process that lasts from the idea to the TSF closure, and the monitoring of the TSF in a reduced extent continues after the finishing of the tailings disposal.
- TSFs are permanent facilities that must meet the needs of ore processing plants and must also be safe and harmonious with the environment.

  Parameters that affect the dynamics of disposal are subject to change, either due to plant or external influences, so quality monitoring is crucial to achieving maximum utilization of the deposit.
- Dams are constructions that are connected with the environment and subject to external influences such as seismic influences, extreme meteorological events, high waters.
- From this, it is concluded that besides the economic justification of selecting variant 2, there is also significant reduction of the risks that would result from the construction of a large dam.
- Finally, if possible, even with the same costs of tailings transport, selecting a disposal variant below the ground level is far better from the standpoint of security than the construction of a large dam.
- In the future, there may be a need for usage of the secondary potential deposits, ie the utilisation of the deposited tailings in the Gradina lake TSF. Such a form of analysis and tests deserves special attention and it will provide guidelines for the reprocessing of ultra fine classes of iron ore from the tailings.

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DOI: 10.7251/BMC170701117L

TRANSPORT OF DEPOSITOR ARS1600(37+33+60)X18(O-V), ON THE P.K FIELD "C"

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ABSTRACT

Open cast mine B and C are the oldest active mines in Kolubara coal basin. They are situated in Western part of basin between Field A (artificial border) and rivers Peštan and Turija (natural borders). Overburden is excavated with four systems, and coal with two levels connected with collecting conveyers and merges with coal excavated from coal system at Field D. All coal is transported toward Prerada (segregation).

Because of the dynamic of development of the coal basin there is often a need for transporting mining machines from one part of the basin to another.

This work gives an insight into transporting one of these machines, depositor, with a detailed showing of the depositor root and all of the following activities that go with the process.

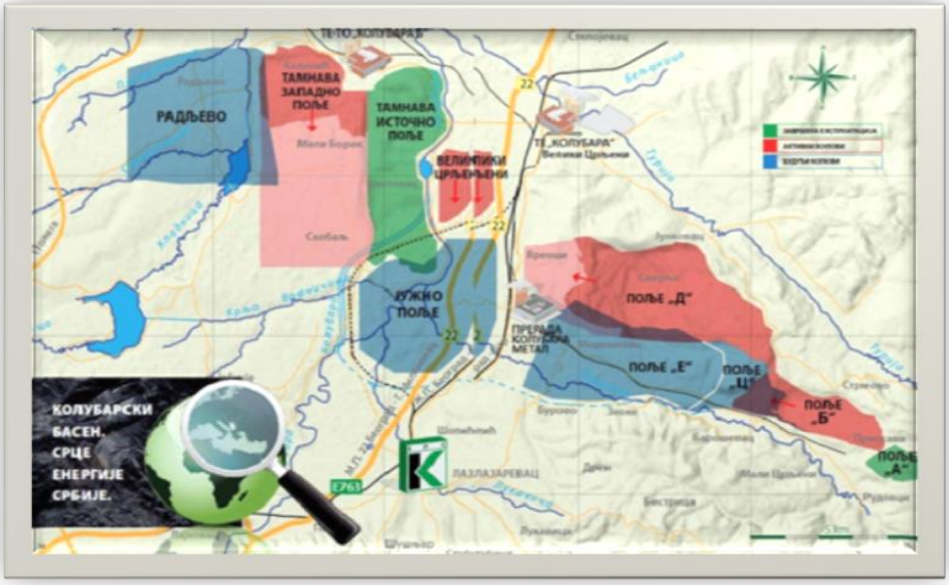
Key words: Surface mine "B/C", stripping, depositor transport

1. INTRODUCTION

The Kolubara coal basin is located in central Serbia and spreads over western Serbia in between settelments Rudovci, on the east, Koceljevo on the west, Lajkovac on the south and Stepojevac on the north. The river Kolubara with its midstream divides the basin into two parts, western and eastern part. In the eastern part of the Kolubara basin are surface mines field "B/C" and "D", the newly opened field "G", the closed surface mine field "A" and future surface mines: fields "E", "F" and "Šopić". In the western part of the basin are located "Tamnava-Istok" field and surface mine "Veliki Crljeni", which are closed, and the new surface mine field "G" and also "Tamnava-Zapad" field that are active, and also a perspective future mine "Radljevo". Field "B/C" has an area of 4,3 km^2 and contains the easterly part of the basin between field "A" and field "C" (artificial borders) and the rivers Peštan and Turija (natural borders). This mine is the oldest active mine in the Kolubara basin.

On the south edge of field "B/C" goes a tarmac road Lazarevac-Aranđelovac, and on the east, trought the inner depository, the road "Baroševac-Strmovo" wich was put in use in september of 2011. The surface mine field "B/C" with its industrial railroad is connected with its coal processing facilities in Vreoci, and with its belt conveyor with the mine field "D". The excavation of the mine field "B" started in 1952, and the firstonnes of coal in 1956. Since the

begining of works on the exploitation of overburden in 1952, and the first tonnes of coal in 1956, in this mine, to **1.08.2017**., the mine has produced **239.075.672 m^3 čm** of tailings and **95.836.671t Ru** of coal and shipped it to its users (wet separation in Vreoci where it is refined and dried, TE "Kolubara u Velikim Crljenima" and wide spread). The production of tailings and coal on surface mine field "B/C" is organized on three BTO systems, and a coal system (first and second floor line), and one midlayer tailings system. Because of the formation of the new BTO system in field "B/C" there is a need for relocating some machinery and reconstructing on the tailings systems so the depositor Ars1600(37+33+60)x18(O-V) has to be relocated from I BTO system to III BTO system.



Picture 1: Overview map of the Kolubara basin with the position of the excavation fields

2. TRANSPORT OF THE DEPOSITOR

Transport of every mining machine in surface mine field "B/C", and also the transport of the depositor is done during daylight with strict guidelines respecting work instructions and taking in consideration the depositors technical characteristics. Besides this, from the aspect of safety and health, it's necessary to have in mind the safety of people that are involved in the transport. Because of this there are many regulations that must be met before, during and after transporting mining machinery.

Before transporting the depositor we must specify the rout of the depositor in detail considering its technical characteristics, and the characteristics of the terrain. The work on the terrain itself should be keeped at a minimum if possible because of the nature of the terrain and objects that can be in the way of the rout, because it increases the price of transport. The depositor rout must be defined precisely and adapted to the technical characteristics of the depositor consider in its longitudinal and transverse tilt as well as specific pressure on the substrate.

The width of the rout must be in accordance with the dimensions of the depositor. In the study on the transport of the depositor, the engineer must complete a complete scheme of the route with all supporting works such as earthwork, the place of attachment of the depositor on the electrical network, etc., so that the responsible person, who manages the transport of the machine, has a predetermined route of the depositor, where there will be additional works. Each stage of the transport must be adequately documented and verified by the people in charge, so that the person who is managing the transport does not have a doubt to what he is doing.

The transport rout of the depositor ARs1600 (37 + 33 + 60) x18 (OV)) from the landfill conveyor C.5 to the landing conveyor III BTO system, 2.10 with the levels of the drive and return station 100-101 and the coordinates of the drive and return station 74 50585,49 18365.9;74 50534,49 17919.7. Transport of the depositor starts from the height side of the landfill conveyor C.5 from the coordinates 74 50002.6,49 19701.4.

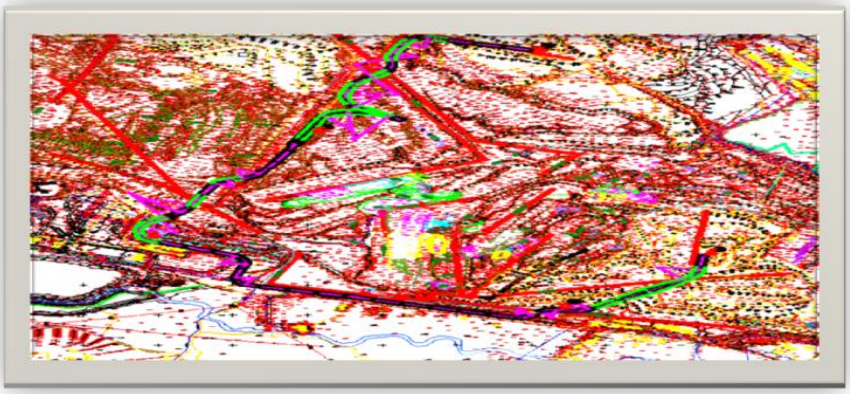
The transport of the depositor will take place during daylight, along the route, which should be prepared with the necessary corrections, with a dredge excavator. **The total length of the route is 5200 m, with tilts not exceeding 5%.** The start of the ARs1600 (37 + 33 + 60) x18 (O-V) depositor transport route is at a level of 228 mnv. The first 300 m of the route has no corrections, and after that the corrections are made by a dredger.

After 2800 m of the route, the new designed route fits into the previously designed route for the transport of the new ARs8800 depositor, so that the levels are monitored on the ground, and there is no need to correct the route. The last 700 m of the route is corrected by the construction of sewerage and embankments, so that in the last 100 m of the route, the route completely fits into the level of the terrain. The track of the ARs1600(37 + 33 + 60) x18 (O-V) depositor ends at 97 mnv, with coordinates 74 50577.1, 49 17999.6.

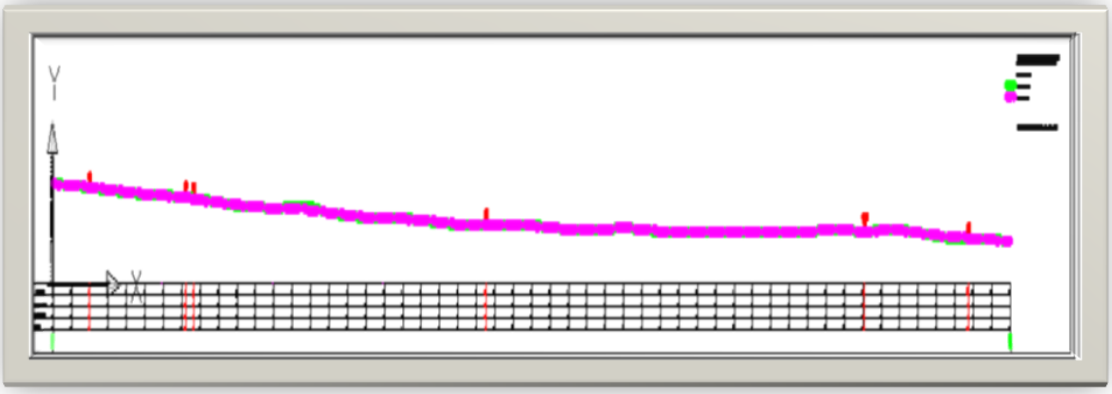


Picture 2: ARs1600 depositor (37 + 33 + 60) x18 (O-V)

Picture 3 shows the layout and the diameter of the curve of the line. In picture 4 is the longitudinal profile of the route, with the track and ground level, as well as the tilts of the route. The maximum allowed tilt of the route for this trap is 5%.



Picture 3: Arrangement and line curve length



Picture 4:Longitudinal profile of the route

3. CONCLUSION

On Kolubara surface mines, the transport of rotary excavators and depositors as well as other mining machines are frequent occurrences.

The transport of the depositor to field "B/C" from one BTO system to another, was conditioned by the reconstruction of the landfill and floor conveyor belts, due to the installation of the new BTO system, and for this reason the need for the development of a comprehensive study on its transport, which had documented every stage of the transport of the depositor, leaving no aspect of transport uncovered by adequate documentation, in which through the diagram, each stage of the transport of 5200 m was explained, thus avoiding any confusion.

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DOI: 10.7251/BMC170701121U

TECHNO-ECONIMIC ANALYSIS OF DRILLING AND MINING PARAMETERS IN THE PROCESS OF EXPLOITATION ON THE DIABASE QUARRY "TRNOVA" NEAR GRADIŠKA

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ABSTRACT

Although the significance of the technical aspects of the mining industry can not be ignored, narrow focus only on technical issues is no longer enough to guarantee economic success, even in the richest ore bodies. On this way results of the economic analysis of particular technological stages of exploitation are an important parameter of productivity and efficiency of the whole exploitation process. This paper presents a model of the economic evaluation of different variants of drilling and blasting, and analyzing the results obtained with the consideration of all the cost of drilling and blasting operation defined in the process of specific grain size production. This is followed by the selection of technically and economically optimal variant of this technological operations in the mineral raw materials exploitation.

Key words: economic evaluation mode, drilling and blasting activities, optimization

1. INTRODUCTION

In modern mining, in most cases, it is almost impossible to correctly and racionaly organise extraction of solid mineral resources, without use of explosive. Mining and drilling are one of the most influential factors for economic exploatation of mineral resources, since application of explosive in process of getting mineral resources allows using of other modern mining techniques.

Technological fazes of drilling and mining in process of exploatation are very important in particular case and topic of this paper, since success and capacity of further technological fazes considerably depend of fragmentation of mined material (capacity of bulldozers, loading units, trucks, units for crushing and separation).

Deposit "Trnova" lays in northern part of Republic Srpska and belongs to Gradiška municipality as administrative-cultural centre. On wider region can be distinguished mountain, hilly and plainy types of landscape. There are two open pit mines in deposit Trnova, quarry "Trnova" (northern part) and "Trnova-jug" (Trnova-south), which is topic of this paper.

In region of Republic Spska deposit "Trnova" represents one of the biggest deposit of diabase in exploatation and it exploits magmatic rock diabase, whose quality characteristics satisfy conditions for producing of agregate for making asphalt, for roads with dense and heavy trafic.

Analysis of physical, technical and mechanical characteristics of rock which makes work enviroment on OP "Trnova" near Gradiška, and with help of further analysis of possible economic savings during drilling and mining, expenses in different possible modes of work are analised and calculated.

2. DEFINING OF PROBLEM AND GOAL OF PAPER

Although significance of techinical aspect, can't be ignored, narrow focusing on only technical topics is not longer enough to guarantee economic succes, even in the richest ore bodies.

Economic skills and knowledge are important partner of technical skills and knowledges, on every step of mining process. Economics of mining and industries in general differ in many ways from classic economic reviews in other branches, because mining contains many uncertainties unlike some other industrial branches.

Therefore, primary task of this paper is marking economic analysis of one technological phase in explotation as important parameter for productivity and economy of whole process of exploatation. Economic analysis, therefore, can be performed for different phases of process of mine exploatation.

Evaluation of costs and economic analysis are today integral part of process of designing and planing.

Main objective of this paper is implementation of economic evaluation of different types of drilling and mining, and analyzing of obtained results, which is followed by selection of technically and economically optimal variant of technological operation in process of exploatation of mineral resource.

Table 1. Diabase reserves

CATEGORY	RESERVES, (m^3)
A	134 642
B	1 749 490
C_1	698 699
$A+B+C_1$	**2 582 831**

Economic evaluation must be based on model which will consider all costs of drilling and mining which apper in process of getting mineral resource of specific fragmentation. Diagrams of dependence of drilling costs of drilling diameters and rock type are shown in figures 1 and 2.

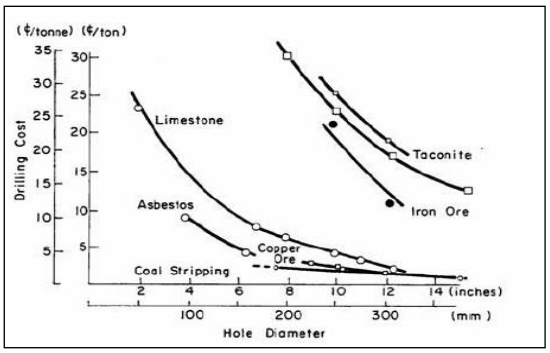


Figure 1- Diagram of drilling costs depending on drilling diameter and rock type

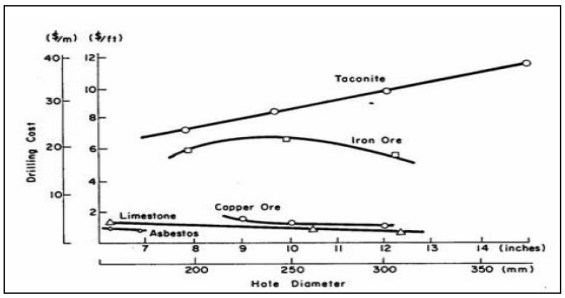


Figure2- Diagram of costs depending of drilling diameter (bigger diameter) and rock type

Deposit of diabase "Trnova" near Gradiška is located on northern slopes of mountain Kozara. Analysis of physical, technical and mechanical characteristics of diabase showed folowing parameters of work environment : compressive strength in dry condition (135,78 MPa), compressive strength in saturated condition (125,8 MPa), compressive strength after freezing (129,6 MPa), porosity (3,12 %), water absortion (0,29 %), volume mass (2,785 gr/cm^3), impact resistance - method "Treton", (10,53%), wear and tear by scraping resistance (Vöhme-u) (10,39 cm^3/ 50 cm^2).

On OP "Trnova" cyclic discontinuous system of exploatation will be applied. Big influences on chosing of exploatation system have : structure of mechanization, shape location and volume of landfill etc. Excavation is being done in bench 10 meters high, with possibility of increasing up to 15 meters.

3. TECHNO-ECONOMIC ANALYSIS OF DRILLING AND BLASTING PARAMETERS

Mining shuld be primarily, focused on technical aspects of getting ore and its eficient processing. But except skill in solving technical questions, can't be ignored that narrow view on only technical solutions does not doarantee succesfull business, even in richest ore deposits.

Economic and technical skills are spreaded in every phase of exploatation and it is important to think on economic way all the way since initial drilling and mining. This does not mean the way mining works are done, must be economicaly optimized, but it also means adoption of the most economic way of fulfillment of mining project. Besides, it is necessary to economically rationally evaluate mining projects.

Economy in minin is unique, because all economic aspects of mining exploatation can not be applied in other branches. In process of producing, means of work are constantly beieng physically weared or technologically outdated due to advent of new ones, more modern and productive. This wear reduces its working ability, and its market value. Market value of that reproductive wearing is called amortisation, whose value depends on level of technical equipment and intesity of wear and lifetime. Besides cost of amortisation, in means of work costs, also enter costs of current and investment maintenance.

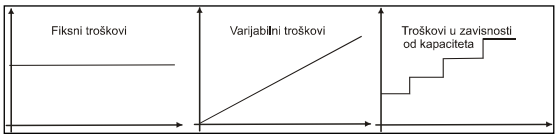


Figure 3- Three basic types of mining costs behavior

Percentual participation of individual costs in mines, in total sum, according to some researchers is :

- costs of drilling 5,5%,
- costs of mining 9,5%,
- costs of loading 10%,
- costs of transport 40%,
- costs of auciliary operations 17%,
- costs of menagment and control 5%,
- other costs 13%.

It is necessary to have valid informations about costs for bringing a series of business decisions. It is very important to know what is gained and what is lost by choosing specific solution when making business decisions.

During purchase of the new equipment, it is common to firstly calculate how high is investment. Costs of maintenance and operational costs are not usually precisely evaluated, although they are in most cases higher then initial investment, if machine's lifetime is considered. Reason for this is far more complicated metodology used for evaluation of this costs.

Base for this economic analysis is establishment of functional connection between technological factors and costs, which by their structure can be fixed or variable, and can be split in three groups:

- purchasing costs of mechanisation (investment or capital costs),
- ownership (possession) costs of mechanisation (depreciation),
- working costs of mechanisation (operational costs, energy costs, oil, lubricant, pneumatics, maintenance, working force etc.)

By term "work environment" we consider rock mass in which specific mining operation is done. Rock mass can be from different origin (eruptive, sediment or methamorph) and can have different physical, chemical and technical properties, which are very important for estimation of drilling and mining parameters.

Drilling mining holes is procedure of making cylindrical cavities in rock mass and mineral resource in which some amount of explosive is put for mining.

Mutual connection of mining hole diameter and maximum allowed size of piece is:

$d=k \circledcirc D$, mm

where: k - is coefficient which depends from degree of rock crushing (k=0,075 for average crushability of rock).

Before picking of mine hole diameter, it is necessary to determine maximum dimensions of pieces for bucket excavator, by next pattern:

$$D \leq 0{,}75 \sqrt[3]{E}\ , \text{m}$$

E=1,5 m^3: minimal volume of excavator's bucket which will be used on OP "Trnova-jug"

$$D \leq 0{,}75 \sqrt[3]{1{,}5} \Rightarrow D \leq 0{,}86 \text{ m}$$

$d=0{,}075 \circledcirc 0{,}86$

d=0,064 m = 64 mm

Recommended diameter for drilling is between 60 and 90 mm (76 mm diameter adopted in calculations). Using diameter bigger then 90 mm is not planed, due to possibility of making too big pieces.

Self-propelled drill "Atlas Copco ROC F-7" is planed to be used for drilling mining holes, and is shown on figure 1:



Figure 4 – Self-propelled drill "Atlas Copco ROC F-7"

Diameters often used for drilling in quarries, from 76 or 89 mm, are used on OP "Trnova" to avoid occurrence possibility of bigger pieces. For comparative techno-economic analysis, we will consider two drills "ROC F6" and "ROC F7" and their working parameters, and analyse costs and effects.

Review of total capital and operational costs of work for both drills, with all spending means normatives is given in table 2. It is necessary to emphasize that evaluation of costs was performed by current market costs, which are:

- fuel 1,85 KM/l,
- oil and lubricants 5,45 KM/kg,
- drilling crown 185 KM/piece,
- drilling poles 295 KM/piece.

Table 2. Review of capital and operational drilling costs

	ROC F6	ROC F7
Capital costs		
Purchase cost	526500 KM	624000 KM
Operational costs		
Depreciation rate	5%	5%
Fuel consumption	14 l/h	18 l/h
Drilling speed through limestone	18 m/h	24 m/h
Oil and lubricant consumption	0,0003 kg/m^3	0,0003 kg/m^3
Drilling worker's gross salary	1800 KM/month	1800 KM/month
Crown consumption	0,000067 piece/m^3	0,00006 piece/m^3
Poles consumption	0,000033 piece/m^3	0,000027 piece/m^3
Lifetime	50 000 ef. hours	50 000 ef. hours
Total costs		
KM/m^3	0,41	0,48
KM/m	3,66	4,25

The term "mining" is considered as using destructive effect of explosive, which is used in mining for destruction of rocks during mining operations below and on the surface.

Table 3. Drilling and mining parameters

	unit	Mining with mining holes with diameter of 76 mm, height bench 10,0 m		Mining with mining holes with diameter of 89 mm, height bench 10,0 m	
Main charge		"AN–FO"	powdery	"AN–FO"	powdery
Auxiliary charge		emulsion	"AN–FO"	emulsion	"AN–FO"
W	m	3,0	3,0	3,5	3,5
a	m	3,0	3,0	3,5	3,5
b	m	3,0	3,0	3,5	3,5
m		1,0	1,0	1,0	1,0
L_{hole}	m	11,54	11,54	11,7	11,7
$L_{underhole}$	m	0,9	0,9	1,1	1,1
$L_{main\ charge}$	m	4,24	3,81	6,77	6,04
$L_{auxiliary}$ charge	m	4,2	4,73	1,43	2,16
L_{cap}	m	3,0	3,0	3,5	3,5
Q_{main}	kg	12,6	11,31	20.1	17,95
$Q_{auxiliary}$	kg	18,06	19,87	8,22	12,42
Q_{total}	kg	30,66	31,18	28,8	30,37
Slowing down	ms	20,0	20,0	20,0	20,0
Amount	m^3	90,0	90,0	122,5	122,5
q	kg/m^3	0,34	0,35	0,23	0,25

By using explosive, with engagement of mechanisms and work force, we achieve great productivity in extraction of mineral resource and today, dozens millions tons of ore, rock and other mineral resources are obtained by mining.

This paper considers variants with different drilling diameters and different explosive combinations as main and auxiliary charges. Based on results and analysis we can determine optimal combination of explosives, and after that geometry of mine holes and mine fields, and construction of explosive charge in hole.

By analyzing drilling and mining parameters we can conclude that the most favorable case is mining with 89 mm diameter and using powdery explosive as main charge and AN-FO explosive as auxiliary charge in hole from the aspect of capacity and specific explosive consumption per mass unit of mined rock material.

Table 4. Economic evaluation of mining variant with drilling mine holes with "ROC F7"

		Mining with mining holes with diameter of 76 mm, height bench 10,0 m		Mining with mining holes with diameter of 89 mm, height bench 10,0 m	
Main charge		"AN–FO"	powdery	"AN–FO"	powdery
Auxiliary charge		emulsion	AN–FO	emulsion	AN–FO
L_{hole}	m	11,54	11,54	11,7	11,7
Q_{main}	kg	12,6	11,31	20,1	17,95
$Q_{auxiliary}$	kg	18,06	19,87	8,22	12,42
amount	m^3	90,0	90,0	122,5	122,5
Costs of drilling	KM/m^3	0,48	0,48	0,46	0,46
Costs of explosive	KM/m^3	0,76	0,67	0,76	0,64
Costs of iniciation		0,13	0,13	0,13	0,13
Detonation fuse	KM/m^3				
Slowdown	KM/ piece				
Σ	KM/m^3	1,37	1,28	1,35	1,23

4. RECAPITULATION OF RESULTS

Economic evaluation is based on average costs of drilling mining holes, costs of explosives and costs of initiation. Costs of drilling mining holes were calculated from capital and operational costs of the operation itself.

In table 4., costs of drilling and mining and economic rating of the best choice mining variant is shown.

By analyzing cost results per mass unit of fragmented material, we can conclude that even by economic rating, the most favourable mining variant is with powdery explosives as main charge and "AN-FO"' explosive as auxiliary charge for chosen height bench 10 m and diameter 89 mm drilling equipment "ROC F6".

Table 5. Costs of drilling and mining and economic rating of the best choice mining variant

		Mining with mining holes with diameter of 76 mm, height bench 10,0 m		Mining with mining holes with diameter of 89 mm, height bench 10,0 m	
Main charge		"AN–FO"	powdery	"AN–FO"	powdery
Auxiliary charge		emulsion	AN–FO	emulsion	AN–FO
L_{hole}	m	11,54	11,54	11,7	11,7
Q_{main}	kg	12,6	11,31	20,1	17,95
$Q_{auxiliary}$	kg	18,06	19,87	8,22	12,42
amount	m^3	90,0	90,0	122,5	122,5
Costs of drilling	KM/m^3	0,41	0,41	0,39	0,39
Costs of explosive	KM/m^3	0,76	0,67	0,76	0,64
Costs of iniciating		0,13	0,13	0,13	0,13
Detonation fuse	KM/m^3				
Slow down	KM/piece				
Σ	KM/m^3	1,30	1,21	1,28	1,16

5. CONCLUSION

From results of analysis done in this paper, it is possible to bring some conclusions :

1. Working environment, is rock material in which drilling and mining is done has crucial importance for defining parameters by which drilling and mining will be done, from economic and technical side.
2. Influence of drilling and mining effects on work of equipment and machines in other technical fazes exploitation - loading, transport and proccessing (crushing and grading), is big because working and producing properties of equipment are based on drilling and mining parameters.
3. Trough analysis of specific explosive consumption and geometry of mine field, it is visible that using bigger mine hole diameter can have favourable effect on drilling and mining on big scale mining.
4. The most favourable combination of explosives in charge is calculated in this paper, and every offered mining variant is economicly evaluated, with definition of all other drilling and mining parameters.

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DOI: 10.7251/BMC170701129T

PARTICLE SIZE DISTRIBUTION OF IRON ORE SLUDGE DETERMINED BY USING DIFFERENT METHODS AND IRON CONTENT BY SIZE CLASS

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ABSTRACT

This paper presents the results of the study of the particle size distribution of the limonite sludge that occurs as a hydro-cyclone overflow in the Omarska iron mine. The primary role of the precise characterization of grains of mineral raw material is in obtaining quantitative data about their particle size and analysis of the distribution of mineral grains by volume, which is a necessary step in controlling the processes of the iron ore beneficiation. Since the overflow of the hydro-cyclone size class is -25 μm, in this paper the following methods of sub-sieve analysis are used: sedimentation method-Beaker decantation; pipetting method by Andreasen-Borner; and Warman cyclosizer. Different methods were used in order to analyze the compliance of the results, as well as comparison of them, because expensive and rapid instrumental methods are often unavailable in practice. Also, the content of iron in the sludge varies, so the correlation of iron content with granulometric particle distribution was analyzed. These analyzes were carried out on two samples of sludge with different average iron content (29.43% and 41.19%). On both samples the iron content was determined per size class. The results showed a largest iron content in the smallest classes (-9+6 microns). These results indicate the necessity of applying some of the methods for selectively separating fine particles (such as selective flocculation, for example) in a further process of preparation.

Key words: particle size distribution, limonite sludge, Beaker, Andreasen-Borner, Warman

1. INTRODUCTION

Particle size is one of the most important factor in determining suitable technique in mineral processing. Many properties are dependent on particle size, such as: electrical properties (chargeability, surface conductivity, dielectric), dispersivity, specific surface area, etc., and they are decisive for the selection of an appropriate method of separating of useful minerals from gangue in stable dispersions, such as sludge [1-3]. Sludge, defined as –20 μm particles,

are an unseparable part of finely ground minerals and may contain significant amounts of valuable minerals. Conventional techniques, such as: gravity concentration, magnetic separation and flotation, are generally ineffective in treating sludge. Thus, sludge are often discarded prior to concentration. Discarded sludge are not only an economical loss, but their disposal may be an environmental concern and costly to manage, as well. Many techniques have been investigated to treat sludge, but these systems are complex and each case is subject of special research [4-8].

In mineral processing plant of Omarska mine (Bosnia and Herzegovina), the iron ore from mineral deposit "Buvač" is prepared by the classification of the coarse classes and magnetic concentration of the classes (-0.500+0.025 mm). Class -0.025 mm represents tailings, but sometimes it contains up to 30% mass with high content of iron. Filter-press selectively treats the rich part of sludge and compensates for the mass loss at new magnetic separators. The structural-texture characteristics of limonite ore from the mine "Buvač" cause to obtain, already upon excavation, the major amount of smaller classes, with grit below 25 μm. Every further procedure enlarges the part of fine classes from which the concentrate of sufficient quality can't be extracted in the existing plant for mineral processing in Omarska, and as such it represents tailings sludge. This sludge is not suitable for deposition because it represents stabile suspension and related to that, a problem for the environment. On the other hand, formation of sludge, with approximately 50% Fe or over 75% limonite as waste, represents a major loss of limonite and decrease of economic effects. In this stable suspension the particles below 10 μm (ultrafine) remain dispersed for a long time. Because the movement of the smallest particles through fluid is slowing down, the traditional procedures of mineral processing of this material are not applicable [9]. Before choosing adequate methods for the treatment of this sludge, it is necessary to know the particle size distribution and iron content by size class.

2. MATERIALS AND METHODS

2.1. Materials

For this study, we used two samples of limonite sludge (Sludge I and Sludge II), which have different iron contents (29.43% and 41.19%) and densities (3.000 g/cm^3 and 3.526 g/cm^3), respectively. A samples of the sludge were taken as an overflow on hydro-cyclone plant for the preparation of iron ore in Omarska mine, Bosnia and Herzegovina. The natural pH of the samples ranged from 7 to 7.2. The mineral phases present in the sludge were investigated and characterized earlier [10]. According to the obtained results of XRD, IR and SEM-EDS analysis, both of sludge samples are composed of major goethite and quartz, less clay minerals, and minor magnetite and todorokite. Also, it was established that the phases composition of two analyzed sludge samples are almost the same, but that their content varies [10]. Therefore, the obtained densities in this paper are in excellent concordance with the obtained qualitative – semi-quantitative sludge mineral compositions, and especially to the ratio of goethite (and other minor Fe minerals) to quartz (and other silicate minerals, such as clay) [10]. It should be mentioned here that such sludge compositions are primarily derived by natural raw materials compositions, which were previously also precisely characterized [11].
As dispersant, we used sodium hexametaphosphate, $Na_6P_6O_{18}$, analytical grade, manufactured by Lach-Ner, s.r.o. (Czech Republic). The working solutions are made with distilled water.

2.2. Methods

For particle size analysis of fine particles, different methods were used, which can generally be divided into traditional sedimentation methods (based on Stokes' Law) and different instrumental methods. All of the methods have their advantages and/or disadvantages. In this study three common methods of sub-sieve analysis have been used: (1) Beaker decantation (sludge I and II); (2) Andreasen-Borner pipette method (sludge I and II); and (3) Warman cyclosizer-elutriation method (only sludge II). A detailed description of the above methods is given in the cited literature [1, 12-14]. Here, we present briefly description of each of them.

2.2.1. Beaker decantation

One of the most commonly used procedures for determining the particle size distribution of the raw material is certainly "Beaker decantation" proposed by E.J. Robert in 1953. [12]. This is a tehnique used to separate mineral sample in two or more size fraction according to the differences in setlling velocities of the particles. It is tehnique which can be used to accurately split a sample at a pre-determined cut size, but has some disadvantages because it is a time consuming method, and especially when spliting including a fine sizes. Also, a dilute solids content is required to stop natural coagulation from occuring.

For the procedure of examination using Beaker, we used 100 g limonite sludge, dried on 105 ^{o}C, two cups of 2 liters and six cups of 1 liter volume. For dispersant, we used 0,002 M sodium hexametaphosphate. The pulp was gently stirred to disperse the particles through the whole volume and then was allowed to stand for the calculated time. The water above the end of the tube was syphoned off, dried and weighed.

Velocity of the particles is calculated according to the form:

$$v_0 = 545(\rho_m - \rho_t) \cdot d^2 \left[mm/s\right] \quad (1)$$

Thime of settling:

$$t_1 = \frac{H_1}{v_0 \times 1{,}05}\left[s\right] \quad (2)$$

2.2.2. Andreasen-Borner method

Determining the granulometric distribution of materials by the Andreasen-Borner method belongs to a group of differential methods. This method is based on the same principle as the sedimentation scale, but here the deposition rate is not automatically registered and continuous. In the pipette method, concentration changes occurring within a settling suspension are followed by drawing off definite volumes, at predetermined times and known depths, by means of a pipette. Although, theoretically, errors can be reduced by the use of more complicated construction and operation, it is highly debatable as to whether this is worthwhile for routine analyses, since conventional apparatus is reproducible to ±2% if operated with care. This technique is a standard procedure since both the Stokes diameter and the mass undersize are determined from first principles. The method is versatile, since it can

handle any powder that can be dispersed in a liquid, and the apparatus is inexpensive. The analysis is, however, time consuming and intensive.

For the procedure of examination using Andreasen method, we used Andreasen pipette, which represents a cylindrical vessel of about 500 ml volume and 2.5 g limonite sludge, dried on 105 ^{o}C. For dispersant, we used 0,002 M sodium hexametaphosphate. The suspension was alloved to settle and at given intervals of time, samples are withdrawn by applying suction to the top of the 10 ml reservoir. The samples ware then dried and weighed, and the weights compared with the weight of material in the same volume of the original suspension. Times of settling were calculated using equation *(3)* and shown in Table 1:

$$t = \frac{18 \cdot h \cdot H}{\left(r_m - r_f\right) \cdot 981 \cdot d^2}\left[s\right] \qquad (3)$$

Table 1: Pipette time for two samples of limonite sludge

Particle size, µm	Pipette time (sludge I)			Pipette time (sludge II)		
	h	m	s	h	m	s
"0" probe						
25		5´	9´´		3´	50´´
18		9´	42´´		7´	19´´
13		18´	11´´		13´	38´´
9		37´	11´´		28´	00´´
6	1	22´	58´´	1	2´	37´´

2.2.3. Warman cyclosizer – elutriation method

One of the widely used methods of elutriation, determination of the granulometric composition of mineral resources, in the laboratories for the preparation of mineral raw materials is in Warman cyclosizer (Finch and Leroux,1982.) [1]. This method of determining the particle size of small diameter is most commonly used for routine testing and control in industrial processes, for particle size ranges of 8-50 micrometers and a density of about 2.7 g/m^3, or 4 micrometer classes for material with higher density.

Particle size distribution of sludge II using Warman cyclosizer were performed in the laboratory Global Research and Development, Mining and Mineral Processing, Maizières-lès-Metz, France. Data were obtained on CYCLOSIZER M_{16} – MARC TECHNOLOGIES, 230 V, 50 Hz under following work conditions: pressure 270 kPa, temperature of water 18.5 ^{o}C, flow rate of water 120 l/min, time for test 15 min, particle size of sample -25 µm, specific density of sample 3.526 g/cm^3.

2.2.4. Chemical analyses

Chemical analyses were performed according to BAS ISO 2597-1:2012 standard for determination of iron in iron ore.

2.2.5. Density

Density of sludge samples is determined by the method using a pycnometer. Samples were dried at 105 ^{o}C to constant weight prior to determination, and distilled water was used at room temperature at 20 ^{o}C.

3. RESULTS

3.1. Results of the particle size analyses using Beaker decantation method

Results of the particle size analyses using Beaker decantation with iron content by size class are shown in Table 2 and Figure1 (Sludge I) and Table 3 and Figure 2 (Sludge II).

Table 2. Particle size distribution of Sludge I

Actual Fraction (µm)	Weight (g)	% Weight	% OS Cum.	% US Cum.	Fe mass%
+25	10.42	10.42	10.42	100	29.05
-25+18	6.77	6.77	17.19	89.58	27.63
-18+13	8.96	8.96	26.15	82.81	27.34
-13+9	12.60	12.60	38.75	73.85	28.71
-9+6	11.88	11.88	50.63	61.25	30.23
-6+0	49.37	49.37	100	49.37	28.73
TOTAL	100	100		0.00	

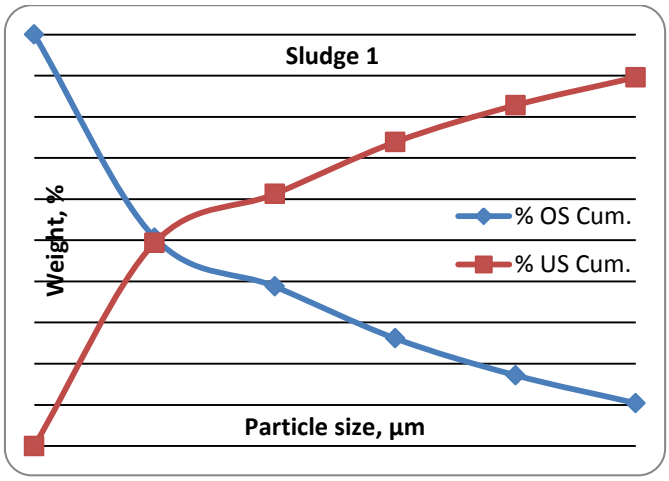


Figure 1. Particle size distribution of Sludge I

The most part of the mass is in fraction -6+0 µm (49.37%, Table 2) which coincides with previous research [15]. This chemical analysis show slight % increase of iron (Fe) in fraction -9+6 µm (30.23%) compared to the initial sample Sludge I (29.43%).

Table 3. Particle size distribution of Sludge II

Actual Fraction (µm)	Weight (g)	% Weight	% OS Cum.	% US Cum.	Fe mass%
+25	13.83	13.83	13.83	100	44.53
-25+18	7.58	7.58	21.41	86.17	38.78
-18+13	7.93	7.93	29.34	78.59	40.09
-13+9	8.29	8.29	37.63	70.66	41.49
-9+6	9.11	9.11	46.74	62.37	43.27
-6+0	53.26	53.26	100	53.26	37.38
TOTAL	100	100		0.00	

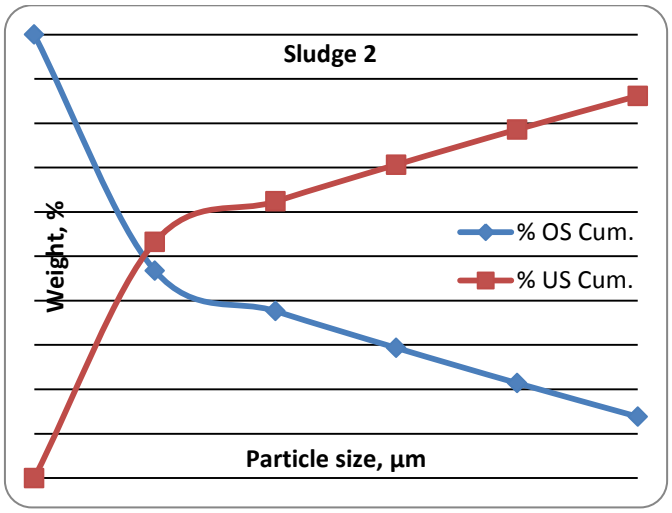


Figure 2. Particle size distribution of sludge II

As in the previous experiment, the most part of the mass is in fraction -6+0 µm (53.26%, Table 3) which coincides with previous research [15]. Chemical analysis show significantly % increase of iron (Fe) in fractions +25 µm (44.53%) and -9+6 µm (43.27%) compared to the initial sample Sludge II (41.19%).

3.2. Results of the particle size analyses using Andreasen-Borner method

Results of the particle size analyses using Andreasen-Borner method with iron content in the fraction -6+0 µm, are shown in Table 4 (Sludge I) and Table 5 (Sludge II). It was not possible to make chemical analysis in all particle size class (except in the fraction -6+0 µm), because of the low mass in all of the other fractions using for testing by Andreasen-Borner method.

Table 4. Particle size distribution of Sludge I

Actual fraction (µm)	Weight of material, g		Weight of material, %		Fe mass%
	sum	dif	W,%	Σ W,%	
"0" probe	0.0917		3.70	3.70	
+25	0.0836	0.0081	3.30	7.00	
-25+18	0.0836	0.00	3.30	10.30	
-18+13	0.0724	0.0112	2.90	13.20	
-13+9	0.0687	0.0037	2.75	15.95	
-9+6	0.0687	0	2.76	18.71	
-6+0	Difference to 100%		81.29	100.00	18.80

Table 4. shows that the most part of the mass is in fraction -6+0 µm (81.29%) which is a significant mass fraction in the sample compared with other fractions. This chemical analysis show significantly % decrease of iron (Fe) in fraction -6+0 µm (18.80%) compared to the initial sample (29.43%).

Table 5. Particle size distribution of Sludge II

Actual fraction (µm)	Weight of material, g		Weight of material, %		Fe mass%
	sum	dif	W,%	Σ W,%	
"0" probe	0.0850		4.11	4.11	
+25	0.0790	0.0060	3.82	7.93	
-25+18	0.0785	0.0005	3.80	11.73	
-18+13	0.0681	0.0104	3.30	15.03	
-13+9	0.0652	0.0029	3.15	18.18	
-9+6	0.0576	0.0076	2.79	20.97	
-6+0	Difference to 100%		79.03	100.00	28.94

Table 5. shows that the most part of the mass is in fraction -6+0 µm (79.03%) which is a significant mass fraction in the sample compared with other fractions. This chemical analysis show significantly decrease % of iron (Fe) in fraction -6+0 µm (28.94%) compared to the initial sample (41.19%).

3.3 Warman cyclosizer

The particle size analyses were performed only on sludge II sample. Results of the particle size analyses using this method with iron content by size class are shown in Table 6 and Figure 3. It was listed original data of diameter range and actual fraction.

Table 6. Particle size distribution of Sludge II

Cyclone No	Diameter range (µm)	Actual Fraction (µm)	Weight (g)	% Weight	% OS Cum.	% US Cum.	Fe mass%
UF 1	44	25	4	8	8	92	43.43
UF 2	33	18.75	1.3	2.6	10.6	89.4	37.94
UF 3	23	14.06	3.1	6.2	16.8	83.2	37.76
UF 4	15	10.54	3	6	22.8	77.2	38.52
UF 5	11	7.91	4	8	30.8	69.2	39.77
OF		5.93	34.6	69.2	100	0	40.92
		TOTAL	50	100			

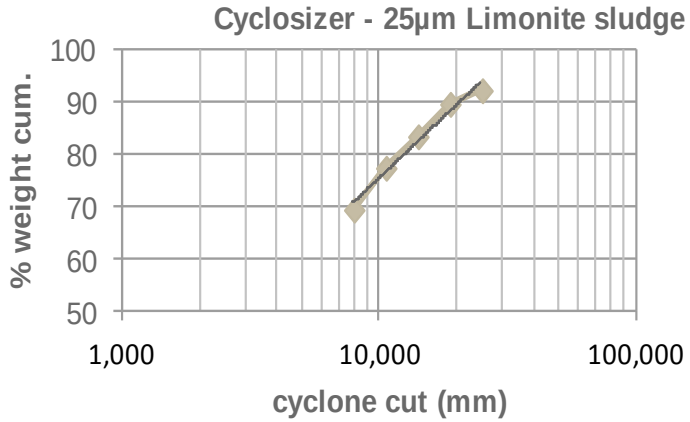


Figure 3. Particle size distribution of sludge II

Results of particle size distribution on Warman cyclosizers also show that the largest proportion of the mass (69.2) is in the class -5.93+0 µm. This chemical analysis show % decrease of iron (Fe) in fraction -5.93+0 µm (40.92%) and % increase of iron (Fe) in fraction +25 µm (43.43%) compared to the initial sample (41.19%).

4. DISCUSSION

As mentioned above, for selecting the most suitable method for mineral processing, one of the most important things is to determine the particle size distribution. The properties that depend on the particle size distribution, such as electrical, reactive and transport properties of particles, are the most important features especially when it comes to fine particles. Also, it is necessary to know the content of useful substances, in this case iron content by size class. Due that the iron content of the overflow in the hydro-cyclone varies, we analyzed the two samples with different initial iron content. Particle size distribution and iron content on sample II were analyzed on a cyclosizer and the first task was to compare these results with the results obtained by some of the other common methods. For this purpose, both samples were analyzed using Beaker decantation and Andreasen-Borner pipette methods.

4.1. Comparison of the used methods

Figure 4 represent comparison of the results of determination of particle size distribution using the three methods (in the brackets are given the original data for particle size limits on the cyclosizer).

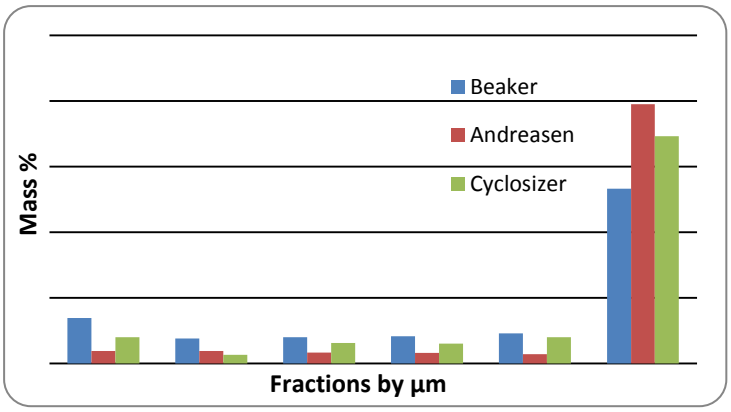


Figure 4: The comparison of content of individual fractions in mass% determined by different methods

The differences between the individual methods were compared using the two-method conversion factor (FC); FC = Method A/Method B. In Table 7 are given calculated values of FC and D (differences in relation to FC = 1 as absolute value), and the sum of D as an indicator of deviation of the results obtained by different methods. Tags B, A and C relate to the methods (B-Beaker decantation, A-Andreasen pipette and C-Warman cyclosizer).

Table 7. Differences between methods by fractions

Particle size (µm)	Compared methods	FC	D	ΣD
+25	B:A	3.6	2.6	4.4
	B:C	1.7	0.7	
	C:A	2.1	1.1	
-25+18(18.8)	B:A	2.0	1.0	3.2
	B:C	2.9	1.9	
	C:A	0.7	0.3	
-18+13(14.1)	B:A	2.4	1.4	3.0
	B:C	2.4	1.4	
	C:A	0.8	0.2	
-13+9(10.5)	B:A	2.6	1.6	2.9
	B:C	1.4	0.4	
	C:A	1.9	0.9	
-9+6(7.9)	B:A	3.3	2.3	4.3
	B:C	1.1	0.1	
	C:A	2.9	1.9	
-6(5.9)+0	B:A	0.7	0.3	0.6
	B:C	0.8	0.2	
	C:A	0.9	0.1	

All three methods have shown the largest mass of tested samples contained in the finest fraction (Figure 4) and the slightest deviation of the result of the methods used to each other just for the finest fraction (Table 7). This indicates that in our case we can apply all three methods. However, in further analysis we rejected the Andreasen pipett method because of the impossibility of determining the content of iron by particle size classes for the above-mentioned reasons.

4.2. Determination of iron content by the class size

In Table 8 are given iron contents by the class size for both of samples. There are some deviations in the iron content by the class size using the different methods for particle size distribution in the case of sludge II (Warman cyclosizer and Beaker). In some classes, the difference is significant, as we can see from Table 8. Compared to the initial content of iron, better matching of average is in the case of the Beaker method for sludge II.

Table 8. Iron content by the class size

Actual fraction (μm)	Sludge II			Sludge I
	%Fe (C)	%Fe (B)	Deviations between C and B (%)	%Fe (B)
Initial sample	41.19			29.43
+25	43.43	44.53	2.47	29.05
-25+18(18.8)	37.94	38.78	2.16	27.63
-18+13(14.1)	37.76	40.09	5.81	27.34
-13+9(10.5)	38.52	41.49	7.15	28.71
-9+6(7.9)	39.77	43.27	8.08	30.23
-6(5.9)+0	40.92	37.38	7.52	28.73
Average	39.72	40.92		28.62

4.3. Comparison of two samples of the sludge iron content

Finally, the results of two samples of sludge with different initial iron content can be discussed. The particle size distribution was determined by the Beaker decantation method for samples with different initial iron content. On both samples the iron content was determined per size class. Calculated average iron contents give a satisfactory agreement in comparison with the initial content. Due to the transparency, the results for two samples are summarized in Table 9.

Table 9. Particle size distribution and iron content by size class for two investigated samples

	Sludge I		Sludge II	
Actual Fraction (μm)	mass %	Fe mass%	mass %	Fe mass%
Initial	100	29.43	100	41.19
+25	10.42	29.05	13.83	44.53
-25+18	6.77	27.63	7.58	38.78
-18+13	8.96	27.34	7.93	40.09
-13+9	12.60	28.71	8.29	41.49
-9+6	11.88	30.23	9.11	43.27
-6+0	49.37	28.73	53.26	37.38
Average	-	40.92	-	28.62

Although in some classes a slightly higher iron content is observed compared to the initial, these differences are not sufficient reason for further research to be done on separate classes. A large mass percentage of the finest class in both samples also supports this. Bearing in mind that the separation of such fine classes is very difficult and complicated, this procedure is justified only in the case of large differences in the content of iron. Particularly interesting is that there is no big difference in the classes of the particle size distribution of the two samples although the mineral phases are present in various contents. This indicates that all mineral phases, both useful and useless, are present in the sludge as very fine particles which makes this system very complex for further separation process. Therefore, our future research will be aimed at defining the surface charge of natural minerals that have been already identified [10,11] as the main components of the system and their behavior in the presence of different dispersants and flocculants.

5. CONCLUSION

The particle size distribution and iron content by size classes of the limonite sludge that occurs as a hydro-cyclone overflow in the Omarska iron mine were studied on two samples with different initial iron content. According to the obtained results, it can be concluded that the both samples are composed mainly of the most finest particles (below 6 microns). The content of iron by particle size classes mostly corresponds to the content of iron in the initial samples, but there is no significant difference in iron content by classes in any sample. The results showed a large iron content only in the finest classes. The rather uniform distribution of iron by class and a large percentage of the smallest particles confirm that all mineral phases in the sludge are present as fine particles regardless of the initial sample. There are not sufficient reasons for further research to be done on separate classes and it will be done on bulk samples. Future research will be focused on the study of surface properties (primarily electrical properties), as for sludge, as well as for the individual phases of which are part of the sludge.

Acknowledgment: *The authors are grateful to Company ArcelorMittal in Prijedor for technical support and enabling the study visit of Miss Lj. Tankosić to the laboratory Global Research and Development, Mining and Mineral Processing, Maizières-lès-Metz, France, for the purpose preparation of her PhD.*

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DOI: 10.7251/BMC170701141B

REVIEW AND CHARACTERIZATION OF HYDRAULIC TRANSPORT OF THE "OMARSKA" MINE TILL 2016.

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ABSTRACT

This paper presents the analysis and verification of parameters of hydraulic transport of tailings from the plant for the preparation of mineral raw materials of magnetic separation of the mine "Omarska" for 2016. For the purpose of transportation of waste sludge from the plant, there is a pipeline of length 5,47 km which connects the sludge pump station and the waste dump lake "Gradina".

On the basis of the calculated for the transport of tailings by pipelines to the waste dump as well as the efficiency of the pumping plant, the possibilities of using a pipeline of a smaller diameter in relation to the already installed Ø177,2/8 mm were considered. The real condition calculation is verified (or is not) a pump that suitably to actual hydraulic transport conditions and installed pipeline.

Key words: hydraulic transport, tailings, disposal of tailings

1. INTRODUCTION

Surface exploitation of mineral raw materials is experiencing an expansion with a constant increase in demand for the world market for different types of these products. Transport as a technological phase of exploitation participates in the overall structure of the exploitation costs of 40-60%. If the transported material requires, a hydraulic trasport is applied. Hydraulic transport makes sense to transport materials that do not change their physical properties or react chemically in contact with water. Today, sand, gravel, limestone, ash, slag, coal and all kinds of ore, their voices and others are mainly hydraulically transported.

The idea water transporting crushed materials appeared very long ago, and the full development of hydraulic transport is associated with the year 1957 when a pipeline for the transport of mined coal from the Cadiz mine (Cadis) to the thermal power plant in Cleveland (Cleveland) m and a diameter of 254 mm with three pumping stations. The important pipeline

for hydraulic transport was put in operation in 1957. This pipeline is still transporting gilsovit (hydrocarbon minerals, which are refined by gasoline, oil and coke) about the mine in the state of Utah to the refinery in the state of Colorado. The length of the pipeline is 116 km and the diameter is 152.4 mm. It has only one pump station with three piston pumps, which two work simultaneously, and the third is a reservoir. Table 1 shows pipeline longer length and longer diameter wich where constracted and started to producing the last few years.[1]

Table 1. Caracteristic length, diameter and mass flow pipeline .[1]

Country	Transported material	L	D	$\dot{m}_s$
		km	mm	10^6 t/year
Canada	Iron ore	645	252	1
Canada-USA	Iron ore	805	458	10
Rusia	Coal	430	400	4
Arizona	Coal	440	457	4,4
Illinois-Chicago	Coal	550	-	-
Jutah-San Diego	Coal	1560	-	-
Arizona	Coal	435	460	-
Polland	Coal	210	250	-

The main goal of this paper is to presents the analysis and verification of parameters of hydraulic transport of tailings from the plant for the preparation of mineral raw materials of magnetic separation of the mine "Omarska" for 2012. For the purpose of transportation of waste sludge from the plant, there is a pipeline of length 5.47 km which connects the sludge pump station and the waste dump lake "Gradina". On the basis of the calculated for the transport of tailings by pipelines to the waste dump as well as the efficiency of the pumping plant, the possibilities of using a pipeline of a smaller diameter in relation to the already installed Ø177.2/8 mm were considered. The real condition calculation is verified (or is not) a pump that suitably to actual hydraulic transport conditions and installed pipeline. [2]

2. TECHNICAL DESCRIPTION

For the needs of transportation of waste sludge from the magnetic separation plant of the Omarska mine, the construction of a pipeline for sludges is 5.47 km length was connected with the sludge pumping station and the landfill of the Gradina lake. Drainage of sludge from the sludge station to the open channel on the dam Medjedja is made of steel pipes Č.05625, longitudinally welded "11. OKTOMURI" Kumanovo, thickness d=8 mm, calculating diameter Ø177, 2/8 mm. [2]

The wall of thickness of the pipe is selected to protect against solid particles in the composition of the sludge. For the same reason, the thickness of the pipe wall from the pumps to the point of installation of the butterfly-return valve, as well as the thickness of the wall of the pipes of all the knees should be 10 mm. Sheme of tailings pipeline is shown in Figure 1. [2]

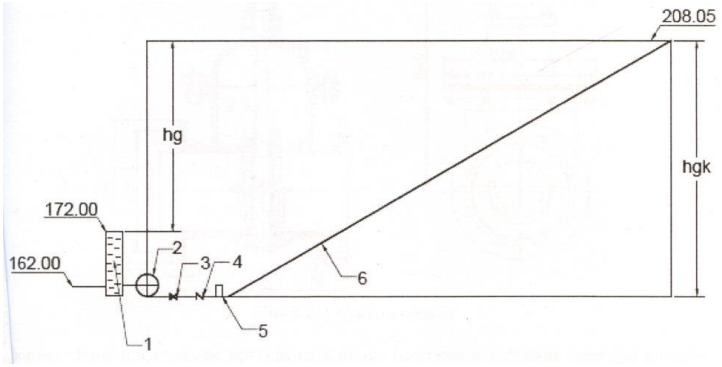


Figure 1. Sheme tailings pipeline [3]

Description:

1. Tailings collector
2. Pump
3. Valve
4. Butterfly–return valve
5. Boiler for compensation of hydraulic impact
6. Pipeline

3. CALCULATION OF THE HYDRAULIC TRANSPORT SYSTEM

Calculation and dimensioning of the transport system [3] implies take place in three stages:

1. calculating missing parmeters (quantity of tailings, ratio od solid and liquid phase, working hours),
2. dimensioning of pipeline (according to the ratio of actual and critical speed) and their choice according of the tipe of pipeline material with adjustment calculated dimension with standard and offer on the market,
3. calculating tailings pipeline (according configuration lenght, diameter and the tipe of pipeline) and selection of pump according market conditions [2].

3.1. Calculating missing parametars

The quantity of tailings wich is produced in magnetic separation plant "Omarska" [2] for six month in 2012. year is shown in table 2.

Table 2.The quantity of tailings, ratio solid and liquid phase [2]

Month	Qantity [t]	Ratio S:L (solid and liquid phase)	Working hours [hours/month]
Janury	66455	1:11,439	354,58
February	27724	1:12,828	46,33
March	108577	1:11,454	559
April	101324	1:12,563	511,58
May	57662	1:12,624	343,17
June	97716	1:11,134	327,75
Mean value	**76576,33**	**1:12,007**	**357,068**

3.1.1 Flow of hydromixtures

The flow of hydromixtures is determine on the basis of quantity solid tailings, density, qouantity of water and concentration of mass [3].

$$P = \frac{Q}{\frac{\gamma}{1000}} + V\left[m^3/h\right]$$

or

$$P = \frac{Q}{\frac{\gamma}{1000}} + \frac{Q}{\frac{C_m}{100}} - Q\left[m^3/h\right]$$

3.1.2 Density of hydromixtures

The density of hydromixtures is value between density liquid phase (water) and density of solid phase wich transported.

$$\gamma_p = \frac{Q+V}{\frac{Q}{\gamma}+V}\left[t/m^3\right]$$

3.1.3 Volume and mass concentration

Volume and mass concentration defines the participation of the solid phase in a mixture with water.[3]

For volume concentration:

$$C_v = \frac{\gamma_p - \gamma_0}{\gamma - \gamma_0} \cdot 100[\%]$$

For mass concentration:

$$C_{m,} = C_v \frac{\gamma}{\gamma_p}[\%]$$

3.2 Dimensioning pipeline

Dimensioning pipeline compare is deffined with really and critical speed transporting. The reall spead of transport should be (10-30)% bigger then critical speed. [3]

3.2.1 Real and criticalspeed:

$$v = \frac{4 \cdot P}{D^2 \cdot \pi}[m/h]$$

Critical speed is speed wich settling material become on the bottom of the pipeline and formation mobile layer. Critical speed also can be speed in which the loss of energy in the pipeline is the smallest. [3]

$$v_{kr} = F_l \sqrt{2 \cdot g \cdot D \cdot \frac{\gamma - \gamma_0}{\gamma_0}}[m/s]$$

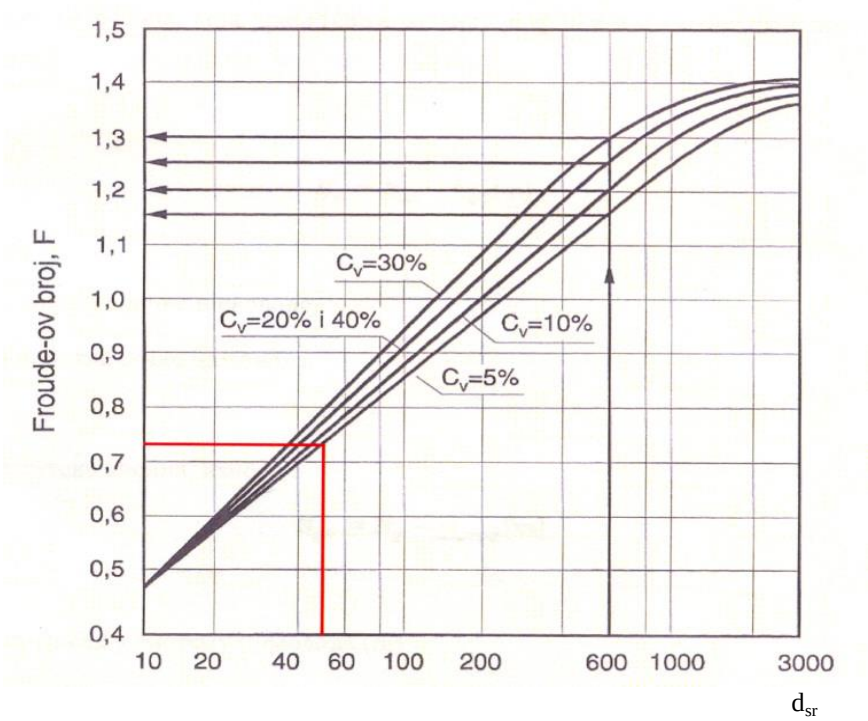


Figure 2. Diagram Mc Elvain and Cave for calculation modification Froud number [1]

3.3 Dimensioning pump effort

Pump effort is work wich should be donne to provide corresponding pressure on the exit from the pipeline, to overcome the geodetic height and to overcome the linear and local resistance of the conveyor system between the input and the output point and the safety supplement called the outlet pressure of the pipeline (Δp_{min}). [3]

3.3.1 Geodetic height

$$H_g = h_{zt} - h_{pt}[m]$$

3.3.2 Final geodetic height:

$$H_{g_k} = H_g - p_{pred}[m]$$

3.3.3 Calculation of resistance

Energy losses are linear and local.. Lineral losses are line losses represent the loss of mechanical energy in overcoming adhesion along the current flow [3].

The often using fomula is Hazen-Williams:

$$i = \sqrt[0,54]{\frac{P}{0,27854 \cdot C \cdot D^{2,63}}}[m/m]$$

Total energy loss due to linear resistance:

$$i_{uk} = L \cdot i[m]$$

Local losses are a consequence of local disturbances of the current flow, such as the existence of knees, valves, closures, narrowing or extending the profile.

$$h_{loc} = \frac{v^2}{2 \cdot g} \sum_1^n \xi_t [m]$$

$$\zeta_{crit} = 0,13 + 1,05 \cdot \left(\frac{r}{R} \right)^{3,5}$$

Finally, the effort required to transport hydromimixture is:

$$H_{hm} = \frac{H_{woter}}{H \cdot R}[m]$$

3.3.4 Selection of the pump

By calculating the pump's effort with the known flow of the hydromechanics, the initial conditions are fulfilled by choosing the appropriate pump.[4]

4. RESULTS OF THE CALCULATING

4.1 Calculated missing parametars

The flow of hydromixture is 2500 m^3/h.On the basis of the known relationship between the liquid and the solid phase [2], the quatitiy of solid sludge can be determined as well as the amount of water in the transported hidromixture.Results of the calculating are presented in Table 3.

Table 3 Input date and calculating parametars

Input date	P	**2500 m^3/h**
	S :L	1:12,007
	γ	2,8 t/m^3
	γ_0	1,0 t/m^3
Calculating parametars	Q	202,198 t/h
	V	2427,787 t/h
	γ	1,052 %
	c_v	2,889%
	c_m	7,699%
	d_{sr}	0,05 mm

4.2 Dimensioning the pipeline

Table 4. The results of dimensioning the pipeline using speeds

Outer diameter of the pipeline	Real speed of transport	Critical speed of transport	$v/v_{critical}$
Ø711,2 mm	1,788	3,638	0,49
Ø419 mm	5,191	2,787	1,86
Ø457,2 mm	4,349	2,913	1,49
Ø508 mm	**3,513**	**3,073**	**1,14**
Ø558,8 mm	2,897	3,225	0,9
Ø609,6 mm	2,429	3,370	0,72

4.3 The pump effort

Pump effort is determend for calculated diameter Ø508 mm pipeline and installed pipeline Ø711,2 mm.

4.3.1 The geodetic height

Table 5. The results of calculating geodetic height

Input date	h_{zt}	208 m
	h_{pt}	162 m
	p_{pred}	10 m
Calculating parametars	H_g	46 m
	H_{gk}	36 m

4.3.2 The line losses and local resistance in the pipeline

Table 6. The results line losses in the pipeline

Input date	Pipeline Ø508 mm	Pipeline Ø711,2 mm
r	0,5017 m	0,7032 m
P	2500 m^3/h	
C	140	120
D	501,7 mm	
L	5465,27 m	
Calculated date		
i	0,017 m	0,00426 m/m
i_{uk}	90,57	23,27

Table 7. The results of calculating line losses in the pipeline

Input date	Pipeline Ø508 mm	Pipeline Ø711,2 mm
g	9,81 m/s^2	
v	3,513 m/s	1,788m/s
ζ_{crit}	3,941	3,911
ζ_v	0,12	0,73
ζ_p	0,1	
Calculated date		
$\sum_1^n \xi_t [m]$	4,161	4,741
h_{loc}	2,62 m	0,77 m

4.3.3 The total pump effort

The results of thetotalcalculationpump effort are given in the table 8 were obtained on base of calculated date forgeodetic height, linear losses and local resistance.

Table 8 The total pump effort

Calculated date	Pipeline Ø508 mm	Pipeline Ø711,2 mm
h_{uk}	93,19 m	24,04
H	134,19 m	65,04 m
K	0,0029	
HR	0,9971	
H_{hm}	134,58 m	65,23

4.4 Selection of the pump

Selection pumpis made on the basis of the summary Q-H diagram of company SVERDMASH.[4]

Table 9 Input data for pump selection

Input date	Pipeline Ø508 mm	Pipeline Ø711,2 mm
H	134,58 m	65,23 m
Q	2500 m^3/h	

For a calculated pipeline (Ø508 mm) with a known flow rate of Q=2500 m^3/h , the the effort H=134,58 m is too high to be able to choose a suitable pump.

For insalled pipeline (Ø711,2 mm) with a known flow rate of Q=2500 m^3/h , the the effort H=65,23 m corresponds to the projected and installed pump type GrT 4000/71a.

5. CONCLUSION

The dimensioning of the pipeline using critical and actual speed analyzed the possibility of using Ø508 mmdiameter pipeline. Based on the effort for the pipeline Ø508 mm and Ø711,2 mm it was found that the piping diameter of the pipeline required much stronger and after that ,more expensive pump.

The performed calculation of the efficiency of the pumping plant and the pipeline showed that existing pipeline Ø508 mm is more suited to current-conditions of hydro transport.

By calucalted the actual state of the hydrotransport, the verification of the projected and installed pump GrT 4000/71a was achieved.

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DOI: 10.7251/BMC170701151T

ASSESSMENT OF THE SITUATIN OF THE DRAINAGE SYSTEM ON THE OPEN PIT OF BUVAČ IN THE PERIOD FROM 2013 TO 2016

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ABSTRACT

The contemporary approach to the process of managing the drainage system and monitoring the effects of their work envisages carrying out control of the operation of all facilities**,** that is the work of the entire system for the protection of the surface and groundwater baths and also developing the continuous monitoring of water events and hydrodynamic processes throughout all stages of the development of the surface mine. The aim of this paper was to collect data based on the amount of precipitation at the monthly level, the state of the water level of the River Gomjenica and the amount of water that will be used to assess the efficiency of the system during the period of operation of the Buvač surface mine from 2013 to 2016.

Key words: drainage, open pit, amount of rainfall, water-level, pumped water quantity

1. INTRODUCTION

The task of drainage is the undisturbed development of the technological process of exploitation, the prevention from damage of mining machinery, equipment, facilities and slopes of the dug, as well as the protection of the limit of the surface mine from the penetration of waters by implementing water protection measures and the construction of facilities provided for water protection.[1]

The technological process of drainage, within the field of exploitation of the mineral resources depends on a number of influential parameters, which within the development of surface exploitation take on increasingly complex forms. At with more deeper depths, the conditions for drainage of open pit are more complex, resulting in a greater number of drainage facilities, as well as a variety of technical solutions that follow the technological drainage process. [2]

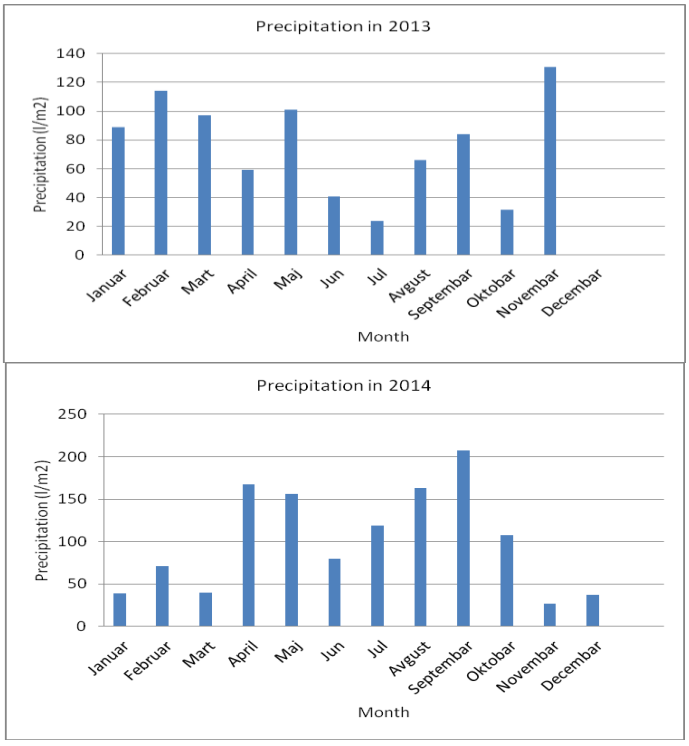
At with more higher depths of exploitation, the conditions for drainage of surface pit mines are more complex, resulting in increased number of drainage facilities. This is particularly true for surface minerals of iron ore, with a biger drainage coefficient, such as the "Buvač".

2. MEASURED VALUE OF THE AMOUNT OF PRECIPITATION FOR 2013, 2014, 2015 AND 2016. YEAR

Monitoring service according to a certain plan of activities carries out activities in its domain in certain time activities: mapping bench and landfills, measuring the level of groundwater and flowing wells, measuring the high of water fall precipitation, recording water levels etc. Table 1 (figure 1) shows the total amount of precipitation at the monthly level during the observed period. [3]

Table 1. The total amount of precipitation on the monthly level for 2013, 2014, 2015 and 2016. year

Month	Precipitation (l/m^2)			
	2013	2014	2015	2016
January	88,8	38,6	44,7	75
February	114,4	70,9	56,4	118,2
March	97	39,5	51,2	76,5
April	59,1	167,4	35,75	43,7
May	101,2	156,3	102,6	85,2
June	40,5	79,8	29,7	57,7
July	23,5	119	45,2	137,1
August	66	163,1	45,6	145,7
September	84,1	207,5	85	98,6
October	31,5	107,5	157,3	99,2
November	130,7	26,7	48,1	53,1
December	-	37,5	3,3	4,8
Total	836,8	1213,80	706,85	994,8



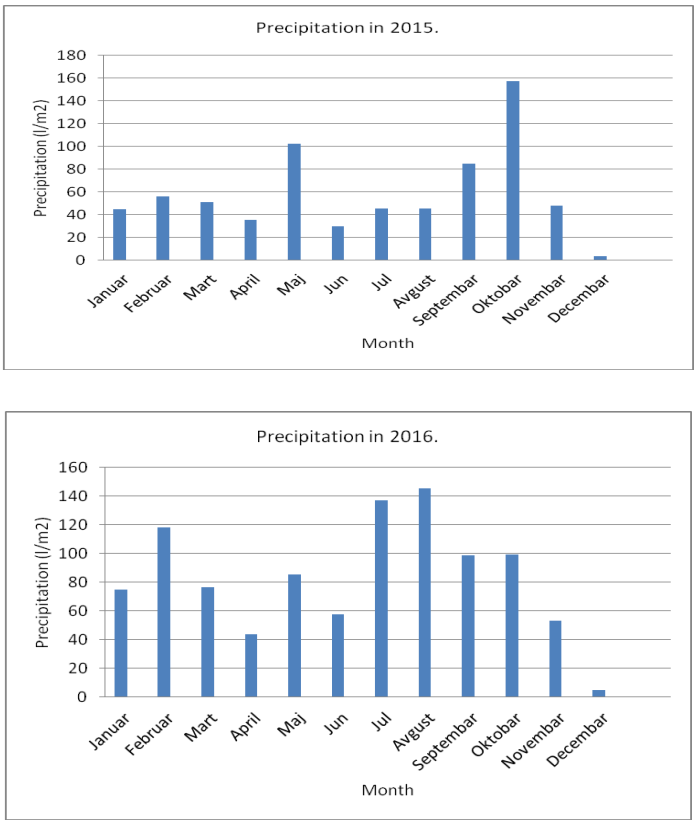


Figure1. The total amount of precipitation on the monthly level for 2013, 2014, 2015 and 2016 year

As it can be seen, in the period January-March 2013, the greatest amount of precipitation was recorded, total 300.2 l/m^2, while for the year 2014 the greatest amount of precipitation was recorded in the period July-September total of 478.1 l/m^2. When it comes to 2015, the greatest amount of precipitation was recorded in the period September-November total of 290.4 l/m^2, and in the year 2016, the period with the greatest amount of precipitation was July-September total of 381.4 l/m^2. The highest amount of rainfall was recorded in 2014.

3. THE ANNUAL REVIEW OF THE WATER-LEVEL OF THE RIVER GOMJENICA FOR 2013, 2014, 2015 AND 2016. YEAR

Within the protection measures of open pit "Buvač" from the water inflow within the exploitation field, the water level of the River Gomjenica is regularly controlled.

Figure 2 shows the annual review of the water level of the River Gomjenica for 2013, 2014, 2015 and 2016 year. [3]

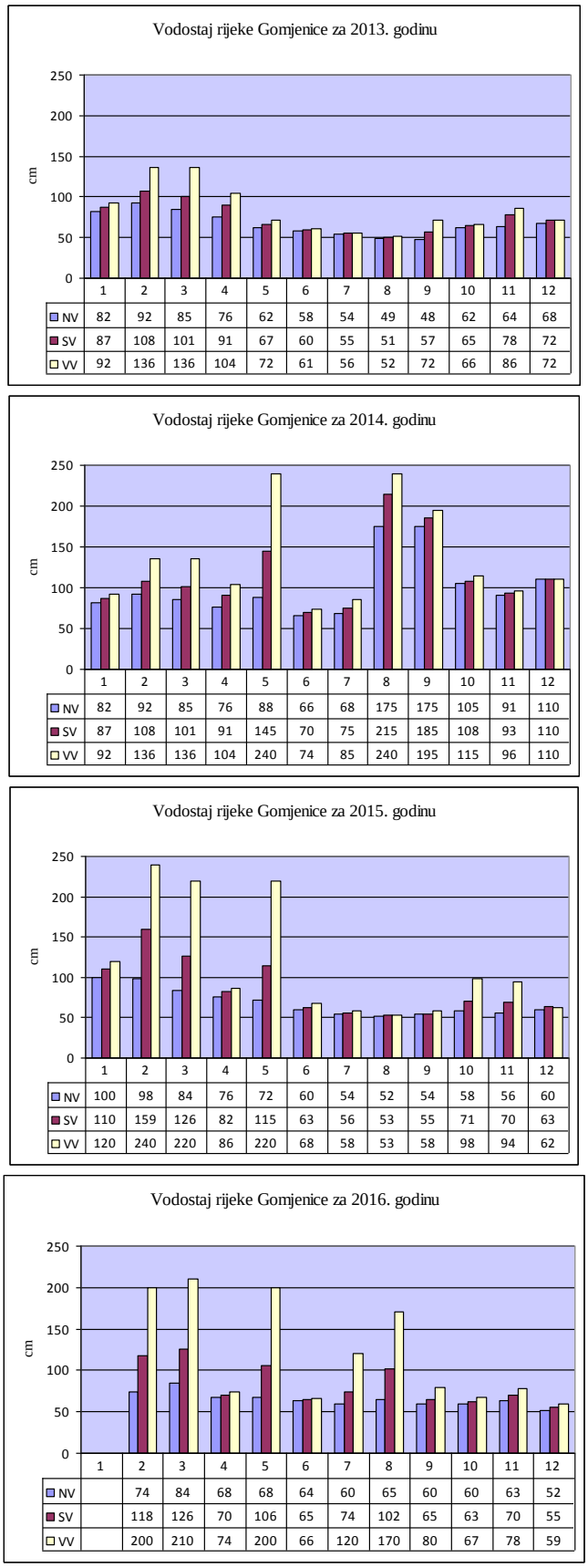


Figure 2. The chart of the annual water-level status of the River Gomjenica for 2013, 2014, 2015 and 2016. year

At with more higher amount of precipitation, the water level in crease, which is also evident from the diagram.

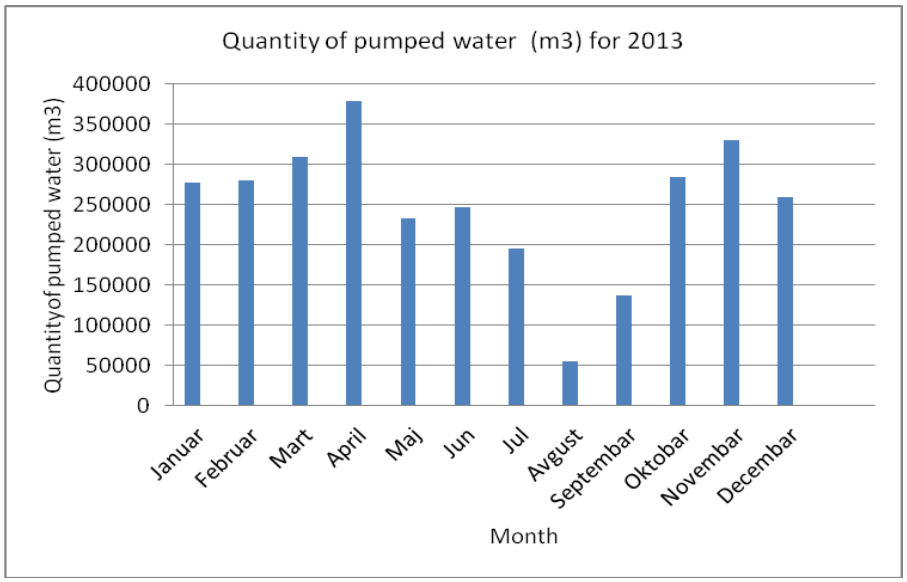
The lowest water level is evident during the dry period of the year. White more over the higher the amount of atmospheric precipitation, the more water level increases linearly. As for the annual review of the water level of the River Gomjenica for 2013, the minimum water level was measured on Aug 26th and Sept 2nd and then it was 49 cm, while the maximum water level was measured twice, on Feb 11th and Mar 18th with the value of 136cm. For the year 2014, the minimum water level of the River Gomjenica was measured on June 20th, with the value of 66 cm, while the maximum water level was measured twice, that is on May 16th and Aug 10th with the value of 240cm. For the year 2015, the minimum water level of the River Gomjenica was measured twice, on Aug 24th and 31st with the value of 52 cm, while the maximum water level was measured on Feb 25th and it was 240 cm. For the period of 2016, the minimum water level of the River Gomjenica was measured twice, on Dec 20th and 27th with a value of 52 cm, while the maximum water level was measured on Mar 24th and it was 210 cm.

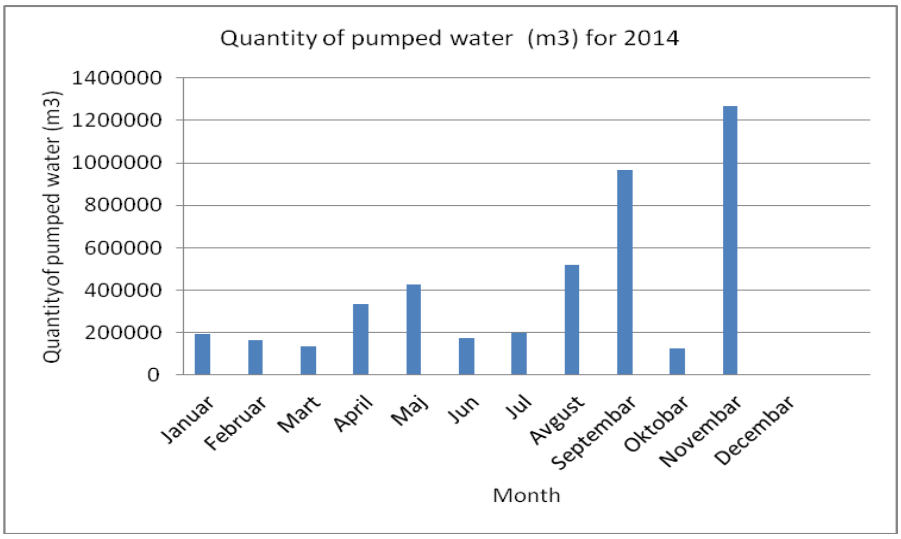
4. THE RESULTS OF THE PUMPED WATER EVALUATION FOR THE MONTHLY PERIOD 2013,2014,2015 and 2016.

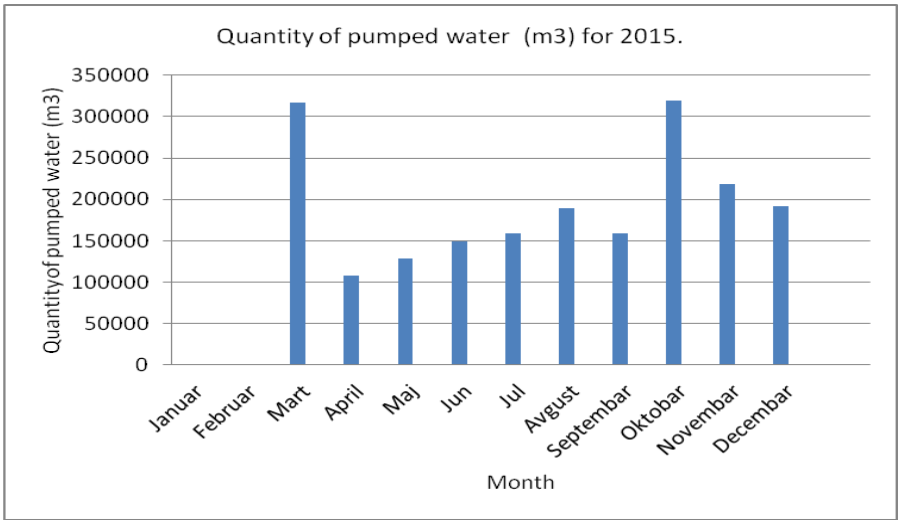
Analyzing daily reports on required hours of pump operation at open pit Buvač and taking into account the effective pump operation time, mechanical and technical delays, as well as the capacities of the available pumps, the data of the pumped water quantities is collected for the observed period in the years 2013, 2014, 2015 and 2016. Table 2 and Figure 3 show the pumped water quantities at the monthly level for 2013, 2014, 2015 and 2016.year. [4]

Table 2. Total quantities of pumped water at the monthly level in the years 2013, 2014, 2015 and 2016

Month	Quantity of pumped water (m^3)			
	2013	2014	2015	2016
January	278073	195835		258283
February	280366	162100		298779
March	309741	136300	316575	302551
April	378314	335400	108025	189997
May	233780	426963	128133	193021
June	246680	175075	148739	138764
July	196141	200650	159245	160837
August	55265	520963	189755	212385
September	137164	967568	158608	123480
October	284901	1205618	319124	213733
November	330202	1267243	219055	208211
December	260100		191650	182953
Total	2 974 485	5593715	1938909	2482994


Quantity of pumped water (m3) for 2013
Quantityof pumped water (m3)
400000
350000
300000
250000
200000
150000
100000
50000
0
Januar
Februar
Mart
April
Maj
Jun
Jul
Avgust
Septembar
Oktobar
Novembar
Decembar
Month


Quantity of pumped water (m3) for 2014
Quantityof pumped water (m3)
1400000
1200000
1000000
800000
600000
400000
200000
0
Januar
Februar
Mart
April
Maj
Jun
Jul
Avgust
Septembar
Oktobar
Novembar
Decembar
Month


Quantity of pumped water (m3) for 2015.
Quantityof pumped water (m3)
350000
300000
250000
200000
150000
100000
50000
0
Januar
Februar
Mart
April
Maj
Jun
Jul
Avgust
Septembar
Oktobar
Novembar
Decembar
Month

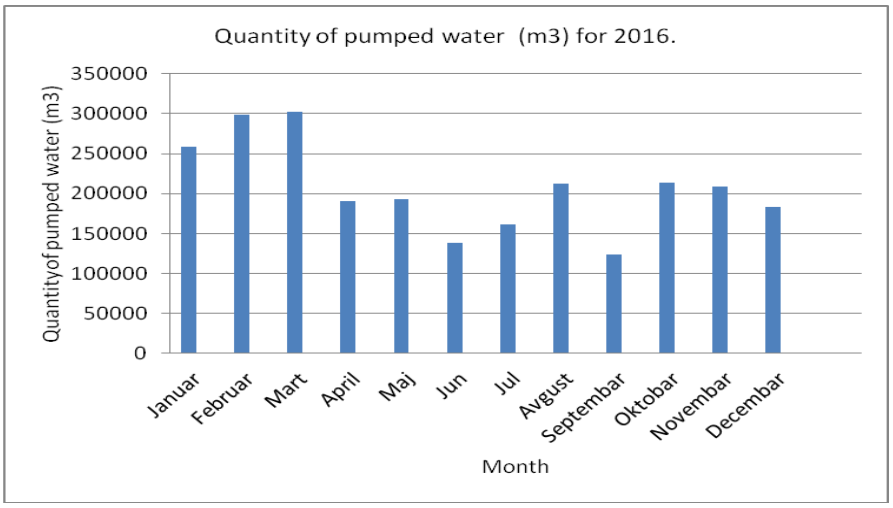


Figure 3. The graphic chart of the total quantity of pumped water in the years 2013, 2014, 2015 and 2016.year

5. CONCLUSION

The obtained results indicate that the level and inflow of water at open pit The "Buvač" extremely varies and it depends on the year period, the amount of precipitation and on water-level which makes it difficult to dimension the capacity of the pumping plant for constant operation.

Upon completion of the construction of each drainage dosed at P.K. "Buvać" and their putting into an operation, the regular monitoring is carried out - monitoring the condition of the same. Also, the records of precipitation, groundwater level and operating hours of pumps are regularly kept by evaluating pump capacity and quantity of pumped water.

In the year 2013, the total amount of precipitation amounted to 836.8 l/m^2, and in the year 2014 it was 1213.80 l/m^2. According to the number of operating hours of pumps and their associated capacities, the amount of pumped water for the year 2013 is 2 974 485 m^3, and for the year 2014 it has the value of 5 593 715 m^3, and for the year 2015 it has the value of 1 938 909 m^3, but for the period 2016, its value is 2 482 994 m^3.

Along with higher amount of precipitation, the water level is rising, which is also evident from the diagram. The lowest water level is evident in the dry period of the year, while with the increased amount of atmospheric precipitation, the water level increases linearly.

The overall evaluation is that all objects at open pit "Buvać" are performed in accordance with the technical solutions given in the project documents, that the state of all hydraulic facilities is satisfactory which means that the established system is functional and reliable.

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DOI: 10.7251/BMC170701159P

MUD MOBILITY TROUGH THE ANNULUS

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ABSTRACT

This paper presents mud mobility, as displacing of mud during casing operation of cementation and influence of misaligment casing on displacing process. In the process of preparing the well for the running casing, the most important factor of the casing is mud preparing. When we speak about mud (drilling fluid), it can act as a Newtonian fluid such as drilling (water, air, oil, etc.) or as the most common case, and these are non- Newtonian fluids. During circulation at various flushing capacities, the mud in the well has a few degrees of flow. Plastic viscosity is an increase in shear stress above the flow point and is caused by mechanical friction resistance. The installation of casing as a process requires circulation of the mud before and after the running of the tube, which speaks about the importance of mud. Under the conditioning of the mud in the preparation well for casing cementation is meant to: reduce the strength of the gel and reduce its plastic viscosity, thus increasing the possibility of displacing of the mud from the annulus.

Key words: mud mobility, muddiespalcing, the influence of the non-centering of the casing, the Newtonian / Non-Newtonian fluids.

1. INTRODUCTION

Drilling process represents a set of different techincal-technological processes realized through the application of a complex system of conducting and monitoring the works. The main goal is that the system must always work reliably and safely with minimal cost of production.

One of the element of technical-technological procedures in the complex drilling system is the preparation of the well for the running of casing.
Running of casing in the well, means the running pipe in well with a smaller outer diameter than the borehole diameter. Casing installed in the well have very important functions, both during drilling and during the lifetime of the exploration.
In the process of preparing the well for the running of the casing, the most important influence of the pipes is mud.

Circulation with mud before cementation is necessary and has the following functions:

- Circulating in the annulus volume for checking whether there was fluid penetration into the annulus,
- Clean the borehole from the material when running casing,

- Erodes a gelatinous or dehydrated mud that is left in extensions and narrowed spaces (non-centric) between the casing and the borehole wall.

In order for the mud to be able to perform these functions, the characteristics of the mud before to the cementation process require adjustment of the correction or conditioning.
Correction of the mud characteristics means reduce the strength of the gel, reduce its plastic viscosity, decrease the density, all in order to better drill the well and later and more efficiently dispalcing the mud from the annulus.

Changes in the rheological characteristics of the mud should be performed before cementation so the mud is as efficiently exited from the annular overflow. This, as well as a number of other conditions between the mud characteristics during running of the casing in the well for cementation, is the problematic of the subject.

2. RHEOLOGICAL CHARACTERISTICS

Mud can behave like Newtonian fluid such as drilling (water, air, oil, and other) or as is the case most commonly known as non-Newtonian fluid. Rheology is a part of mechanics and studies the deformation and flow of matter. When the force acts on the figure causes its deformation, and in the liquid or liquid fluid such force forces the flow. The flow rate or fluid flow is divided into two types:

- Continuous flow
- Uncontrolled flow

Continuous flow is the flow that does not change, and there are two types of flow laminar and turbulent.

Laminated flow is understood to mean the same flow when individual particles tend to move in straight lines parallel to the course without changing the speed. Turbulent flow is a type of flow that is characterized by swirling flow of fluid particles within the flow.

Characteristic of uncontroled flow is the description of the motion of the fluid without a steady flow. The characteristic formation of this flow is at the launch itself, then when the diameter of the pipe through which the fluid circulates is changed.
Flow rate changes with change of velocity are conditioned by the values of Reynolds number and critical speeds.

Mud like non-Newtonian fluid differ from Newtonian fluids, and the essential difference is in the velocity of flow. In Newtonian fluids, the shear stress is directly proportional to the shear rate and fluid is triggered at the same moment when the force becomes larger than the zero and that is given with equation:

$$\tau = \mu \bullet v \qquad (1)$$

Where:
μ-viscosity
v-shearing speed

Non-Newtonian fluids are fluids in which viscosity is in function of flow conditions. Into the muds we have two types flow fluids: Bingham's plastic fluids and Psydoplastic fluids.

Bingham's plastic fluids are those in which the shear stress changes linearly with shear rates but require a certain force to initiate liquidity unlike Newtonian fluids. Pseudoplastic fluids are a fluid model that better mimics real fluid flow and is applied to shear rates less than 150.

Applies for muds with a low content solid phase and for low shearing speeds that are being applied today. Like Newton's liquidity, pseudoplastic fluids are beginning to run at the same time as some force is applied to them, unlike Newtonian liquids, stress on the shear is not proportional to the rate of shear. Flow equation:

$$\tau = k \bullet vn \qquad (2)$$

Where:
k-consistency index
n_n- index of reactive flow behavior

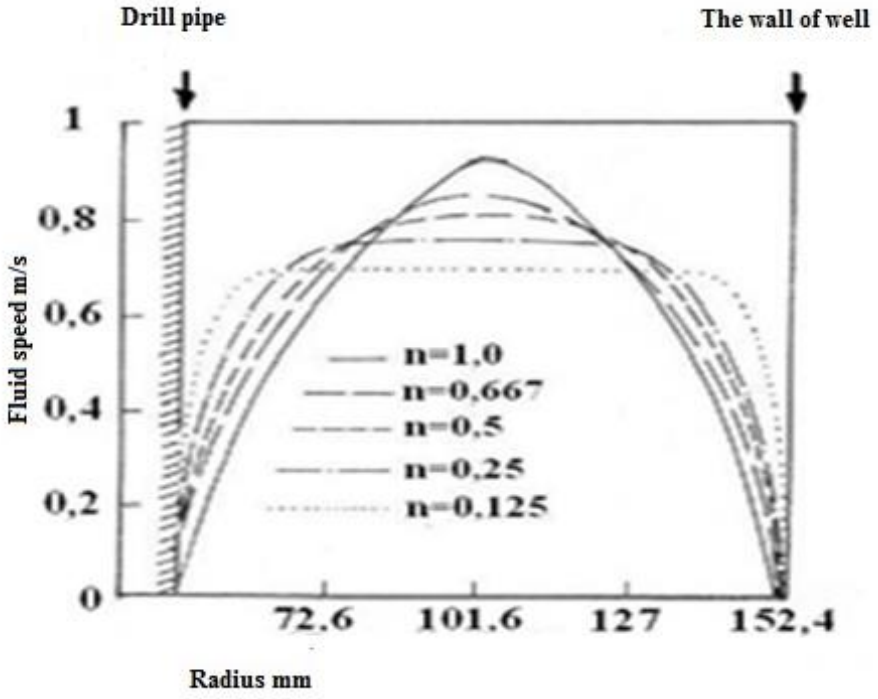


Figure 1Mud speed profiles

Figure 1 shows the profile of the mud speed depending on the "n" factor. In Bingham's fluids, the flattening of the flow profile is achieved by increasing the flow rate of the mud and thereby reducing the relationship between the plastic viscosity and the flow boundary, thus giving lower "n".In the case of extended channel or winding channel "n" it should be lower (0.4-0.6).

3. MUD MOBILITY

The quality of mud is designed to:

- clean the well from the material wound when installing casing,
- check for any fluid flow into the well
- initiated by a gel-like or dehydrated mud that was left in the extensions between the casing and the borehole wall.

The ability of fluid to develop a gel in inaction is called thixotropy. It is considered that fluid is thixotropic under the following conditions, when it can:

- to develop a gel at inaction after mixing and
- to take advantage of the prevailing properties after starting

The basic role of thixotropy or mud properties is that when the interrupt circulation with the development of the gel, it keeps the suppressed particles in a leaking state thus preventing their deposition. The mud mobility is determined by the following relation :

$$\mathrm{Pi} = \frac{1}{V_f \cdot G_{10}\,\mathrm{min}} \qquad (3)$$

Where:
Pi-mud mobility
V_f- the filtrate volume (m^3)
G_{10}min-minimum strength of the gel

As can be seen from equation (3), the mud is more mobile if the volume of the filtrate and the strength of the gel are weighed by the minimum values.

Figure 2 shows the percentage of displaced mud in the function of the proportion of the speed of the cement solution to the annulus and the mud mobility.

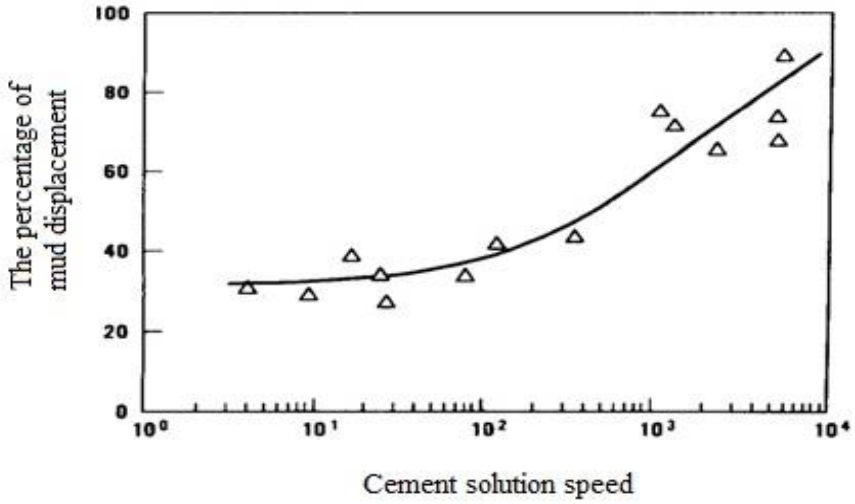


Figure 2 Influence mud mobility and speed of the cement solution on the efficiency of displacing

The percentage of displaced mud can be increased by decreasing the filtration rate of mud and the gel strength.

4. DISPLACING OF MUD IN CEMENTATION OF CASING (CONDITIONING OF MUD IN PREPARING THE WELL)

Under the conditioning of the mud in the preparing of the cement borehole is meant to: reduce the strength of the gel and reduce its plastic viscosity thereby increasing the possibility of fuller displacing mud from the annulus. In most cases, the reduction of the solid plastic viscosity gel follows the reduction of the mud density. Therefore, it is very important that during drilling all the occurrence of mud loss and penetration of layer fluids into the well are monitored and recorded in order to determine the limit values of the hydrostatic pressures.

Mud in the drilling is designed for the drilling process with properties that correspond to the conditions under drilling but the efficiency of displacing the one that requires the need to correct mud properties or improve its rheological properties. Immediately prior to the beginning of the cementation, the two basic characteristics of the mud must be changed: density and rheology. Mud density should be changed to the safety margin, because the density is limited by the pressures of the layers during drilling. The strength of the gel and the plastic viscosity of the mud are the factors that must also be adjusted to increase its mobility.

Modification of mud rheology can be accomplished by the simple addition of water (which reduces the density) or the dispersant for mud. It is necessary to circulate the mud while the rheological properties did not rise in the design range. It is important to circulate one borehole volume (minimum volume) and should be done before extractiondrill pipes. In other words, non-conditioned mud have enough time to gelaurate during the pseudostatic period (tool extraction, EC recording, installation of casing).

Mud circulation is also necessary after the installation of casing. The two basic rules are always present when determining the time necessary for the circulation of the mud before the cementation:

- carry out the circulation in the volume of the annulus for the purpose of checking that there has been no fluid penetration into the annulus,
- carry out the circulation in the volume of the casing column due to the safety that it passes through the passage of cement pulp and plugs

This circulation should take the most time-consuming time until 95% of the measured well volume is circulated. Conditioning of mud at this stage is aimed at :

- clean the well of the coated material, during the installation of casing,
- check the fluid flow in the well,
- homogenizes the mud after treatment in the pools,
- reduces the flow rate and plastic viscosity because most of the muds are thixotropic
- erodes a gelatinous or dehydrated mud that is left in the extensions and in the narrowed (non-centric) areas between the casing and the borehole wall.

5. THE INFLUENCE OF THE NON-CENTERING CASING ON DISPLACING PROCESS

The ideal center of casing is achieved if the outer wall of the casing pipe is far from the wall of the well for 19.05 mm. . In practice, this is very difficult to achieve so that API 10 D specification permits the casing to be away from the wall of the well by a maximum of 12.7 mm so that the minimum value of centricity :

$$CE = \frac{12.7}{19.05} = 67\% \quad (4)$$

Centricity is achieved by the arrangement of centralized with elastic or rigid ribs on the casing. For Fluid Power-Law flow modes where the ratio of the diameter of the borehole and the casing Δl the efficiency of the circulation depends of centricity (CE per API standard must be at least 67%) and from the value of the n- fluid flow index. Typical examples of circulating efficiency for n = 0.5 are shown in Figure (3).

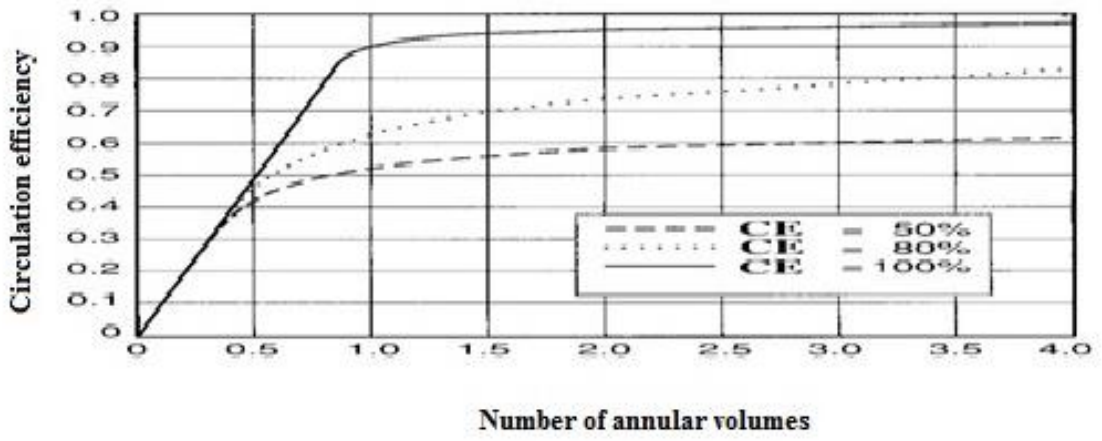


Figure 3 Efficiency of Power-Law Fluid circulation in noncentric interstate for Dc / D = 0.8 and n = 0.5

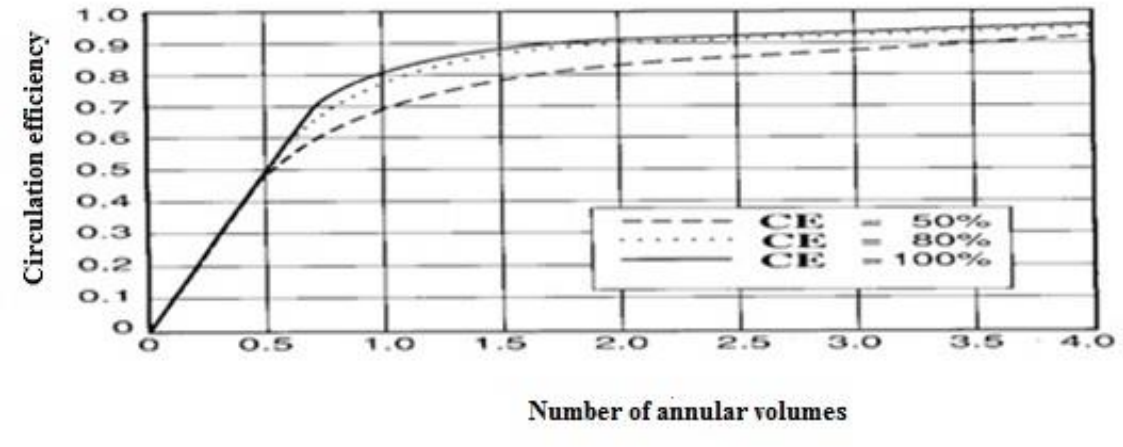


Figure 4 Efficiency of circulation for Bingham plastic fluids in noncentric annulus Dc / Db = 0.8 and $\zeta = 0.174$

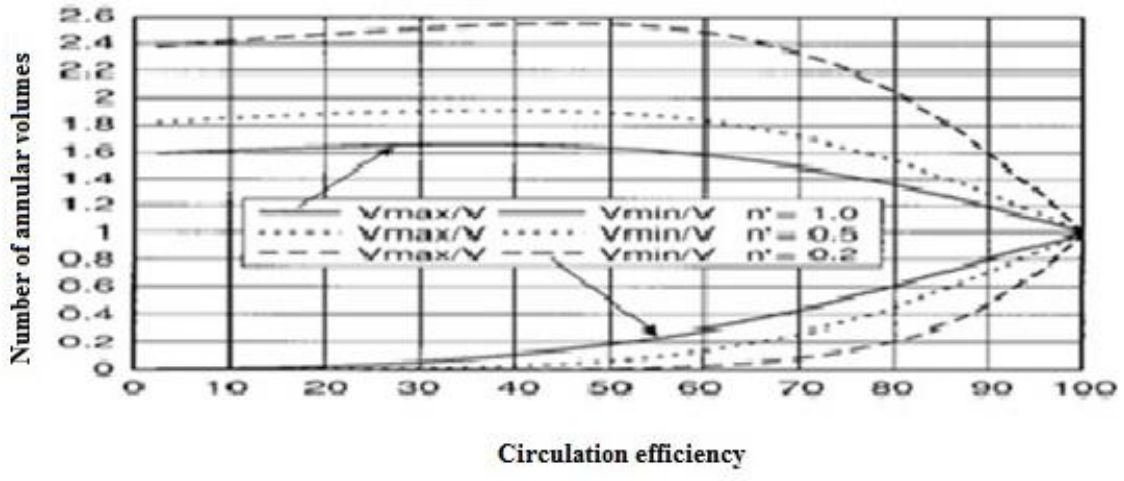


Figure 5 Efficiency of circulation for Newton's fluids in noncentric annulus for Dc / Db = 0.8

6. CONCLUSION

In the complete process of installing casing in the well, the most important is rheology working fluid. Correlation and monitoring of rheological properties such as gel strength, plastic viscosity in this process have the most important influence. Gel and plastic viscosity correction aims at achieving greater mud mobility and bigger displacing of the mud from the annulus. Changing these values of rheological properties is not simply because their change changes the density of the mud which in no case should be smaller than the safety margin. When modifying the mud with water or the necessary dispersers to obtain the above-mentioned characteristics of the mud, these rheological properties have to be established, and this is done by fluid circulation, at least the size of one well volume. In addition to the above-mentioned rheological properties that we can cure to some extent in favor of demanding more efficient and complete displacing there are other influencing parameters on the dipslacing process of mud such as center annulus and mud flow regime. The influence of centricity of the annulus on displacing efficiency has been demonstrated that centricity in the case of greater than 80% achieves very good results of fluid flow efficiency. In cases where the center intensity of the annulus is less than 80% the displacing efficiency is reduced but is in function of time and capacity. The main factor is to have the circulation or annulus displacing completed in one or two cycles or the volume of the borehole, thus the annulus boreholes have a negative impact on the efficiency of the displacing because it would require a longer circulation time and correction of the capacity to be supplied to the borehole.

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SYSTEM ENGINEERING, RISKS, MANAGEMENT AND MODERN TRENDS

DOI: 10.7251/BMC170701167V

INTEGRATED CONTROL AND COAL QUALITY MANAGEMENT MODEL

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ABSTRACT

The problem of coal quality management is now in the focus of research, innovation and engineering mining and energy projects, environmental protection projects, exploitation, energy and ecological efficiency. Besides concentrating on the problem, the paper focuses on the phases of coal quality control, on the assumptions for establishing a system for controlling and managing the quality of coal, and using the model of Mine and Thermal Power Plant Ugljevik, a topology model for controlling and managing the quality of coal has been presented.

Key words**:** coal, control, management, quality, mine and thermal power plant ugljevik

1. INTRODUCTION

In the exploitation of coal and coal-based thermoenergetics, one of the priority problems to which innovation and development researches pay special attention today is about controlling and managing the quality of coal. Why? There is a message which is more frequent today that, because of survival, we need to develop and introduce new technologies of "sustainable progress", including management technologies, which primarily relates to quality management.

Various interpretations of quality management are present, and it is often interpreted as the process of approximating the parameters of the product to the target (desired) parameters. A wider approach goes further and insists on the principle that quality cannot be seen only in the parameters of the final product, but that it is a planned process from the initial to the final stage. In keeping with this approach, a contemporary view on the problem of coal quality management is more attributive and it is not observed only through the prism of natural characteristics of coal - the variability of quality and homogenization before entering the boiler plant.

2. REVIEW AT THE PROBLEM

The management of coal quality has a wider systemic dimension that considers the equilibrium of coal quality correlatively through the process of geological research, through exploitation, transport and deposition, combustion and exploitation of the coal potential, intermittent operation regime and energy efficiency of the thermal power plant, through ecological effects, environmental protection, cost reductions and similar. This approach, Figure 1, requires the establishment of a system for identification, control (in real and extended time) and management (implementation of procedures) at all stages and at all levels of the actual process, including:

- Organization and structure of the system;
- Implemented business, production and development strategy;
- Work environment;
- Mineral-raw material resource(s);
- Production technology;
- Human resource;
- The environment;
- Limitations and ecological conditions.

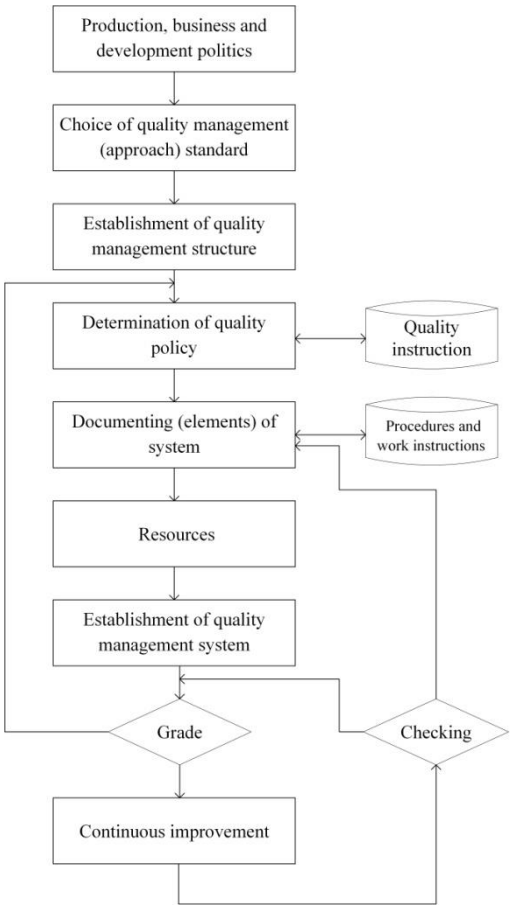


Figure 1. Principal algorithm for quality control

In accordance with this approach, the coal quality management systems applied in practice are based on the phase management principle from coal mining in the mine to incineration in the boiler plant fire boxes and thermoenergetic combustion effects. The logical and physical topology of such coal quality management systems is complex, the goal is focused on the achievement of the given quality of coal with as few differences as possible, and multiple benefits, the most important are:

- Optimization of exploitation (excavation, loading, transport, deposition, etc.) of coal;
- Reduction of coal cleaning costs, if the cleaning is implemented in the technological process;
- Reduced wear on the working wheels of mills;
- Increasing the efficiency of coal combustion;
- Avoiding the glueing of the slag into the fire boxes
- Extension of the exploitation time of the aero mixture channel;
- Maintenance of boiler operating parameters without auxiliary fuel or capacity increase;
- Reducing the load of electro filters;
- Reduction of emissions of polluting components;
- Prevention of technological delays due to inadequate quantities of coal of adequate quality;
- Positive economic effects;
- Positive ecological effects, etc.

For the functioning of the thermal power system of the mine-thermal power plant, a direct, adequate and fast control-reaction to the change in the quality of coal is very important. Real-time measurement is satisfactory when at least four parameters are followed:

1. Quantity of coal mined (flow, weight);
2. Ash content in coal;
3. Moisture content in coal;
4. Heat value of coal.

On coal quality management systems in most of the mining and energy companies in the world, only these four parameters are monitored, there are management systems in which nine parameters are monitored, most prominent in the mines in Germany. In principle, two methods of measuring coal quality are applied:

1. Automatic sampling and analysis of coal, manual manipulation off.
2. Automatic flow radiometric and microwave measurement of coal quality parameters. The measuring device is set, for example, above the belt conveyor, manual manipulation is off.

3. COAL QUALITY CONTROL PHASES

The architecture and topology of the coal quality control system varies depending on the operational configuration of the mining and energy complex, the type and technical capabilities of the equipment on which the components of the system are installed. System architecture is built by hardware components and operational rules of control, decision-making and management, specific to every phase of the technological and technological process.

Control of excavation, loading and transport of coal: It is usual today that coal digging is planned and directed based on the mathematical model of the ledge. Regardless of the applied mathematical approach for forming a model, it cannot have more accuracy than the accuracy of the input data collected by geological explorations. Depending on the lithological structure, the tectonics and other features of the ledge, the volume and the quality of the performed geological explorations, the reliability of the processing of the collected geological data, it happens that the model deviates from the actual state. Therefore, it is nimportant to directly monitor coal quality parameters during excavation. This is feasible thanks to measuring devices - real-time coal quality analyzers, which are mounted on the digging machines. This kind of monitoring of the quality of coal at the site makes it possible to effectively reactivate the radioactive prevention of unwanted excavation of waste and / or coal of poor quality, and timely homogenization planning. The efficiency of coal quality control in this technological phase ensures efficient homogenization and reduction of the costs of coal homogenization.

Transportation of coal: During transportation, coal is mixed, the measurement of coal quality is practiced in two positions. The first one is before coal is deposited at the stockpile of the thermal power plant, and the second is after the takeover of coal from the stockpile.

Coal processing: In mines with coal-cleaning and processing plants, the on-line analyzer measures the quality of inlet and outlet coal from the plant. The input and output measurements allow efficient process management and reduction of coal cleaning costs.

Stockpiling and homogenization of coal: Besides the harmonization of the technological transition between the mining production and coal consumption in the thermal power plant, the coal coal deposition is the third function of coal dumping is homogenization. The goal is coal quality of sustainable quality, as the input resource of the final technological phase - energy conversion. The control and management tasks at the landfill refer to: measurement of coal quality, reciprocal landing navigation, mix control, control of coal deposits at the landfill, and mapping of deposited coal to quality.

Power plant - energy conversion of coal: For the energy conversion and operation of the boiler plant of a thermal power plant, it is preferable to maintain "sustained" coal of uniform quality. The variability in coal quality adversely affects the production capacity and operation of the plant, can lead to technical, technological and production problems in the thermal power plant. In thermoelectric power plants, the quality of coal is controlled before the entry in the bunkers, which enables the conductive filling of the bunkers according to the quality of the coal, and the controlled discharge to achieve uniform quality. In case of a deviation, by adjusting the coal feeder, the boiler operating mode can be achieved, or at the worst, the regime can be maintained at the required level by adding auxiliary fuel.

4. ASSUMPTIONS FOR ESTABLISHING THE CMQC SYSTEM

The efficiency of establishing a system for controlling and managing quality of coal (CMQC) in real time does not depend only on the intent, but also on the fulfillment of the assumptions:

1. Readiness and determination of the management for the introduction of supervisory-management technology;
2. The competence of the collective for the acceptance of supervisory-management technology;
3. Necessary technical, technological and organizational infrastructure of the real system for implementation of supervisory-management technology;
4. Material opportunities;
5. Implementation of the relevant standards, primarily the ISO 73.040: 2015 series relating to coal, including qualitative parameterization, technical and chemical characteristics of coal.

For the effective implementation and application of the system of control and quality management of coal, it is not imperative, but the existence is significantly of integrated computer supported in the supervisory and control system of the mine and the power plant is significant.

6. DEFINING THE TOPOLOGY OF THE CMQC MODEL

Defining the conception of the control and quality management of coal model, in principle, as well as any other activity of definiteness, it assumes the specificity of the real system and compliance with the starting point resulting from the real conditions and constraints.

From the topological structure coal production system and conversion of its energy (mine-power plants), with entities: work environment with coal deposits, coal mining, crushing, transport, disposal, preparing coal and boiler plant of the thermal power plant depends on architecture of the CMQC model, but attention must be paid at the following:

- The selected measuring and communication equipment should be designed for operation in difficult conditions (extreme temperatures, wind, dust, precipitation etc.);
- Control devices and equipment should be adapted for installation on machinery and equipment of the production system;
- The architecture of the CMQC system is rational, functional and modular;
- The architecture of the CMQC system is adaptive to changes in the real system;
- It is recommended to select standardized equipment and software for installation in the CMQC system;
- The architecture of the CMQC system in operational and developmental sense should be flexible;
- It is desirable to integrate the CMQC system into the ambient of the control and management system of the mine and thermal power plant;

Due to the rapid technological progress and the changes it brings, it is understood that the architecture and the selected equipment to establish a system CMQC of the highest technical and technological performance regardless of the higher investment costs, an increase according to the experience is about 25-30%.

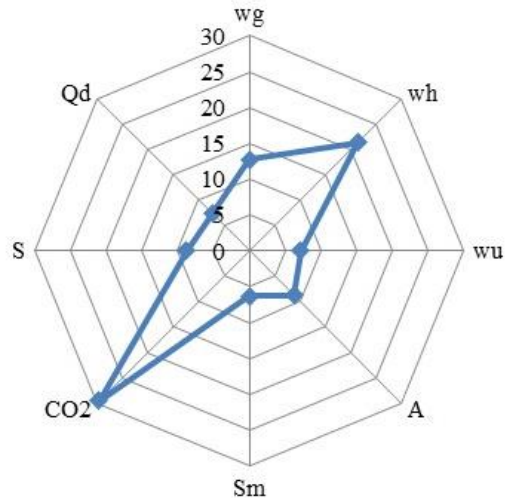
7. EXAMPLE OF THE CMQC MODEL

To complete the thought and message of this text, the conceptual design of the CMQC system of Mine and Thermal Power Ugljevik will serve contrastive analysis of the characteristics of the working environment and the variability of the quality of coal from the open pit mine "Bogutovo village", Figure 2, with experiences of the similar energy complexes in the world, confirmed the technological feasibility and economic efficiency of the establishment of the CMQC system.

Analysis of the functional structure and technological environment energy complex, brings to the conclusion that the system CMQC in Ugljevik should be established on a centralized basis, and that centralization of control and management, in this case there has no alternative solution.

The reason for this is the need for a direct real-time connection between the data acquisition points for coal quality and feedback points on the technological process for the quality of coal at the entrance to the TE. Topologically, the CMQC system should be of a two-layered supervisory-management hierarchy, which ensures adequate functional integrity of the system's entities.

According to conceptual solution the control, that is, the measuring segment, Figure 3, is structured so that it procedurally monitors the quality of the coal from the ledge and excavation at the site of the open pit (position A) by means of crushing, the transport (position B), of the deposit (the position C and D), to the entrance of the coal into the furnace boiler plant (position E), or alternatively, the cleaning of coal, of the excavation on the site of the surface pit (position A), the crushing - that is the entrance to the purification plant (position F), to the output of clean coal (position G) . By this arrangement of measuring points, phase control of coal quality from deposits and geological model forecasts is established, through technological phases, to the final product that enters the process of energy conversion or is delivered to the market. By measuring the quality of coal in seven positions (A, V, C, ..., G), a comprehensive monitoring of the quality parameter is established.



Interpreter:
wg- coarse humidity, wh- internal humidity,
wu - total humidity, A - ash, Sm - combustible matter,
CO2 - carbon dioxide, S - sulfur, Qd- lower heat value.

Figure 2, Comparative review of the percentile parameter deviations of coal quality of the open pit mine "Bogutovo selo"

Position A. Based on the geological model of spatial changes characteristics of coal quality in the ledge and sampling on the floor / front excavation surface mine, comparative data are obtained on which a trustworthy location picture of the quality of coal at a given position in the ledge is formed. As noted, the mathematical model of ledges, regardless of the applied model approach, cannot be more precise than the accuracy of the input data obtained by geological explorations. That is why it is important to measure the parameters of coal quality at position A and to check if the quality of coal is equivalent to the forecast. Measurement data in this position enable timely operational action in real time on the correction of coal excavation (poor quality, waste logging, etc.), timely planning of receipt deposit and mixing, or homogenization of coal. Efficient control at position A is a requirement for successful homogenization and delivery of TE coal, of required quality, cost reduction, etc.

Positions B, C, D, F and G are points of permanent flow of radiometric and microwave measurement in real time: content of ash, moisture, thermal values and delivered quantity of coal.

Parnaby plant. Coal cleaning process in autogenous suspension, it is impossible to effectively operate without control in real-time quality parameters of coal at the entrance and exit of the plant. Radiometric and microwave analyzers, Figure 3, at positions F and G, provide effective monitoring and management of the cleaning process. The goal is coal of the required quality and lower cleaning costs.

Coal stockpile. Features of stockpile are: amortization of transition between the mine and thermal power plant technology, reducing thermal power plant dependency production of coal deliveries from the mines, increasing the operating reliability of thermal power complex as a whole and the fourth feature stems from the required thermal energy conversion for smaller coal quality fluctuations. For the purpose of equalizing the coal mixing quality, real-time measurements of quality parameters are necessary, navigation receipt depositing, control of mixing, stocks tracking and the mapping of the coal deposited according to the quality of the coal. Radiometric and a microwave analyzer, Figure 3, in the position in real time measures the quality of coal to be supplied from the open pit, and devices to positions C and D the coal quality deposit lines that are delivered to the TE. Monitoring the quality of coal at the C and D positions before the bunkers in the thermal power plant, provides the possibility of selective charging and controlled discharge depending on the quality of the coal in bunkers.

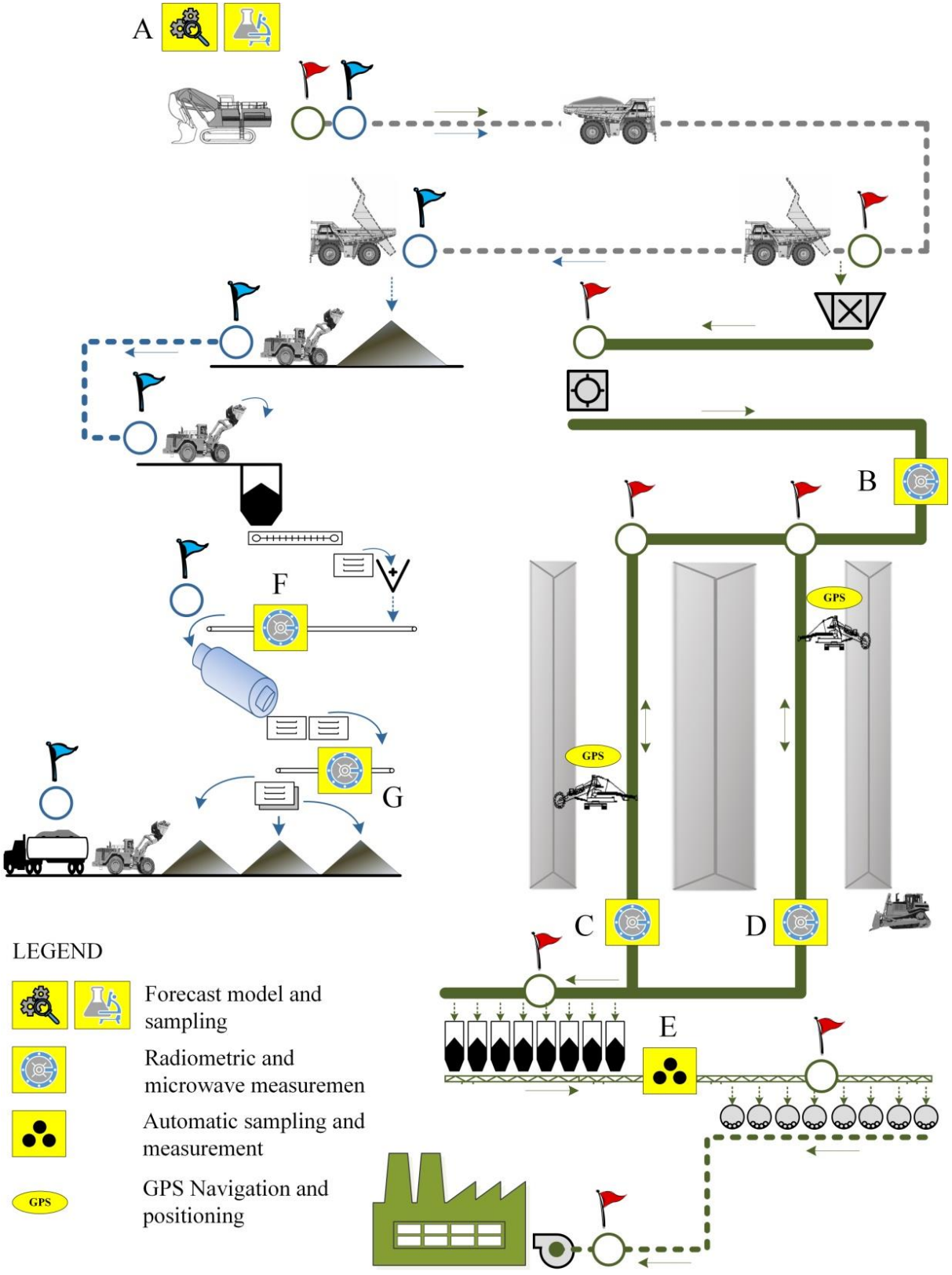


Figure 3, Topology of the conceptual model of the system CMQC Mine and Thermal Power Plant Ugljevik

In order to establish complete monitoring and management of the mining and energy complex, a computer production and business monitoring and control system is needed, with the CMQC sub-system in the structure.

8. CONCLUSION

This textual interpretation has no intention to describe the problem of controlling the coal quality in details, it is unrealistic for the space designed for work. Working implies to the approach to the solving of the task of coal quality management with the conclusion, using the example of Mine and Thermal power plant Ugljevik, that the problem of control and management of coal quality exceeds from the local level to the level of integrated production and business monitoring and management over mining- energy complex as a whole.

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DOI: 10.7251/BMC170701177M

PARCIAL MODEL OF PRODUCTION REQUIREMENTS RELATED TO A DIFFERENT FINAL CONSUMPTION COMPONENT

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ABSTRACT

Public Enterprise Electric Power Industry of Serbia is in the phase of restructuring, financial consolidation. The documents of foreign and domestic consultants insist on a significant increase in productivity and efficiency. The analysis should include economic, social and environmental aspects, as well as national policy and legislation. Such demands are also based on modern proven control mechanisms, where the effects of all of the components of such a large and complex system must be perceived at all times. So far, a number of symposiums have been presented in several papers in which the significance of the application of cross-sectoral models in large and small production systems has been emphasized. In this paper, from the numerous models, only the application of the partial model of production increase in the EPS production system was analyzed.

Key words: cross-sectional models, I-O tables, energy, mines.

1. INTRODUCTION

Inter-sectoral models have long taken a significant place in analyzing the macroeconomics of many countries in the world. The long-term development of these models has significantly expanded the scope of its application. The results of the research show that from the cross-sectoral analysis, numerous and significant knowledge about: direct and return production relations of the entities in the system, two-way interdependencies of the system and environment, ie, On the nature and intensity of the dependence of the system on environmental influences, on the manner of organization and distribution of production tasks, by the entities, their value structure, the volume and structure of consumption, the degree of influence of its changes on the production of the system, the cost structure, the price of production achieved etc.

2. CALCULATE THE INPUT-OUTPUT TABLE

Input-output tables are known as transaction tables. The interdependence that exists between the isolated sectors stems from the fact that every industry branch uses the products of other industries. The resulting products will be used by other industries. Input-output tables can in practice be expressed in monetary values (stable currency) or in any physical unit. The most difficult part of the transaction is the creation of a transaction table and it is the basis for all further calculations. On the basis of the transaction table, the tables of technica coefficients are produced, which normally prescribe the production norms according to the following formula:

$$a_{ij} = x_{ij}/X_j$$

Where they are: the sector and the sector j; X_{ij} - total value of the production of the i-th sector, used to achieve the entire production of the j sector; X_j - value of total production of all production sectors.

	Sectors: $x_1, x_2, \ldots, x_n$	Σ
Sektors: $x_1, x_2, \ldots, x_n$	Reproduction consumption (B)	
		Σ Σ

ΔZ	In	Iz		Op	Σ
Final Consumption (C)					
					Σ

DF
B + C
Σ

	$d_1, d_2, \ldots, d_n$	Σ
AM	Components of a social product(A)	
NE		
SP		
SoP	AM+PL+VP	
SGP		
ΔZ^-		
IX		
DF	DBP + UV	Σ

x_i - Total value of the production of the i-th sector
ΔZ^+ - The remaining stocks of the i-th sector
IN - Investments
IX - Export
PC - Personal consumption
Gc - General consumption
DF - Distributed funds

AM - Amortization
NE - Net earnings
SP - Surplus product
SoP - Social product
SGP - Social gross product
ΔZ^- - Reduction in inventory
IM - Import

Figure 1, Simplified schematic representation of the symmetric input-output table

Technical coefficients of all production sectors form a matrix of technical coefficients $A = [a_{ij}]_{(n,n)}$. Sectors producers are presented in rows, and the sectors are processors by columns of the matrix (table).The next matrix derived from technical coefficients is a matrix containing inverse coefficients.

The matrix [I - A] -1 represents an inverse matrix where: I -is a unit matrix; A - matrix of technical coefficients.

Table 1. Basic input-output table of production sectors RB "Kolubara" and "TENT" (x10^3 RSD)

	Reproduction consumption									Total
	PK I	PK II	PK III	PK IV	KP	TE I	TE II	TE III	TE IV	
PK I	0	0	0	0	0	0,01054	0,17637	0	0	
PK II	0	0	0	0	0	0,03439	0,38059	0	0	
PK III	0	0	0	0	0	0,39622	0,01550	0,37216	0,44867	
PK IV	0	0	0	0	0	0	0	0	0	
KP	0,06349	0,06335	0,05307	0,06501	0,00115	0	0	0,01135	0	
TE I	0,01076	0,01580	0	0	0	0,05067	0	0	0	
TE II	0,01056	0,01540	0	0	0	0	0,02995	0	0	
TEIII	0	0	0,05753	0,05211	0,03405	0	0	0,02798	0	
TE IV	0	0	0	0	0	0	0	0	0,02500	
Total:	0,08481	0,09455	0,11060	0,11712	0,03520	0,49182	0,60241	0,41149	0,47367	
Total material cost (TMC)	0,35499	0,23622	0,14603	0,19151	0,17171	0,69780	0,63380	0,75522	0,56581	3,75309
Amortization (AM)	0,08028	0,26913	0,12142	0,02064	0,10310	0,19454	0,12245	0,11957	0,17342	1,20455
Net earnings (NE)	0,13765	0,05723	0,21808	0,31017	0,29688	0,02454	0,01336	0,05233	0,03902	1,14926
Surplus product (SP)	0,42708	0,43742	0,51447	0,47767	0,42831	0,08312	0,23039	0,07288	0,22175	2,89309
Social product(SoP)	0,64501	0,76378	0,85397	0,80849	0,82829	0,30220	0,36620	0,24478	0,43419	5,24691
Distributed funds(DF)	1	1	1	1	1	1	1	1	1	9

On the basis of the matrix of direct technical coefficients and unit matrix, an inverse matrix of technical coefficients was set:

Table 2. Matrix of direct technical coefficients

$$[I - A]^{-1} = \begin{vmatrix} 1,002 & 0,003 & 0,000 & 0,000 & 0,000 & 0,011 & 0,183 & 0,000 & 0,000 \\ 0,005 & 1,007 & 0,000 & 0,000 & 0,000 & 0,037 & 0,396 & 0,000 & 0,000 \\ 0,006 & 0,008 & 1,023 & 0,021 & 0,013 & 0,427 & 0,020 & 0,392 & 0,471 \\ 0,000 & 0,000 & 0,000 & 1,000 & 0,000 & 0,000 & 0,000 & 0,000 & 0,000 \\ 0,064 & 0,064 & 0,055 & 0,067 & 1,002 & 0,026 & 0,038 & 0,033 & 0,025 \\ 0,011 & 0,017 & 0,000 & 0,000 & 0,000 & 1,054 & 0,009 & 0,000 & 0,000 \\ 0,011 & 0,016 & 0,000 & 0,000 & 0,000 & 0,001 & 1,039 & 0,000 & 0,000 \\ 0,003 & 0,003 & 0,062 & 0,057 & 0,036 & 0,026 & 0,003 & 1,053 & 0,029 \\ 0,000 & 0,000 & 0,000 & 0,000 & 0,000 & 0,000 & 0,000 & 0,000 & 1,026 \\ \textbf{1,102} & \textbf{1,118} & \textbf{1,140} & \textbf{1,145} & \textbf{1,051} & \textbf{1,582} & \textbf{1,688} & \textbf{1,478} & \textbf{1,551} \end{vmatrix} \text{ Ukupno:}$$

3. MODELI PRODUCTION

Production models provide for the determination of a change in production value under the influence of a change in one or all components of final consumption. The basic form of the total production model is the basis from which the model of total production increase can be identified. The determined value of the total production increase is also transferred to imports. Hence the relation of the conditional increase of imports should be added to the production model. Instead of the total production model, it is also possible to form a partial production model, bearing in mind that final consumption can be explained on several co-ordinates. There is also a partial model of production growth resulting from the explanation of the increase in final consumption and one such model has been elaborated in this paper.

4. PARCIAL MODELS OF PRODUCTION INCREASE

In these models, instead of total, the components of final consumption appear, which will lead to changes in production. For the illustration, the component of the final in-use is selected. By decision of the production and technical college of PE "Elektroprivreda Srbije", investment funds are planned for raising the level of technical and organizational efficiency on the surface mines of RB Kolubara and thermal power plants "Nikola Tesla". During the planned year they were realized in the following values (Table 3).

Table 3. Comparative overview of planned and realized investments and planned depreciation in the sectors RB "Kolubara" and "TENT"(x10^6RSD)

Sector name	Planned amortization	Planned investments	Realized Investments	Difference (2-3)		Difference (3-4)	
1	2	3	4	5		6	
PK I	268	80	20	+	188	+	60
PK II	2.000	2.950	500	-	950	+	2.450
PK III	1.200	4.660	450	-	3.460	+	4.210
PK IV	17	275	180	-	258	+	95
Rehabilitated equipment in mines	-	600	220	-	600	+	380
Total mines	3.485	7.965	1.150	-	4.480	+	6.815
KP	358	365	170	-	7	+	195
TE I	3.616	3.671	500	-	55	+	3.171
TE II	2.126	1.695	220	+	431	+	1.475
TE III	550	128	9	+	422	+	119
TE IV	208	0	0	+	208		0
Total thermal power plants	6.500	5.494	729	+	1.006	+	4.765
TOTAL	10.343	14.424	2.269	-	6.093	+	12.155

The table shows the discrepancy between planned depreciation and planned investments, as well as between planned and realized investments. These differences are more evident in RB Kolubara than TENT.

By looking at the table, the following questions can be answered:

– Why are investments planned in a larger amount than is provided by the aggregate depreciation and accumulation value? This is the case with most production units.
– Is the planning of investments submerged, and if not, how will their non-realization be reflected in the future functioning of this strategically important thermal power system?
– Are investment activities carried out on time, bearing in mind the procedure in the implementation of the Law on Public Procurement.

What is particularly important for managers is to look at returnees that may result from untimely investment, especially when considering the coal industry and the thermal power sector as a whole in the process of restructuring, and that the banks of the current main investors have withdrawn from further investing everything It's on our coals.

From the explained increase in final consumption, the partial model of production increase for investments will be analyzed.

$$[I - A] - {}^{1}\Delta J = \Delta X^{J}(1)$$

Where are they:I - unit matrices; A - matrix of technical coefficients; ΔJ - additional increase in investment spending; ΔX^{J} - conditioned production value.

The following values of the realized increase in investment spending are given by sectors:

Table 4. Production growth by investing additional investments ($\times 10^3$ RSD)

		Reprodukciona potrošnja										Investments ΔJ	Production growth ΔX^j
		PK I	PK II	PK III	PK IV	KP	TE I	TE II	TE III	TE IV	Total		
PK I		0	0	0	0	0	7.889	41.611	0	0	49.500	20.000	69.500
PK II		0	0	0	0	0	25.741	90.879	0	0	116.620	500.000	616.620
PK III		0	0	0	0	0	296.001	3.681	27.606	0	327.288	450.000	777.288
PK IV		0	0	0	0	0	0	0	0	0	0	180.000	180.000
KP		4.413	39.063	40.960	11.702	307	0	0	842	0	97.287	170.000	267.287
TE I		748	9.743	0	0	0	38.009	0	0	0	48.500	700.000	748.500
TE II		734	9.653	0	0	0	0	7.113	0	0	17.500	220.000	237.500
TE III		0	0	44.621	9.380	9.101	0	0	2.075	0	65.177	9.000	74.177
TE IV		0	0	0	0	0	0	0	0	0	0	0	0
Total		5.895	58.459	85.581	21.082	9.408	367.640	143.284	30.523	0	721.872	2.249.000	2.970.872
Total material cost (TMC)		24.672	145.658	113.508	34.473	45.896	522.304	150.527	56.020	0	1.093.058		
Druš. proiz.	AM	5.579	165.951	94.378	3.715	27.557	145.613	29.082	8.869	0	480.744		
	NE	9.567	35.289	169.511	55.831	79.352	18.368	3.173	3.882	0	374.974		
	SP	29.682	269.722	399.891	85.981	114.482	62.215	54.718	5.406	0	1.022.097		
	SoP	44.828	470.962	663.780	145.527	221.391	226.196	86.973	18.157	0	1.877.814		
Production		69.500	616.620	777.288	180.000	267.287	748.500	237.500	74.177	0	2.970.872		
Import		-	-	-	-	-	--	-	-	-	--		
TOTAL:		69.500	616.620	777.288	180.000	267.287	748.500	237.500	74.177	0	2.970.872		

$$\Delta J = \begin{vmatrix} \Delta J_{PK\ I} \\ \Delta J_{PK\ II} \\ \Delta J_{PK\ III} \\ \Delta J_{PK\ IV} \\ \Delta J_{KP} \\ \Delta J_{TE\ I} \\ \Delta J_{TE\ II} \\ \Delta J_{TE\ III} \\ \Delta J_{TE\ IV} \end{vmatrix} = \begin{vmatrix} 20.000 \\ 500.000 \\ 450.000 \\ 180.000 \\ 170.000 \\ 700.000 \\ 220.000 \\ 9.000 \\ 0 \end{vmatrix}$$

From Table 4, it can be noticed that additional investment investments were higher in the PK II segments of coal and TE and electricity production. Although PK III had smaller additional investment from PK II, it realized a higher production increase. The largest investment was made by the TE I, and it achieved the highest increase in production.

5. CONCLUSION

The very very modest application of cross-sectoral models in large and small business systems shows adaptability, efficiency in resolving the level of problem differences. The great advantage of these model approaches is in the examination and diagnosis of the state of cross-sectoral relationships, dependencies, disturbances, and the like. In complex mining and thermal power systems.

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DOI: 10.7251/BMC170701183P

MODEL OF MULTICRITERIUM RANGE OF COAL DEPOSITS SUITABLE FOR UNDERGROUND GASIFICATION

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ABSTRACT

The underground gasification of coal is a modern "clean" technology for obtaining quality energy - SYNGAS and has not been explored in these areas at all. In the Republic of Macedonia, the method of underground gasification of coal is occasionally actualized in the form of various studies, but in the world is quite present. This technology has the potential to reduce greenhouse gas emissions in the process of enrichment of coal deposits. Any investment and attention paid to its development is welcome if we think for a better energy future.

Key words: coal, underground coal gasification, multicriteria range

1. INTRODUCTION

The underground coal gasification is one of the main technologies that enables the introduction and implementation of unconventional coal mining methods in order to minimize emissions of harmful gases. This technology provides opportunities for further ecological exploitation of the coal.

Namely, for the selected three variant solutions, 4 main criteria are selected, as well as the other six additional criteria. The selection of the criteria is made on the basis of contemporary research in this field.

Four main criteria are: minimum depth of the coal seam (is not greater than 12 meters), minimum power layer (is not less than 2 meters), ash content (not exceeding 35 %) and sulphur content (up to 5%). These four criteria give an opportunity to define a model that determines how many of the total number of investigative holes accomplish these conditions.The analysis was made for 5 values of the ash content of 15, 10, 25, 30 and 35%. These are 5 subversion solutions.

In this way, for each of the 3 variant solutions, output tables are given the percentage participation of the fulfilled 4, 3, 2 and 1 condition.

2. SELECTION AND IDENTIFICATION OF CRITERIA FOR MULTICRITERIALRANGE

Solving of the multi-criteria problem means choosing the "best" alternative from a variety of available alternatives, where the notion of "best" decision-maker can be interpreted as "preferred". The criteria that have the greatest influence on the model resolution have been identified.

The following 7 criteria have been selected are:

Criterion 1: Coefficient of utilization of coal reserves

Criterion 2: angle downs

Criterion 3: Structured – tectonics characteristics

Criterion 4: Hydro - geological features

Criterion 5: Configuration field

Criterion 6: Charge for exploitation by conventional methods

Criterion 7: Ability to use conventional methods

Multicriterial model is defined by descriptive marks required is their transformation into numerical values.For this purpose the simplest is using a linear scale transformation.

0	1	2	3	4	5	6	7	8	9	10
	distinctively low		low		medium		high		distinctively high	
worst		many low		secondary lower		secondary higher		very high		best

Figure. 1. Linear transformations for the quality atributs

After the transformation of the attributes qualities, the input multi-criteria model (table) is obtained. Resolving multicriterial model was made by the PROMETHEE II method. There are 6 types of common criteria: Ordinary criterion, Quasi criterion, Criteria with linear preference, Criterion level, Criteria with linear preference and indifference area, Gauss's criterion

3. MODEL OF MULTICRITERIUM RANGE

The entry model prepared for solving the PROMETHEE II multi-criteria method is given in the following table.

Table 1.

No.	Alternatives	Rate of efiiciency of coal reserves with UCG K1	Angle K2	Structural and tectonics characteristics K3	Hydro - geological charactristics K4	Landscape aproach K5	Economical value K6	Application of conventional method of exploitation K7
		%	°deg				EUR/t	
1	Alternative 1	e1	a1	c1	h1	l1	p1	m1
2	Alternative 2	e2	a2	c2	h2	l2	p2	m2
3	Alternative 3	e3	a3	c3	h3	l3	p3	m3
MIN/MAX		MAX	MAX	MAX	MAX	MAX	MIN	MAX
W	%	50	10	10	5	5	10	10
		0.5	0.1	0.1	0.05	0.05	0.1	0.1

Resolving multicriteria model was made by the PROMETHEE II method and was used academic software version VISUAL PROMETHEE. Below are the results of the resolution of the multi-criteria model:

Statistical indicators

The following table provides basic statistical indicators for the input multi-criteria model.

Table 2. Statistical indicators

Feature	K1	K2	K3	K4	K5	K6	K7
MIN	e_2	a_3	c_2	h_2	l_1	p_2	m_1
MAX	e_3	a_1	c_1	h_1	l_2	p_3	m_2
Average value	Av_1	Av_2	Av_3	Av_4	Av_5	Av_6	Av_7
Standard deviation	Sd_1	Sd_2	Sd_3	Sd_4	Sd_5	Sd_6	Sd_7

Characteristics of the criterion functions

Table 3. Caracteristics of criterium function

Feature	K1	K2	K3	K4	K5	K6	K7
Min/Max	max	max	max	max	max	min	max
W	0.50	0.10	0.10	0.05	0.05	0.10	0.10
Type function	Ordinary	Level	Level	Ordinary	Level	Ordinary	Level
Indifirenca	-	1.00	1.00	-	2.00	-	1.00
Preferences	-	3.00	3.00	-	5.00	-	3.00

Net flow

The following are the values of the net flow over the PROMETHEE II method.

Table 4. Total values according PROMETHEE II

Alternative	Phi	Phi+	Phi-
A_3	0.3375	0.5625	0.2250
A_1	0.1250	0.5250	0.4000
A_2	-0.4625	0.1875	0.6500

Diagrams from the determined model according to the PROMETHEE method

After solving the multi-criteria model, the diagrams can be drawn according to the PROMETHEE method.

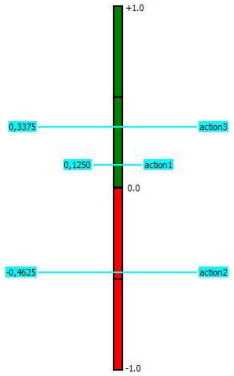


Figure.2. Diagram according PROMETHEE II method

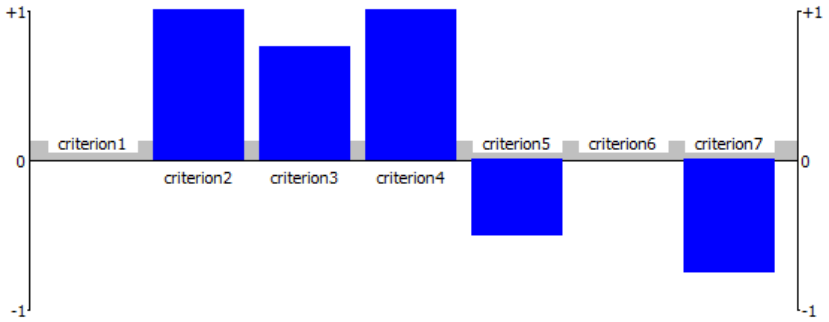


Figure. 3. Diagram of active criteriums for alternative 1

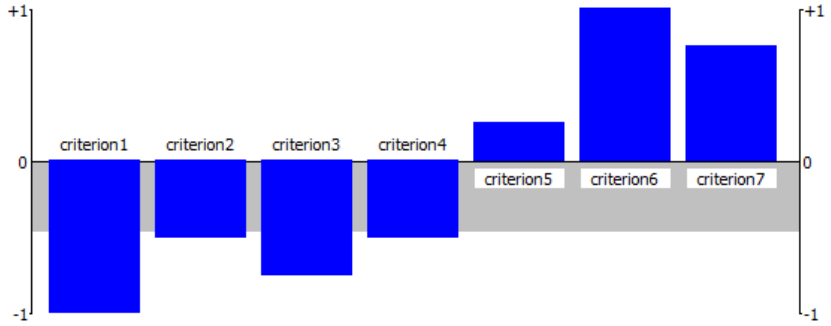


Figure. 4. Diagram of active criteriums for alternative 2

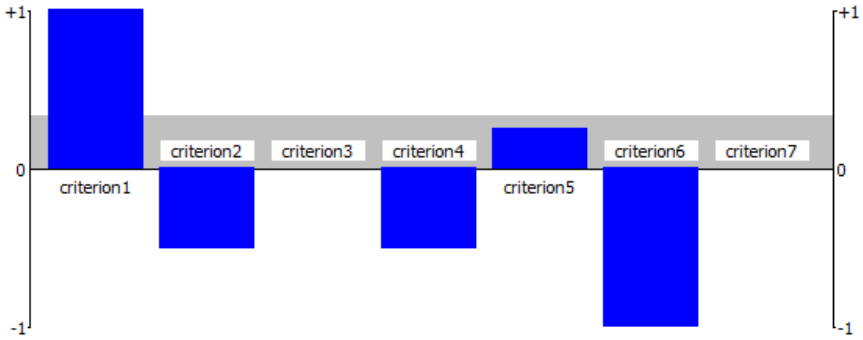


Figure. 5. Diagram of active criteriums for alternative 3

The next diagram is given PROMETHEE - GAIA analysis. GAIA (geometrical analysis for interactive aid) is an interactive geometric view of the solution from the PROMETHEE method.

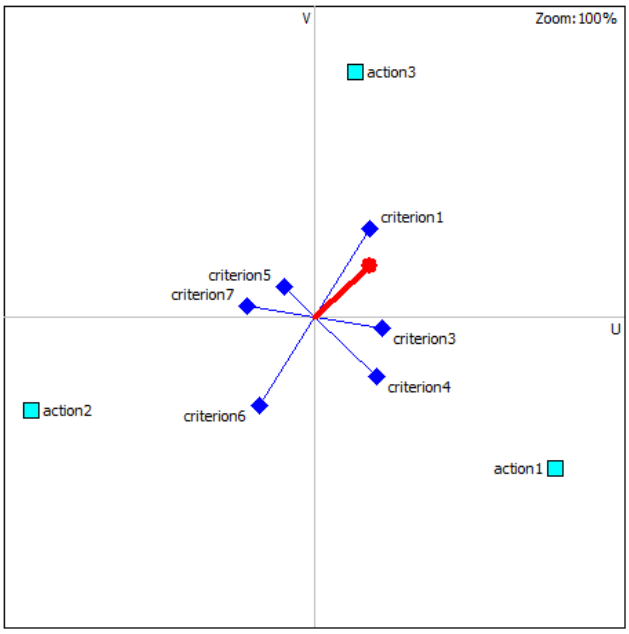


Figure.6. GAIA diagram

According to the PROMETHEE II method, it can be concluded that the first ranked alternative is Alternative A_3, which has a value for the net flow of 0.3375 and has the highest priority for the application of underground gasification as a method that is environmentally acceptable. Next to the rank list is the alternative A_1 and without the priority with the negative value of the net flow is the alternative A_2. The same is unsuitable for the application of the underground gasification technology of coal.

1. CONCLUSION

Coal is still one of the major energy sources in many countries in the world. According to the predictions of many well-known scientific institutes, it will be dominant in the next 20-30 years. The main problem is the emission of harmful gases during its exploitation, but above all in its combustion. The main directions in the exploration of coal are to find technology for their exploitation, but also their transformation into a gas whose combustion would be environmentally friendly, that is, it would aim for the so-calledClean technology with zero emission of harmful gases.

The Republic of Macedonia is a signatory to the Energy Community Treaty, where according to this agreement, special emphasis is put on environmental protection through the introduction of new modern technologies for exploitation of coal with minimal emissions of harmful gases.

It is necessary to make a pilot probationary exploitation for experimentally verifying the possibility of applying the technology for Underground coal gasification.

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DOI: 10.7251/BMC170701189E

IMPLICIT 3D SOLID MODELLING WITH SUPPORT MACHINES

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ABSTRACT

Solid modelling in mineral resource estimation is important phase since block model is built with respected to the orebody. If the solid model of the ore deposit is not consistent with approximate real case, error will accumulate progressively through the estimation steps. Interpretation in classical approach, where sections are taken through drillholes and polygonized with geological and mining knowledge, is very critical and the process is always time consuming. In this study, support vector machine (SVM) algorithm compiled by authors is offered as a solution for 3D solid modelling. The proposed implicit method is based on Gaussian radial basis functions and set of northing sections are classified as ore zone and no-ore zone, respectively. The input variables are indicator values derived from boreholes of correspondant sections and the output, 3D solid, is created by combining all classified sections. The SVM is fast when compared to the classical slice sectioning and 3D solid model reveals consistent result.

Key words: Orebody, Support vector machines, 3D solid model.

1. INTRODUCTION

It is an undeniable fact that mining plays an vital role in the development of countries through the rational use of science and technology. When countries with a high added value based on mining are examined, it appears that almost all of the mining activities are done within the framework of a resource-reserve coding standards. These codes are listed such as NI43-101/CIM [1], JORC [2], PERC [3], SAMREC [4], which are collected under the umbrella of the CRIRSCO (Combined Reserves International Reporting Standards Committee). In Turkey, nowadays content and the framework of the repoting standard UMREK (National Resources and Reserves Reporting Comittee) commenced.

Report standards are classified according to a certain level of confidence in the presence of underlying orebody. Although the level of confidence is defined on the basis of geostatistical estimates or simulations at the centroids of the block model, it is directly dependent on how closely the orebody is modeled with respect to the ground-truth.For example, a geological section through deposit where no ore is present will lead to the formation of a block pattern, in other words a non-existent ore presence. Thus, more precise ore-bed rock discrimination is achieved, the less systematical error will appear through the steps of resource estimations.

Today, with the development of mining software, the sections taken by hand are digitally transferred to the computer environment and offer the user instant 3-dimensional visualization. Each section drawn by hand is actually an interpolation. The user creates a polygon by hand, taking into account the impact distance of the borehole on the section taken, and a series of polygons then forms the volume of orebody. Every error made in this sectioning method is systematically carried to every stage extending to the end of the mineral resource estimate. Since each user's geologic interpretation will be different from each other, there is no criterion of how much error is made in the sectioning process.

Orebodies are formed by the combination of various geological and atmospheric laws and appropriate conditions in the earth's crust. Naturally, the orebody is systematically differentiated from other rocks and is located in a certain pattern. Support vector machines method, which has an important place among the machine learning methods, has been used to reveal this pattern in the study. Through this method, the data patterns arising from the analysis of the existing data are categorized and classified. In this respect, the grades below a certain threshold and not to be used in mine planning are taken into consideration during the sectioning process. An algorithm that classifies the boreholes based on cross-sections according to their threshold grade is compiled with the MATLAB program. The proposed implicit method is based on Gaussian radial basis functions and it is applied to an epithermal gold deposit in order to build 3D solid model of the mineralization. Mineralized zones in each section is classified with SVM based on indicator variables of drill-holes falling in corresponding sections and finally these sections are combined in order to create 3D solid model. In addition, generalized accuracy of the classification is established by cross validation by test and validation data.

2. METHODOLOGY

Support vector machines are set of related supervised learning methods used for classification and regression and they are regarded as generalized linear classifiers. SVMis based on the quadratic optimization problem, and does not require any knowledge of the distribution function for the data. In SVM classification, data optimally separated into the two categories of n-dimensional hyperplane (hyperplane) form. In the case of binary classification, the aim is to find decision boundary that partitions the underlying vector space into two sets, one for each class. The formula for the output of a linear SVM is given as:

$$f(x) = w^T x + b$$

where w is the normal vector to the hyperplane, x is the input vector, and b is the margin offset. The separating hyperplane is the plane $f(x) = 0$. The nearest points lie on the planes $f(x) = \mp 1$.

The margin m is defined by the distance of the hyperplane to the nearest of the positive and negative examples. The aim of the SVM is to obtain maximum margin that is $\min_{w,b} m$ subjected to $y_i(w^T x_i + b) \geq 1, \forall i$ where wherex_i is the ith training example, and y_i is the correct output of the SVM for the ith training example. The value y_i is +1 for the positive examples in a class and –1 for the negative examples.

Finding maximum margin is defined as an quadtratic program (QP) problem in which the objective function is dependent on Lagrangian multipliers namely, α_i. The dual formulation yields:

$$\min_{w,b} m = \frac{1}{2}\sum_{i=1}^{N}\sum_{j=1}^{N} y_i y_i (x_i \cdot x_j)\alpha_i \alpha_j - \sum_{i=1}^{N} \alpha_i$$

under constraints, $\alpha_i \geq 0, \forall i$ and $\sum_{i=1}^{N} y_i \alpha_i = 0$

where N is the number of training samples,

Cortes and Vapnik [5] introduced a modification to the original optimization statement which allows the failure of an example to reach the correct margin. But it is penalized with slack variable ξ_i, wide margin trade-off parameter C. Then, the dual formulation is modified under constraint $0 \leq \alpha_i \leq C$.

Non-linear SVM classifiers are generalized as follows:

$$f(x) = \sum_{j=i}^{N} y_i \alpha_j K(x_j, x) + b$$

where $K(x_j, x)$ is kernel function between input vector x and training vector x_j.

The nonlinear case slightly alters the quadratic form but α_i are still computed by QP such as,

$$\min_{\alpha} m = \frac{1}{2}\sum_{i=1}^{N}\sum_{j=1}^{N} y_i y_i K(x_j, x_i)\alpha_i \alpha_j - \sum_{i=1}^{N} \alpha_i$$

with constraints $0 \leq \alpha_i \leq C$and$\sum_{i=1}^{N} y_i \alpha_i = 0$

There are numerous kernel functions available for SVM, such as Gaussian, polynomials, and neural networks. In this study, Gaussian Radial Basis Function Kernel (GrbfK) $K(x_i, x_j) \equiv e^{-\gamma \|x_i - x_j\|^2}$ is applied.

The QP problem is solved when three conditions are met for all i, $\alpha_i = 0$, $0 \leq \alpha_i \leq C$, and $\alpha_i = C$.

There are many different kernel functions that DVM uses in nonlinear distinction situations. Frequently used functions are Gauss, polynomials and neural networks.Quadratic problem programs (QP) are often processes that result in complex equations and long solution times. Sequential Minimal Optimization (SMO) method which partitions large QP problems into small sections is used in the study and detail information can be referred in Platt [6].

The method of binary classification is adhered to several classifications by some modifications. Two main methods for multiple classification in the literature have been proposed. These are One-vs-all (OVA) [7] and One-vs-one (OVO) [8].

3. CASE STUDY

In the field, 22 sections are created with bandwidth of 50 m and each section is classified by means of high grade ore, mid-grade ore and waste (red, blue, and black, respectively) (Figure 1). Soil samples and assay values from boreholes which fall into corresponding bandwidth are used as an input in each slice and total data count is 23150. Results of the each slice then imported into block model centroids in order to have better visualization of the orebody (Figure 2). In this study, Gaussian kernel function is used which is defined by $K(x_i, x_j) \equiv e^{-\gamma\|x_i - x_j\|^2}$. 10-fold cross-validation was applied for training and validation steps.In the SVM algorithm, the Gaussian kernel function parameter pair (C, γ) automatically selected for the generalized accuracy. The parameter pair which gives the highest accuracy value according to the learning and test data is used as the prediction parameter. In the case of more than one parameter pair giving equal accuracy value, the pair that gets faster solution time is selected.

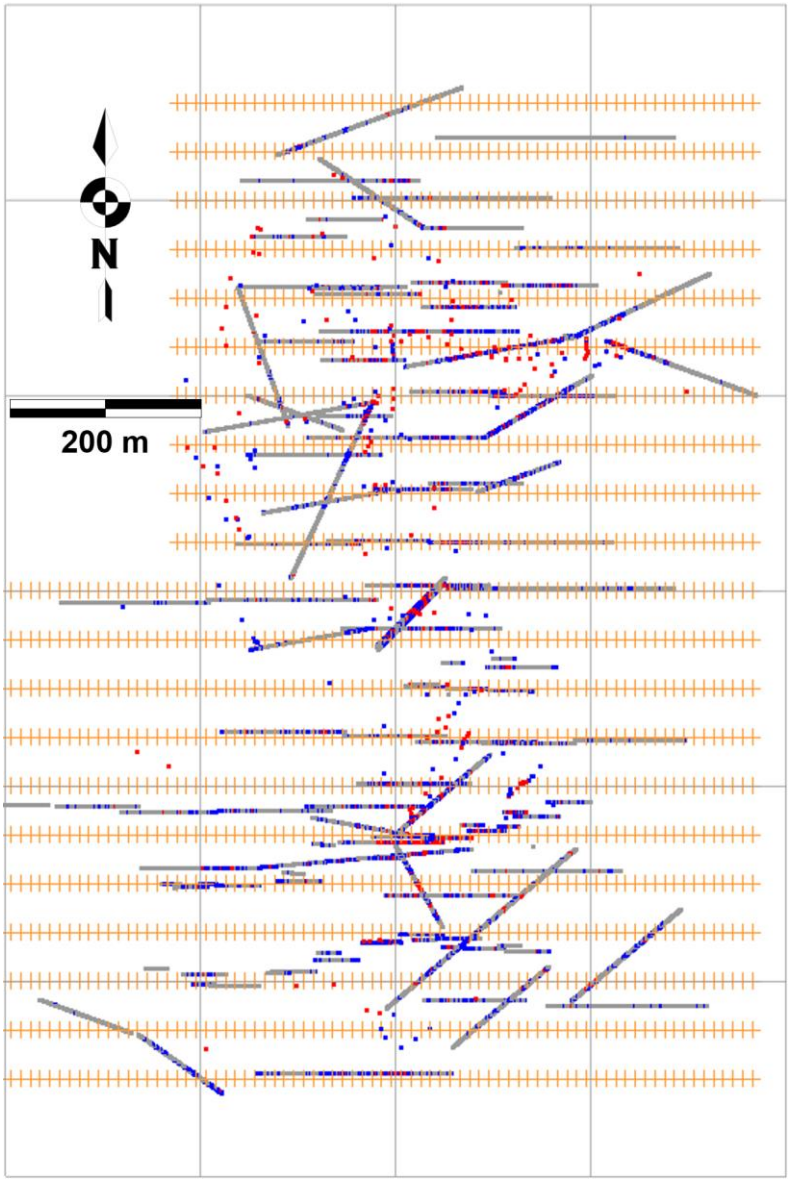


Figure 1 Sections, soil, and borehole data.

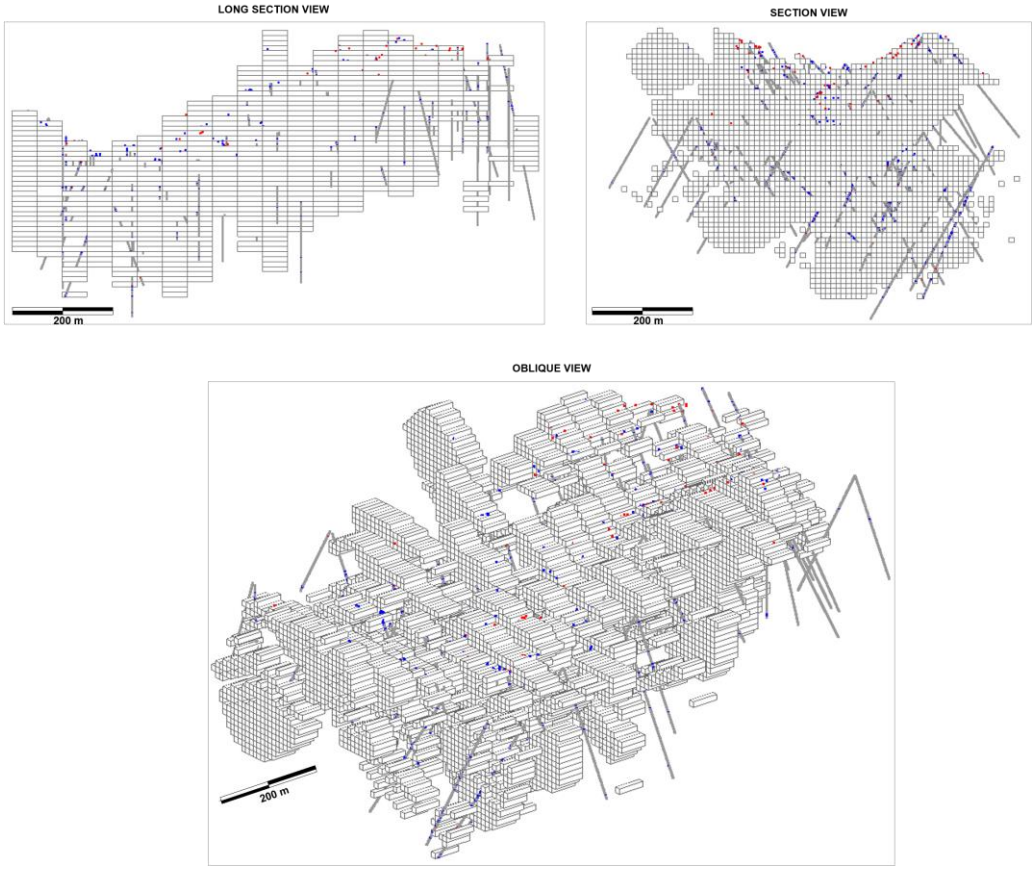


Figure 2 Orebody geometry generated from

The blocks categorized according to the threshold grade output by the algorithm are given in Figure 3. When the results are evaluated, it is seen that the results are consistent with the borehole data and threshold values are also honored.

Blocks estimated using SVM are indicative for the geological section and can be used as templates at the final stage of the final determination of the ore boundaries. This template is transferred to the mining software and the section is digitized by hand drawing in this environment. The final result is shown in Figure 4.

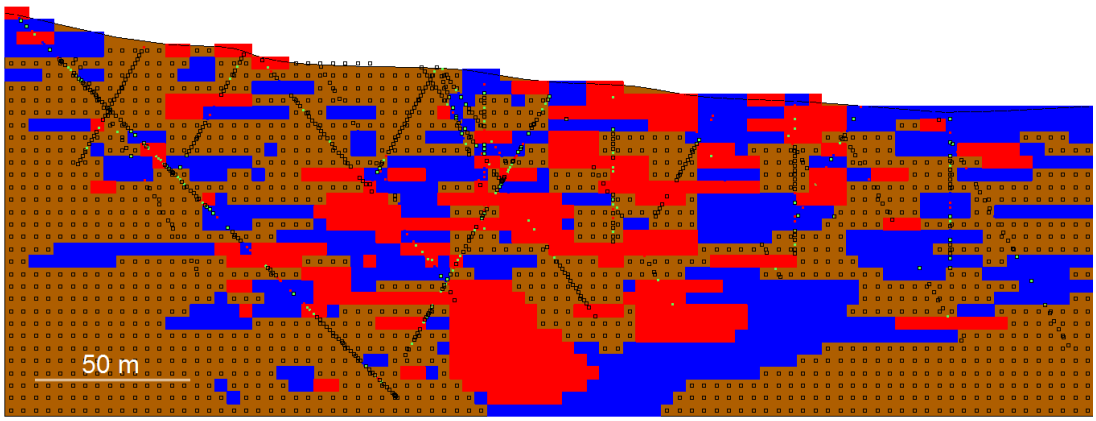


Figure 3 One example support vector machine section. Orebody section (blue: low grade, red: high grade)

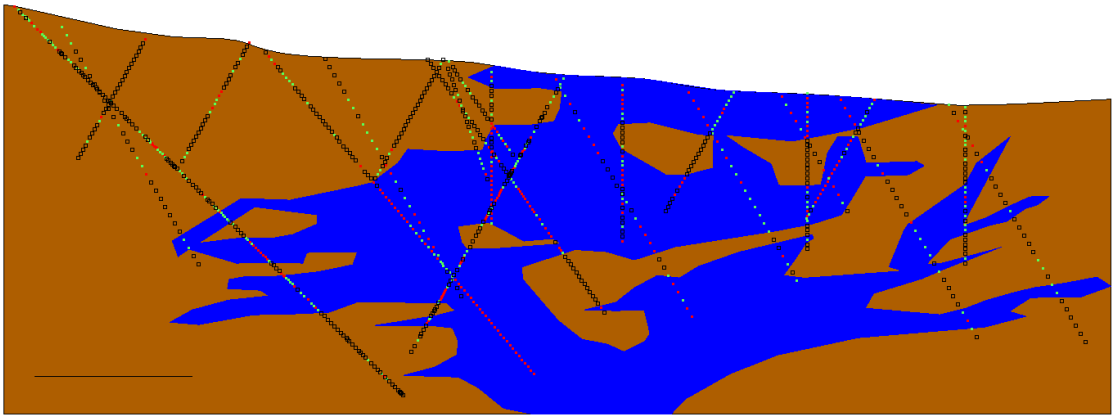


Figure 4Interpreted digitized section of an example section

4. CONCLUSION

In this study, a cross-sectional based solid modelling method is proposed in order to guide the user in decision making. The concept of ore-waste discrimination plays an important role in the mineral resource estimation. Mine planning may be the subject of miscalculation when it is made for mineral assets under threshold grade. The algorithm, which is capable of forming a faster section in the case of different threshold grade values, provides a great convenience especially for ore deposits which are difficult to model. However, this method should be used only as a guide in decision making.

The generalized accuracy of the SVM algorithm presented in the study as a result of cross validation can also be considered as an important criterion for the accuracy of the solid model.

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DOI: 10.7251/BMC170701195A

ESTIMATION OF IRON CONTENT OF A DEPOSIT USING DEEP NEURAL NETWORKS

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ABSTRACT

Ore resource estimation is one of the primary step of pre-feasibility/feasibility studies of mine projects. Feasibility of the projects directly related with the spatial distribution of the target content which has economic interest. For this reason, spatial distribution of the ore content has to be estimated. This paper aims to estimate the iron content of an iron deposit using deep neural networks. 42 drillholes are drilled at the deposit. In order to perform estimations solid model (wireframe model) of the deposit is generated using sections method and block model of the deposit is also generated. Using Matlab® a deep neural network is trained for drill hole data. This trained model is used to estimate the iron content of the block model. Results of the estimates are compared with raw data and trend analysis is performed to validate the estimations. Results shows that estimations are unbiased and can be used for further for mine planning, pre-feasibility and feasibility studies.

Key words**:** Deep neural networks, Iron ore

1. INTRODUCTION

The estimation of the variability of the target variable of the ore resource plays vital role for mine planning and other sequential steps at the life of a mine. At today's technology and economic conditions only limited number of data can be collected from a prospective ore resource. Due to the limited availability of the data, variability of target variable where no data is collected has to be estimated.

In ore resource estimation generally classical geostatistical methods are used such as kriging methods. Unfortunately, in order to apply kriging methods, generally cumbersome steps have to be followed such as trend removal, data transformation, variogram modelling and generation of kriging plans. Also, stationarity is the main assumption of the kriging which is hard to exists in real world. In order to by-pass these laborious steps machine learning algorithms (ML) can be used.

ML algorithms are alternative geostatistical methods in ore resource estimation to traditional methods. ML is sub-branch of artificial intelligence and can be defined as family of

algorithms that estimates the outcomes of the specific events without being explicitly programmed. Decision trees, Support vector machines, Ensemble methods, Neural networks are just some examples of the ML [1-4]. Due to the simplicity of the algorithms they are used in various branches of engineering such as electrical, automotive, computer and various other fields [5-7]. Mining is not-exception to these areas of engineering. [8-10]. Specific to ore resource estimation ML algorithms are used in various researches [11-14].

In this study, Deep Neural Networks (DNN)are used to estimate the iron grade distribution of an iron ore deposit. For estimation purpose, firstly, raw data is composited in to equal lengths. Ore body of the iron deposit is modelled using wire-frame modelling technique. Later block model of the ore body is generated. In order to estimate the iron grade distribution on block model, DNN model is trained and finally using this trained model iron grade distribution is estimated.

2. RAW DATA AND COMPOSITING

47 drillholesare drilled at the deposit. All of the drillings are intersected with the iron ore. Due to proprietary of data, location of the deposit is not exposed. Raw data is composited to equal length samples and length weighted averages are calculated. Composites are shown in Fig. 1.

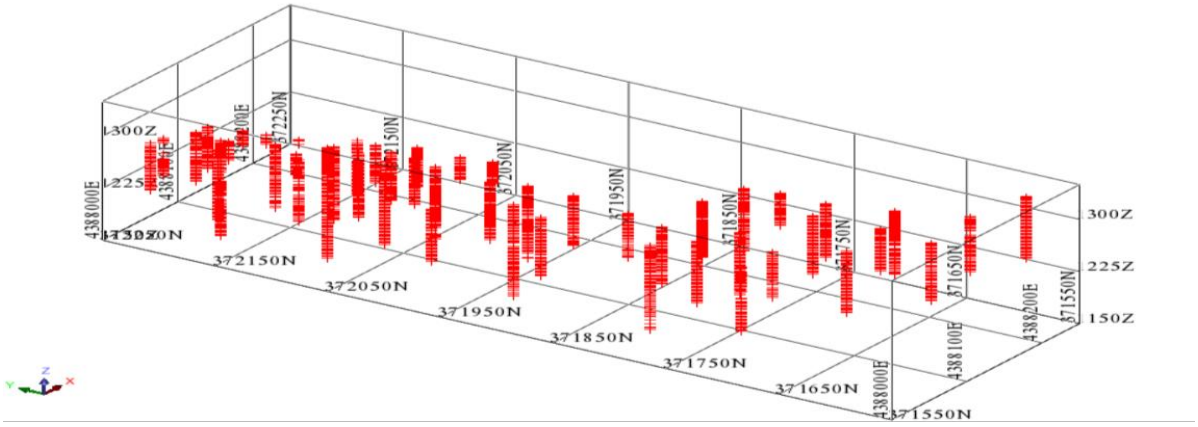


Figure 1. Oblique view of the composites

Summary statistics of the composites are given in Table 1.

Table 1. Summary statistics of the composites.

No of Data	Minimum (%)	Maximum (%)	Average (%)	Variance	Skewness
2701	0.01	56.28	17	200	1.16

3. ORE BODY WIREFRAME AND BLOCK MODELLING

Ore body is modelled using wire-frame modelling technique. In order to create wire-frame model, consecutive sections are used to create strings that represents the ore body. Later this strings are triangulated to create volume of ore body. The created wireframe model is shown in Fig. 2.

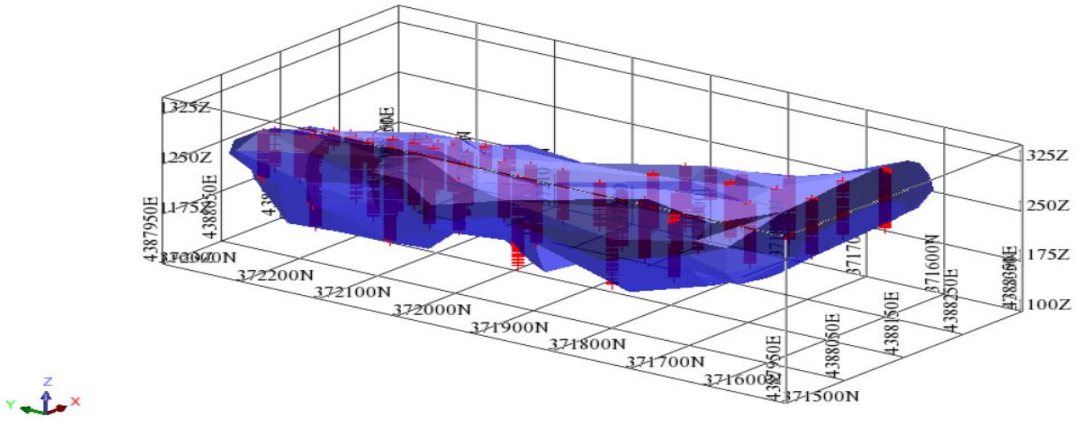


Figure 2. Wireframe model of the deposit.

Total volume of the wireframe model is approximately 10 million m^3.

Ore resource estimation cannot be performed directly on wireframes. Block model that honors the wireframe model has to be created. For this reason, blocks having 20×20×10 m in X, Y and Z directions respectively are used in block modelling. The block model is shown in Fig 3.

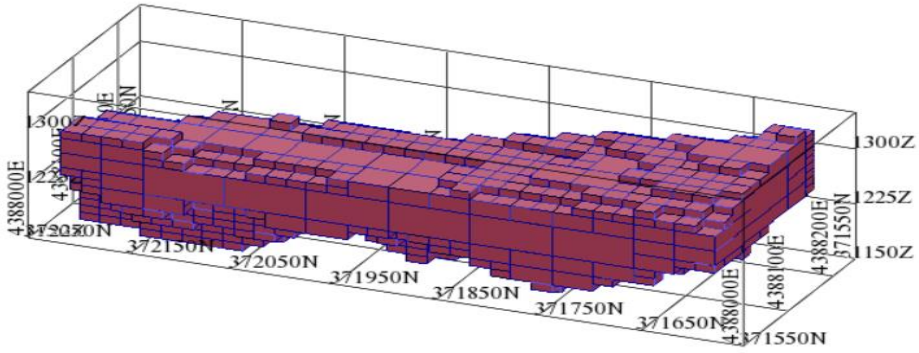


Figure 3. Block model of the iron deposit.

4. ESTIMATION USING DEEP NEURAL NETWORKS

DNNs are ML algorithms that learns the relationship between input and output variable(s). Deep neural networks are specialized form of Artificial Neural Networks (ANN). Artificial neural networks introduced in 1943which is inspired in animal brain. A node which stores the information with 4 input is shown in Fig. 4

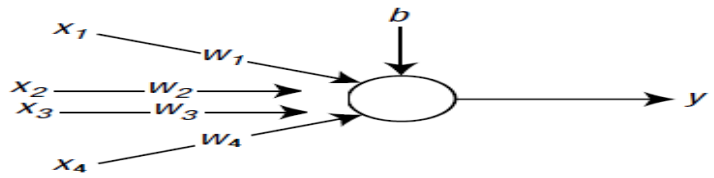


Figure 4. An example node with weightings and bias parameter

The weighted signal collected at the node can be calculated as;

$$v = (w_1 * x_1) + (w_2 * x_2) + (w_3 + x_3) + (w_4 + x_4) + b$$

where b is bias term.

Later final value of the node is calculated using the activation function associated with the neuron as;

$$y = \varphi(v)$$

Deep neural networks are ANN with more than two hidden layer and example of DNN is shown in Fig. 5.

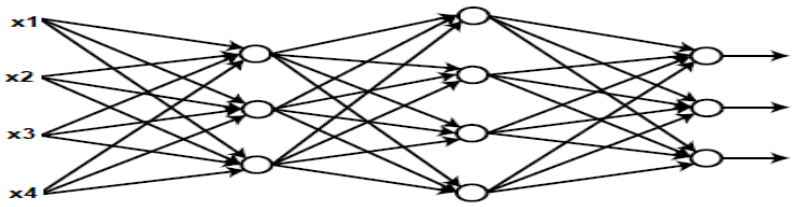


Figure 5.An example DNN.

Deep neural networks are approximately evolved after the ANN are first introduced. In 1986 back propagation algorithm is introduced which make deep neural networks calculation viable and practical.

5. DEEP NEURAL NETWORKS FOR RESOURCE ESTIMATION

In resource estimation using DNN input and output variable(s) have to be determined. In general, in resource estimation input variables are selected as X, Y and Z coordinates of the composites and output is selected target variable measured at these locations.

Following the determination of the input(s) and output(s) the model have to be trained with appropriate activation function with appropriate DNN architecture.

Finally, by training, DNN model becomes ready to make estimation at unsampled locations. Simple feeding the coordinates of X, Y and Z as input where the grade of the target variable is to be estimated. Trained model simply generates the output of grade by using weights associated with all nodes of DNN.

6. ESTIMATION OF THE IRON CONTENT USING DEEP NEURAL NETWORKS

6.1. Selection of the Activation Function

There is numerous activation function is available in use of DNN estimation. Linear, Hyperbolic tangent sigmoid, log-sigmoid function are just examples s-among many. Ore resources are defined on positive space which means that it is not possible to take negative values. For this reason, every activation function cannot be used. The functions which guarantiesthe positive outputs. Log-sigmoid function is one of appropriate candidate among the alternative. Log-sigmoid function is special case of logistic function with formula;

$$log - sig(x) = \frac{1}{1 + e^{-x}} = \frac{e^{x}}{e^{x} + 1}$$

where x is defined in $\pm\infty$

Log-sigmoid function is defined in $[0,1]$ range which makes outputs of the DNN positive. For this reason, log-sigmoid function is used in this study as the activation function.

6.2. Deep Neural Network Architecture Selection

There is no consensus over how to select the architecture of the DNN. For this reason, trial and error methods seems to be best approaches. The [3 15 15 15 10 5 1] architecture is selected in this study where 3 and 1 represents the input and output layer respectively. Remaining integers shows the number of nodes at each hidden layer.

6.3. Training of Deep Neural Network Model

In training of the DNN sample set is subdivided in three categories which are training, validation and testing purposes. Training data set is used to train the network and network is adjusted respect to its error. Validation sub-set is used for halting of the calculations. Calculations halts when generalization converges. Testing sub-set is used for network performance monitoring. This data does not have direct effect on network training. 70 % ,15% and 15% of the data is used for training, validation and testing respectively. In all calculations MATLAB® is used which makes calculations and model tuning easy and error free.

7. RESULTS AND DISCUSSIONS

All blocks are estimated as described at previous section. Resulting block model is shown in Fig. 6.

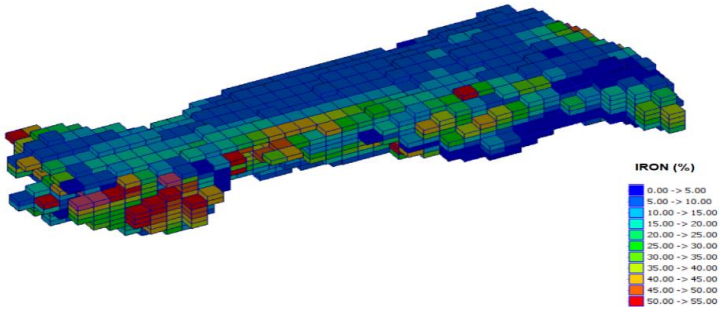


Figure 6. Estimation results.

The summary statistics of the estimations are given in Table 2.

Table 2. Summary statistics of the estimates.

No of Data	Minimum (%)	Maximum (%)	Average (%)	Variance	Skewness
2471	0.01	56.89	13.71	131	1.50

When Table 1 and 2 compared minimum and maximum of the composites and estimates are nearly identical. While average and variance of the estimations are slightly lower than composite values. Skewness of the estimates are higher than composites. Results shows that estimation results are unbiased from the global statistical perspective. This means that estimates can be used for further mine planning and ore reserve estimation purposes. This study also shows that DNNs can be used to ore resource estimation for iron deposits.

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DOI: 10.7251/BMC170701201T

NEW CONCEPTS OF PREVENTING / COMBATING THE RISK OF COAL IGNITION

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ABSTRACT

The phenomena of spontaneous combustion occurred in coal mines are the cause of the chemical reaction of the mineral substance with atmospheric oxygen, with the predisposing factors oxidation tendency (self-ignition) of the carbonaceous material, oxidizing environment and thermodynamic system conditions.

An a priori knowledge of the risk of self-ignition by laboratory determination of the self-ignition parameters and the classification of coal in this regard, provides information necessary for the hazard scenario and prevention plan.

Among the modern methods of preventing / combating the spontaneous combustion phenomenon can be mentioned the monitoring of the underground atmosphere through the fire indices and the use of the nitrogen inerting technology of the exploited space.

This paper is oriented on identify the particularities that occur during the phenomenon of spontaneous combustion through pilot bench investigations, unique in Romanian and modern equipped on which coal oxidation behavior was studied in terms of applying nitrogen inertisation.

By analyzing classical fire indices variation compared to the new ones identified during the development of spontaneous combustion phenomenon in normal and under conditions of nitrogen inertisation were obtained research results that will lead to improved understanding of mine fires, the dimensions on profound scientific bases of measures to prevent and combat the phenomenon of spontaneous combustion.

Key words: coal, risk, spontaneous combustion, fire indices, nitrogen inertisation.

1. INTRODUCTION

The risk of self-ignition of coal represents the probability of generating a phenomenon of spontaneous combustion due to intrinsic and extrinsic factors that contribute to its emergence separately or combined. Depending on the laboratory method of determining in Romania are five groups of risk of self-ignition of coal.

Classic fire indices used for monitoring in Romania, respectively, Graham rate, Breathing rate, index of ethylene and acetylene have created the picture of evolution - involution phenomena of spontaneous combustion occurring in coal mines, for various reasons, objective or subjective. [1]

It is known that at the time of occurrence of this phenomenon, a number of measures are applied for the control of combustion in various ways, by the isolation of the affected deposit for a period of time for workers' safety and the underground coal layer from exploitation area and neighborhoods. Period of isolation remains below observation by periodic measurements of gas concentrations and temperature, air sampling the closed area and physical - chemical analysis in the laboratory.

Depending on the method of fighting applied the isolation period lasts from several months to years, which restrict the extraction activity. For the reduction of isolation is frequently used a system with noncriogenic nitrogen inertisation of the exploited area, which is intended to reduce the combustion intensity, as a result of reducing the concentration of oxygen and cooling the area and to prevent explosions.

Same for inertisation application, the affected area is monitored and the measurements will draw up the chart of evolution-involution of the phenomenon of spontaneous combustion, but in terms of excess nitrogen and artificially decreasing oxygen concentration, based on which are calculated the classic fire indices, appears the question of whether calculated fire indices can accurately reflect the reality on the ground.
The INCD INSEMEX researchers experimentally modeled the on a test bench, the oxidation behavior of coal under nitrogen inertisation application on samples taken from exploitable coal layers from the Jiu Valley mining unit. So, several coal samples were tested in terms of predisposition to ignition.[2]

2. CLASSIFICATION OF COAL ON THE BASIS OF SELF-IGNITION RISK

Self-ignition of coal represents a particular risk for mining industry and appealing to this notion used in the management of health and safety at work, the risk of self-ignition of the mineral substances, is classifying the carbonaceous substance in relation to the method of determining the laboratory, which corresponds to a function of classification (table 2.1)[3]

- Method of determining the risk of self-ignition in gaseous oxygen environment with classification function: the temperature gradient $\Delta T/20'$;
- Method of determining the risk of self-ignition in a liquid medium (perhydrol) function classification: the reaction rate v_r (^{0}C / min.).

Table 1.

Group classification	The behavior of coal	
	in gaseous oxygen environment / the temperature gradient ΔT/20′	in a liquid medium (perhydrol) / the reaction rate v_r (^{0}C / min.).
Group I	Coal without risk of self-ignition, ΔT/20′< 5°C/20′	Coal without risk of self-ignition, v_r< 3°C/min.
Group II	Coal with low risk of self-ignition, 5°C/20′≤ΔT/20′≤ 20°C/20′	Coal with risk of self-ignition, 3°C/min ≤v_r≤ 10°C/min.
Group III	Coal with medium risk of self-ignition, 20°C/20′≤ΔT/20′≤ 35°C/20′	Coal with pronounced risk of self-ignition, v_r>10^{0}C / min.
Group IV	Coal with high risk of self-ignition, 35°C/20′≤ΔT/20′≤ 50°C/20′	-
Group V	Coal with pronounced risk of self-ignition, ΔT/20′> 50°C/20′	-

3. THE TEST BENCH

The automated assembly tracking system of evolution-involution spontaneous combustion parameters have the following components:

- Enclosed heater resistance of 500 W, 4 pcs. in series and 4 pcs. reserves;
- The monitoring system of coal bed temperature with thermocouple sensors with measuring range 50-1600 ° C;
- Temperature measurement system from the air cushion with the sensor "type PT100 resistance thermometer with measurement range of 50-400 ° C;
- TSI model 4043 flow meter flow of 0-100 l / min with calibration certificate;
- Adjustable valve - proportional valve - 3 way 24 VDC loop control;
- Combustion chamber (reaction vessel);
- Specialized software for programming and monitoring operating parameters of evolution-involution process characteristics of spontaneous combustion (Figure 1.)

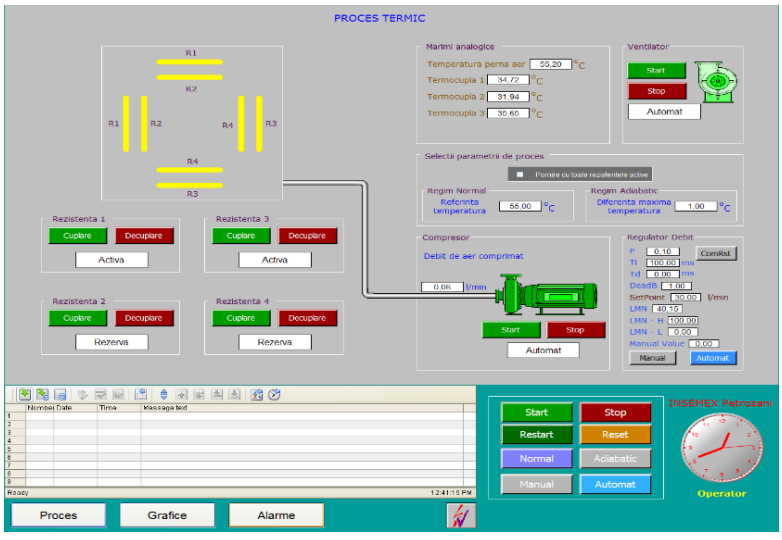


Figure 1. The automated system interface for the thermal process

4. EVOLUTION - INVOLUTION OF SPONTANEOUS COMBUSTION UNDER NITROGEN INERTISATION

4.1. Method for measuring and tracking the phenomenon

After pre-testing the susceptibility to ignition of the coal mining units located in eastern Jiu Valley by oxygen and hydrogen peroxide oxidation method was selected the coal with the highest reaction rate (the more reactive coal).[1]

There been sampled the coal stope individual no. 35, layer 3, Block V - slice XXX from EM Lonea, which have been milled to a suitable particle size by INSEMEX methodology and introduced into the combustion chamber of the test stand (reaction vessel).

The supply of nitrogen was conducted through a branch of the air circuit, from the 50-liter cylinder with nitrogen, charged to a pressure of 200 bar of (volume 10 m^3 N_2-10000 l). In the combustion chamber of the experimental stand, by means of the three thermocouple was automatically measured the temperature variation on the bottom, middle and top of the charcoal column.

Working regime of the test bench was consisted the predefined input parameters:

- Air flow rate the circulated through the sample of coal in the combustion chamber 50 l /min;
- In the insertisation terms, the air flow circulated through the sample in the combustion chamber, 40 l/min and the nitrogen flow rate 10 liters/min.;
- The temperature of the air cushion in both cases: 120 ° C;
- Air relative humidity circulated by samples 54-64%;
- The weight of the samples tested: about 100 kg;
- Particle size samples : 0-10 mm.

The observed parameters:

- sample temperature;
- the air cushion temperature
- changing the oxygen concentration during the process;
- changing gas concentrations, resulting in the evolution and involution of oxidation (CO_2, CO, CH_4, H_2S, C_2H_4 and C_2H_2);

The total flow rate of the air circulated through the coal samples:

- non-inerted sample: 911 636 l (911.6 m^3)
- sample inerted 368 032 l (368 m^3).

Total nitrogen flow through the sample inerted 100,000 l (100 m^3).

The process monitoring and direct measurement of gas concentrations and gas sampling also gas chromatographic analysis and ORSAT to certain temperature thresholds were as follows:

- Direct measurements gas from 10 to 10 ° C, with gas sampling, from t = 50 ° C to T = 200^o C temperature in the sample;
- At T = 200^o C, direct measurement gas, the gas sampling from 50 to 50 ° C, until the temperature the sample T = 500^o C;
- At T = 500^o C, direct measurement gas, the gas sampling from 100 to 100 ° C until the maximum temperature reached in the sample.

For the coal sample modeled without nitrogen, the temperature in the combustion chamber has evolved to the point 770^0 C, then entered the involution phase , and the nitrogen-treated sample has evolved to the point temperature 257^0 C then entered the phase of involution.

At temperatures exceeding 120^0 C (temperature of the air-cushion default parameter) in the sample, the system is self-sustaining process by increasing the rate of spontaneous combustion at every rise in temperature of 100 C, was not conditions of heat exchange with the outside environment, air cushion taking heat from coal. Maximum temperature recorded in the air cushion in the first sample tested 210^0 C and 150^0 C in nitrogen-treated sample.

Temperature measurements were recorded on a computer, gas concentrations were measured directly with portable analyzers and in laboratory by samples taken for chemical analysis with ORSAT and gas chromatograph.

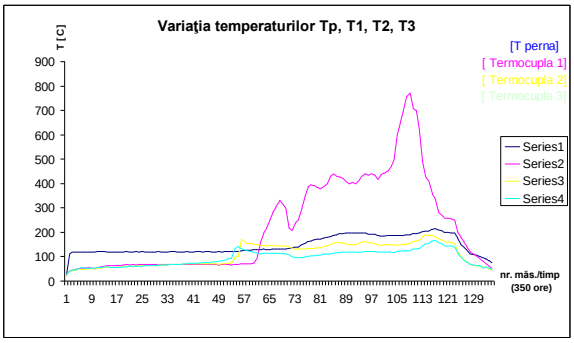
4.2. Temperature and gas concentrations measurements and parameters interpretation

For the coal sample modeled without nitrogen, the temperature in the combustion chamber has evolved to the point of 770^0 C, then has entered in the phase of involution, and the nitrogen-treated sample has evolved to the point temperature of 257^0 C then which entered the phase of involution.

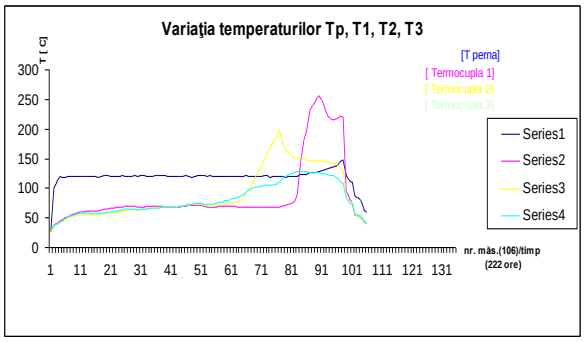
Issues seen:

- T_1 increased from the ambient temperature, constant up to a temperature of 257 °C, then entered into the regression phase;
- Temperature T_2 increased from the ambient temperature constantly to a temperature of 200 ° C, then entered into the regression phase;
- Temperature T_3 increased from the ambient temperature up to a temperature of 128 °C, then entered into the regression phase;
- Oxygen concentration ranged from 10.6 to 20.0% O_2;
- The concentration of carbon dioxide ranged from 0 to 5.2% CO_2;
- The concentration of methane (CH_4) ranged from 0 to 0.15% vol;
- Carbon dioxide concentration ranged from 0 to 2.1440% CO;
- The concentration of hydrogen sulfide ranged from H2S 1 to 2 ppm;
- Hydrogen concentration ranged from 62 to 192 ppm H_2;
- Ethylene C_2H_4 ranged from 0 to 180 ppm;
- Acetylene C_2H_2 ranged from 0 to 4.4 ppm;

Curves of temperature versus time are shown in Figure 2.a (nitrogen untreated coal sample) and in Figure 2.b. (coal sample treated with nitrogen).



a)



b)

Figure 2. Curves of temperature versus time

Temperature variations which were mentioned above, represents the development of spontaneous combustion involution process lasted for 350 hours in untreated coal sample and 222 hours for the sample the coal inerted with nitrogen. For a more relevant description of the phenomenon from the total of 240 measurements were selected 33 measurements, in the temperature thresholds.

5. NEW FIRE INDICES IDENTIFICATION

5.1. Tracking the phenomenon of spontaneous combustion by fire indices: Graham, Breathing, ethylene, acetylene

In the coal samples modeled by programmed heating on the experimental test bench was watched the process of spontaneous combustion by the indices commonly used in Romania and which may indicate the development of spontaneous combustion phases, depending on the values determined.[2]

Fire indices R_1, R_2, ethylene and acetylene determined by the temperature as the measurements of the gas concentrations O_2, CO and CO_2 are listed in Table 2, for the sample without nitrogen and Table 3 for the sample with nitrogen. The tables include:

Sample without nitrogen (Table 2)

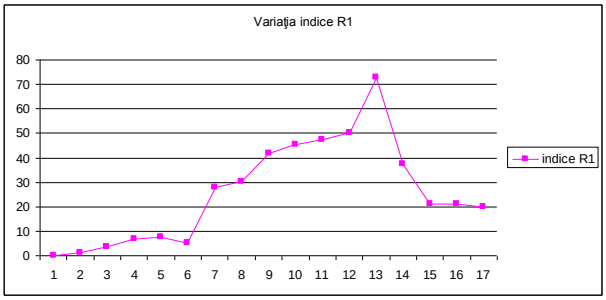
Nr. crt.	Tp C	T1 C	T2 C	T3 C	Concentraţii de gaze (%) O2	CO2	CH4	CO	INDICE GRAHAM	INDICE RESPIR.	INDICE C2H4	INDICE C2H2
0	1	2	3	4	5	6	7	8	9	10	11	12
1	110	**40**	38	41	20.8	0.05	0	0	0.000	35.71		
2	121	**60**	55	57.4	20.5	0.09	0.1	0.006	1.364	20.45		
3	119	69	**70**	71.3	20.3	0.2	0	0.022	3.438	31.25		
4	120	68	**70**	79.6	20	0.32	0	0.062	6.596	34.04		
5	120	65	69	**80**	19.8	0.39	0	0.085	7.456	34.21		
6	121	67	72	90.3	19.1	0.6	0.15	0.096	5.217	32.61		
7	122	69	101	141	16	7	2.7	1.376	27.854	141.70		
8	124	70	**160**	**140**	15.9	7.04	2.9	1.525	30.258	139.68	0.00031	0.00
9	131	**200**	146	113	1.4	10.9	25.6	8.192	41.924	55.78	0.00300	0.00003
10	130	**220**	145	113	1.2	11.5	25.6	8.96	45.390	58.26	0.00220	0.00006
11	140	**250**	132	65.8	0.4	11.7	25.6	9.7	47.225	56.96	0.00030	0.00002
12	149	**300**	132	97.2	0.8	10.8	0.011	10.09	50.20	53.7	0.00040	0.00010
13	155	**350**	132	97.6	1.6	5.9	0.023	14.08	72.802	30.51	0.00040	0.00010
14	197	**400**	150	120	1.0	10.1	0.230	7.5	37.6	50.75	0.00060	0.00005
15	186	**500**	148	117	0.7	12.4	0.9	4.23	20.94	61.38	0.00740	0.00007
16	186	**600**	146	122	0.7	10.8	0.6	4.23	20.94	53.46	0.00500	0.00007
17	188	**700**	150	124	0.9	10.1	0.8	3.95	19.75	50.50	0.00880	0.00004

Sample with nitrogen (Table 3)

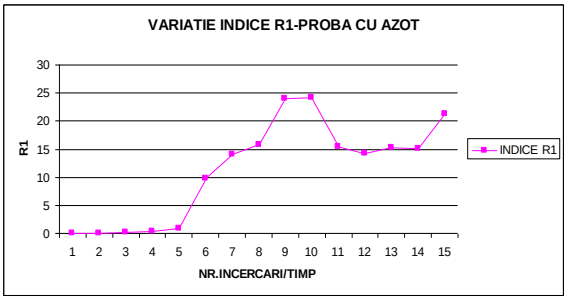
Nr. crt.	Tp C	T1 C	T2 C	T3 C	Concentraţii de gaze (%) O2	CO2	CH4	CO	INDICE GRAHAM	INDICE RESPIR.	C2H4	C2H2
0	1	2	3	4	5	6	7	8	9	10	11	12
1	112	41	39	39	19	0	0	0	0	0		
2	120	**60**	55.09	57.8	17.8	0.17	0.007	0.0026	0.083	5.41		
3	121	65.2	57.8	59.7	17.7	0.06	0.006	0.0067	0.209	0.83		
4	120	**70**	62	64.3	17.5	0.06	0.009	0.0105	0.305	1.74		0.000022
5	120	**70**	**70**	**70**	17.2	0.13	0.003	0.029	0.775	3.48		0.00002
6	120	67.5	85.1	90	16.9	0.8	0.1	0.392	9.703	19.80		0.00050
7	120	67.5	**100**	97.2	17.6	0.82	0.1	0.465	14.09	24.84		0.00130
8	120	67.5	**120**	102.3	17.2	0.8	0.06	0.586	15.83	21.62		0.00130
9	119	68.06	**170**	106	16.9	0.9	0.05	0.96	24.0	44.12	0.00004	0.00070
10	120	68	**190**	107.4	16.7	1.24	0.06	1.024	24.15	29.25	0.00040	0.00120
11	120	68.9	**200**	109.7	13.6	2.24	0.15	1.128	15.36	30.52	0.00044	0.00180
12	120	72.4	160	123	12.8	2.69	0.1	1.152	14.22	26.53		0.00060
13	122.5	143.9	152.3	128	12.5	3.0	0.15	1.28	15.23	35.7		
14	123.7	**180**	150.4	128	12.0	3.45	0.15	1.344	15.1	38.8		
15	124	**200**	149	128	11.4	4.72	0.1	2.016	21.22	49.68		
16	127	**250**	146	127	10.6	5.20	0.1	2.144	20.8	50.5		

The charts related to the Graham R1 index variation are listed below in 3.a - 3.b figures.

a) Sample without nitrogen



b) Sample with nitrogen



a) Index R_1 sample without nitrogen

b) Index R_1 sample with nitrogen

Figure 3. Graham R_1 index charts variation

5.2. New fire indices usable to inertization with nitrogen

By inertization with nitrogen (N_2), there is a significant artificial decrease in the oxygen concentration and not as a phenomenon due to oxidation. Under these conditions, by fire indices indices R_1 and R_2 dependent on the oxygen concentration, presents deviations from normal values related to the evolution-involution phenomena of spontaneous combustion. In this respect it is necessary to identify indices of fire to be independent of the changes in oxygen concentration. New fire indices that can meet this condition are generally reports of gas concentrations resulting from oxidation of coal, whose values do not depend on artificial variation of oxygen content.

We shall have the fire indices determined in absolute value. Such identification is the first index I_1 as the ratio of R_1 and R_2:

$$I_1 = \frac{R1}{R2} = \frac{\frac{\Delta CO}{\Delta O2} * 100}{\frac{\Delta CO2}{\Delta O2} * 100} = \frac{\Delta CO}{\Delta CO2} \quad (1)$$

It is seen from equation (1) the index I_1 does not contain oxygen and therefore the absolute value of the index will be independent of the concentration of oxygen.

Also, the reverse of index I_1, may be identified as an indicator I_2 as the report of the R_2 and R_1, the absolute value:

$$I_2 = \frac{R2}{R1} = \frac{\frac{\Delta CO2}{\Delta O2} * 100}{\frac{\Delta CO}{\Delta O2} * 100} = \frac{\Delta CO2}{\Delta CO} \quad (2)$$

I_3 index, the ratio in absolute value of the concentration of carbon monoxide (CO) and the concentration of hydrogen (H_2):

$$I_3 = \frac{CO}{H2} \quad (3)$$

I_4 Index the ratio in absolute value of the concentration of carbon dioxide (CO_2) and the concentration of hydrogen (H_2):

$$I_4 = \frac{CO2}{H2} \tag{4}$$

I_5 Index the ratio in absolute value of the concentration of ethylene (C_2H_4) and the concentration of hydrogen (H_2):

$$I_5 = \frac{C2H4}{H2} \tag{5}$$

I_6 Index the ratio in absolute value of the concentration of acetylene (C_2H_2) and the concentration of hydrogen (H_2):

$$I_6 = \frac{C2H2}{H2} \tag{6}$$

I_7 Index the ratio in absolute value of the concentration of ethylene (C_2H_4), and the concentration of acetylene (C_2H_2):

$$I_7 = \frac{C2H4}{C2H2} \tag{7}$$

However, the use of the absolute index of carbon monoxide q_{co} is appropriate for a with nitrogen inertisation.

$$q_{CO} = C_i \,.\, Q \,.\, 10^{-3}[\ l/min\ ] \tag{8}$$

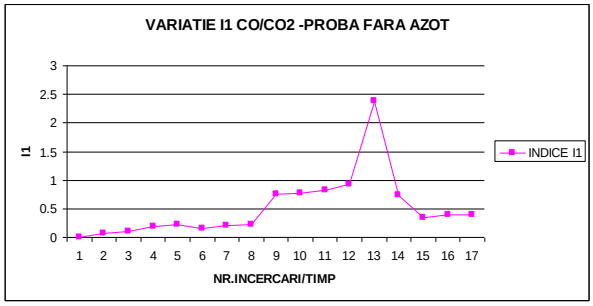
In (8) the terms of the expression is the concentration of CO, vehicle and constant flow uniformity.

5.3. Graphical representation of the fire indices

To get a picture of the phenomenon of spontaneous combustion carried into the reaction vessel, for the samples modeled on the experimental stand output data obtained from measurements and gas chromatographic determination were processed with the by analyzing the comparative evolutions involutions of the fire indices with Graham R1 as reference on boards of values from the table nr. 4.1 and 4.2, from subchapter 4.1.

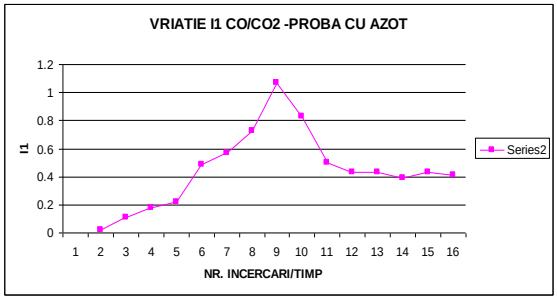
5.3.1. Index I_1 $I_1 = \frac{\Delta CO}{\Delta CO2}$

a) Sample without nitrogen



a) Index I_1 sample without nitrogen

b) Sample with nitrogen



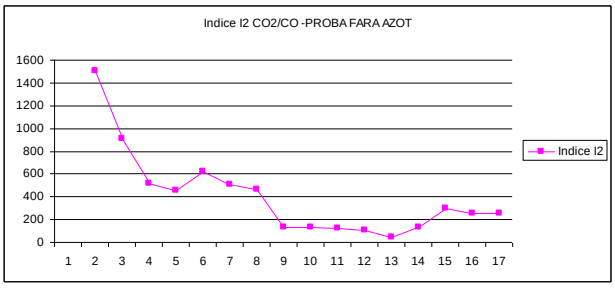
b) Index I_1 sample with nitrogen

Figure 4. I_1 Chart variation

Given that Graham R_1 index is the fire index that presents variation compared to the temperature in the furnace respectively concentration of CO, results that it follows the spontaneous combustion. From the two graphs it is observed that the index I_1 follows closely R_1 index variation.
If the phenomenon of spontaneous combustion modeling, using nitrogen inertisation, it is noted that the variation in the index I_1 is symmetric with R_1 index variation in the terms process development without nitrogen inertisation, the fact that the index I_1 varies fairly with R_1 index.

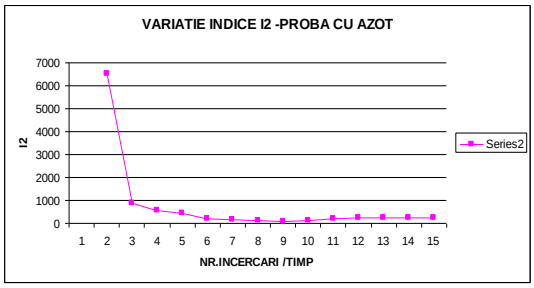
5.3.2. Index I_2 $I_2 = \frac{\Delta CO2}{\Delta CO}$

a) Sample without nitrogen



a) Index I_2 sample without nitrogen

b) Sample with nitrogen



b) Index I_2 sample with nitrogen

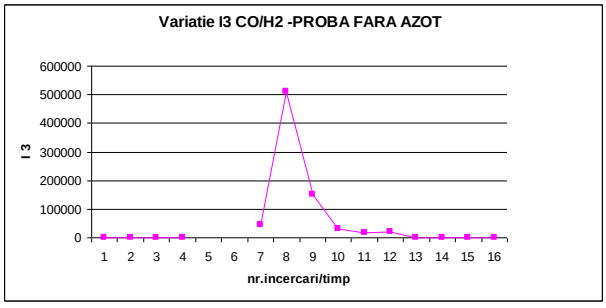
Figure 5. I_2 Chart variation

From the diagrams, it is noted that the index I_2 varies inversely related with the index R_1 so that the maximum value of the index R_1 corresponds to the minimum value of the index I_2.

Modeling application with nitrogen to coal sample index I_2 is inversely proportional to the index R_1, so that the minimum value of the index corresponding to the maximum value R_1 index I_2 which tends asymptotically to the abscissa axis.

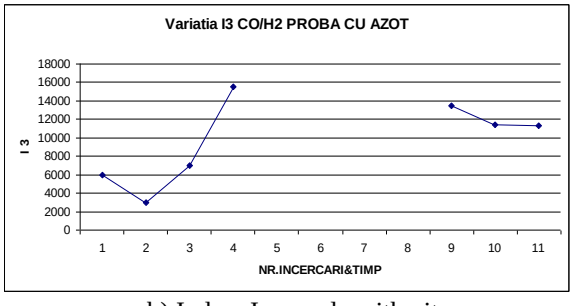
5.3.3. Index I_3 $I_3 = \frac{CO}{H2}$

a) Sample without nitrogen



a) Index I_3 sample without nitrogen

b) Sample with nitrogen



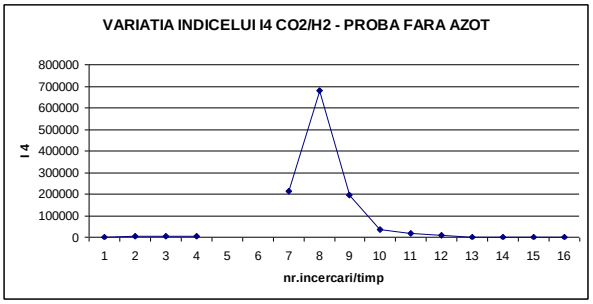
b) Index I_3 sample with nitrogen

Figure 6. I_3 Chart variation

From the charts above was observed for both cases that index I_3 follows the path of R_1 index variation.

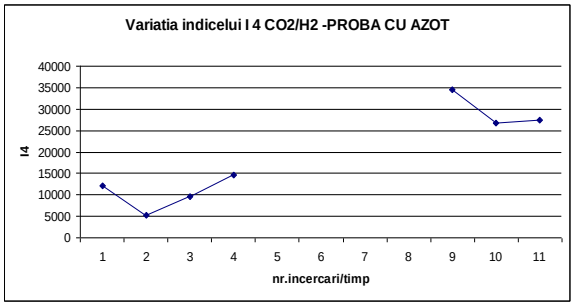
5.3.4. Index I_4 $I_4 = \frac{CO2}{H2}$

a) Sample without nitrogen



a) Index I_4 sample without nitrogen

b) Sample with nitrogen



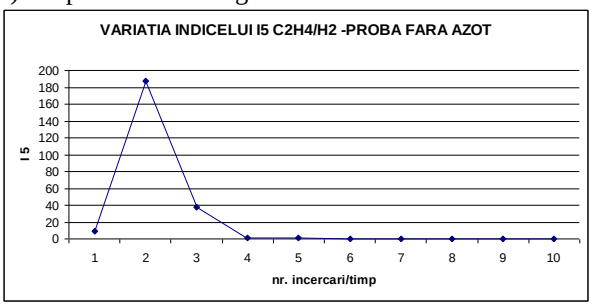
b) Index I_4 sample with nitrogen

Figure 7. I_4 Chart variation

The nitrogen-free sample from I_4 index diagrams results that, it follows accurately the path of the index R_1 in the phase of evolution and involution during the process of spontaneous combustion. With nitrogen inertisation, I_4 index variation has a similar behavior index variation R_1.

5.3.5. Index I_5 $I_5 = \frac{C2H4}{H2}$

a) sample without nitrogen



a) Index I_5 sample without nitrogen

b) Sample withnitrogen

?

b) Index I_5 nitrogen sample

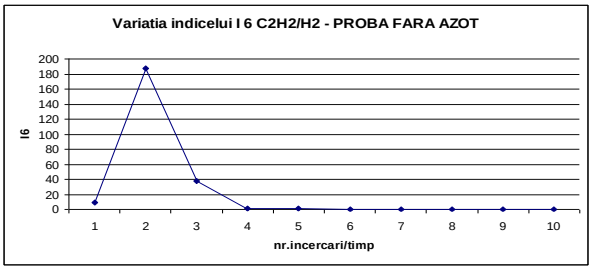
Figure 8. I_5 Chart variation

From the diagram shown previously there is a change in the of the index I_5 inversely proportional to the change in the index R_1. Also, a linearization curve of I_5 index in the second phase of the evolution of R_1.

In the case of inerted coal sample given that the phenomenon developed only up to a certain temperature threshold it was not possible to achieve a variation in the index I_5 compared to R_1.

5.3.6. Index I_6 $I_6 = \frac{C2H2}{H2}$

a) Sample wothout nitrogen



a) Index I_6 sample without nitrogen

b) Sample with nitrogen

?

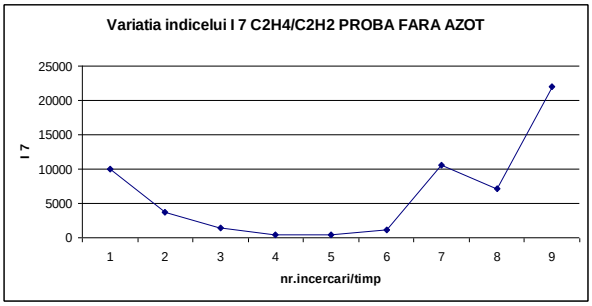
b) Index I_6 nitrogen sample

Figure 9. I_6 Chart variation

From the charts, there is a change in the index I_6, inversely proportional to the change in the index R_1, especially early in the evolution process of spontaneous combustion. For the inerted sample I_6 index behaves similarly as the index I_5 being unable to record a comparative variation R_1.

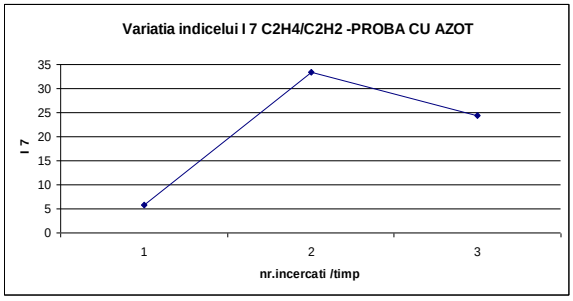
5.3.7. Index I_7 $I_7 = \frac{C2H4}{C2H2}$

a) sample without nitrogen



a) Index I_7 sample without nitrogen

b) sample with nitrogen



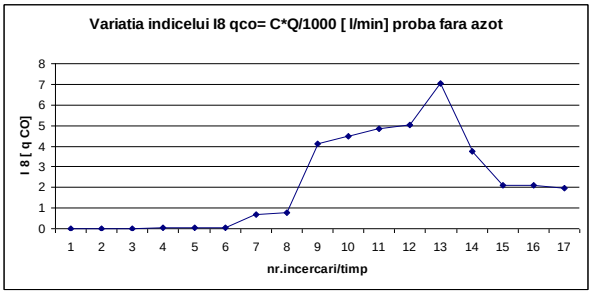
b) Index I_7 sample with nitrogen

Figure 10. I_7 Chart variation

The analysis chart shows that the index I_7 index varies inversely related with changes in R_1. For nitrogen inerted sample I_7 index varies proportionally with the index R1. Due to insufficient data, I_7 index variation in these conditions is uncertain.

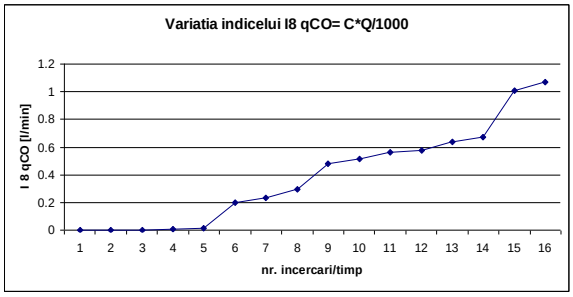
5.3.8. Index I_8 $q_{CO} = C_i \cdot Q \cdot 10^{-3}$[l/min]

a) sample without nitrogen



a) Index I_8 sample without nitrogen

b) sample with nitrogen



b) Index I_8 sample with nitrogen

Figure 11. I_8 Chart variation

From the diagrams presented, in both cases, there is an accurate variation in the index q_{CO} compared with the index R_1, both during development and in the phase of involution of the phenomenon of spontaneous combustion.

6. CONCLUSIONS

Following the comparative analysis of possibilities to use new fire indices used in both normal and under nitrogen inertisation in relation to the process of spontaneous combustion resulted the following:

1. In normal development of spontaneous combustion process can be used the I_1, I_2, I_3, I_4, I_7 and I_8 with high certainty, and indexes I_5 and I_6 with a high degree of uncertainty;
2. In the case of conducting the phenomenon of spontaneous combustion under nitrogen inertisation, the I_1, I_3, and I_8 indices can be used with high certainty, I_2 and I_7 indices can be used with high uncertainty, and indexes I_5 and I_6 are recommended for use.

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DOI: 10.7251/BMC170701213A

POSSIBLE APPLICATIONS OF BUSINESS INTELLIGENCE IN THE MINING INDUSTRY

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ABSTRACT

We are contemporaries of a new industrial revolution, whose driving forces are based on data and knowledge, digitalization, automation and the continuously developing information technologies.

The mining industry is no exception to this trend. In the last decade the big mining companies have implemented different types of mass information systems for management of individual production processes or the necessary modules from modern ERP systems. The volume of collected primary information naturally grows together with the development and integration of the technology. These are large and complex datasets that are difficult to be processed.

Business Intelligence offers a solution to these problems . BI is a set of methods for the acquisition, processing, analysis and visualization of data in order to extract quickly meaningful and useful business information.

An important feature of BI systems is their ability to process large amounts of data in real time, which makes them particularly useful in making management decisions in the dynamic conditions and processes in the mining industry.

Key words: Business Intelligence, Mining, Possible Application

1. INTRODUCTION

Nowadays almost all spheres of our activities are invariably associated with the collection, processing and presentation of data. The need for rapid processing and reaction stimulates the emergence of new digital products and services, the development of business processes and scientific methodologies. The trend is global and covers all areas – security, climate, health care, efficient use of resources, energy, intelligent transportation systems, smart cities, digitization of public services, etc.

The mining industry is no exception to this trend, as digital technologies have been massively introduced in the last decade.

All companies, including the mining ones, require flexible and real-time information to allow dynamic change of the business processes on the basis of effective business solutions. The strategic objective of any company is to achieve better results using less resources, or maximum optimization of business processes. [1]

The Business Intelligence (BI) systems, which include business analytical models, technologies, information and practices for making the most informed and optimal management decisions, help for the solution of this problem. Through extracting information and regularities, BI systems can evaluate the information received and on this basis to produce forecasts for the development of the company.

2. BUSINESS INTELLIGENCE – ARCHITECTURE AND SOFTWARE

In order to effectively use the basic functionality of BI, the presence of structured data stored in relational databases, NoSQL, Data Warehouse, Big Data, etc. is required.

The producers of BI systems need to achieve maximum acceleration of the analytical processes (if possible, in real time) via a high-performance combination of hardware and software. The standard relational databases are not flexible enough to meet the challenges of Big Data. The architecture of BI systems allows for connecting to relational databases, but also for serving large amounts of information that are not based on standard relational models.

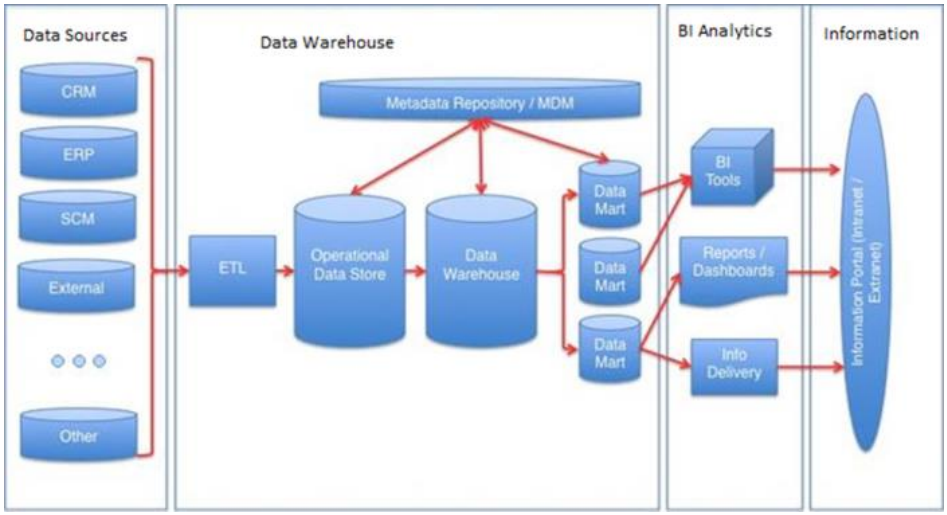


Figure. 1. Business Intelligence solution

BI systems rely on interactive Dashboards, which are equipped with very good graphical interfaces. They are used to easily cover the needs of the business and give summary information online. They also provide graphic visualization of data that reveals the regularities between them.

The advantage of Dashboards is that they provide monitoring of data from large-volume business reports on high level. In case of deviation from the key performance indicators (KPI) this allows the entering in the reference depth (drill down).

Another important advantage of BI systems is the ability to group the information by different criteria and dimensions, which transforms the report according to the specific needs of the business.

According to the Gartner Group's ranking [2], the two leading BI software products are *Tableau* and *Power BI.*

Both products have their advantages, which make them favorites for BI tools.

Tableau has been the market leader for the fifth consecutive year as it is one of the most dynamically developing BI solutions. Available in *Tableau Desktop, Tableau Server* and *Tableau Online.* [3]

Tableau Desktop is an application for the visualization of data and allows you to quickly generate interactive displays (dashboards), beautiful graphs and reports.

Tableau Server is a corporate platform for Business Intelligence that offers powerful tools for analyzing and visualizing data. It provides high availability and security and allows multiprocessor processing. It works on both physical and virtual machines. Tableau Server provides online multi-user platform, designed for sharing and collaborating with reports and visualizations.

Tableau Online is Cloud-based Tableau solution for Business Intelligence. It provides flexibility and ease of application for powerful data visualization. Since it is cloud-based, the need for expensive servers, management software and IT support drops off and it can be expanded according to the users' needs.

Tableau has an extremely scalable client-server architecture (Figure 2) that serves mobile, web and desktop clients.

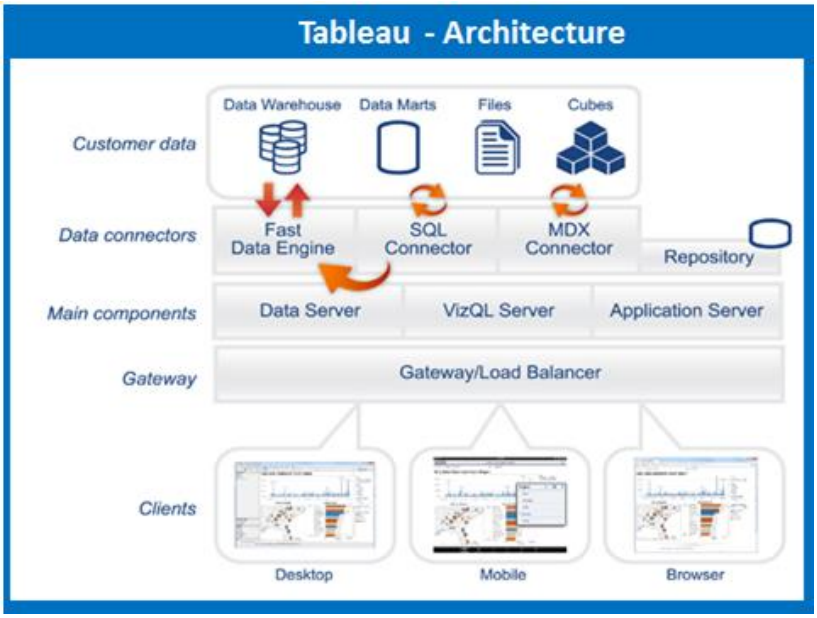


Figure 2. Architecture of Tableau

The methods for connection and retrieval of data are two in the Tableau – Live and In-memory. The users can determine what method to use.

Power BI is a cloud-based Microsoft product that provides business analytics services. The product is a collection of powerful intuitive tools and functionalities for statistical analysis and visualization of large data sets. Power BI is built on Azure, which is Microsoft's cloud service platform.

The Power BI architecture is based on two clusters - the WFE (Web Front End) Cluster and the Back End Cluster (Figure 3). WFE takes care of the initial connection and authentication, and then Back End processes all subsequent interactions with the user. [4]

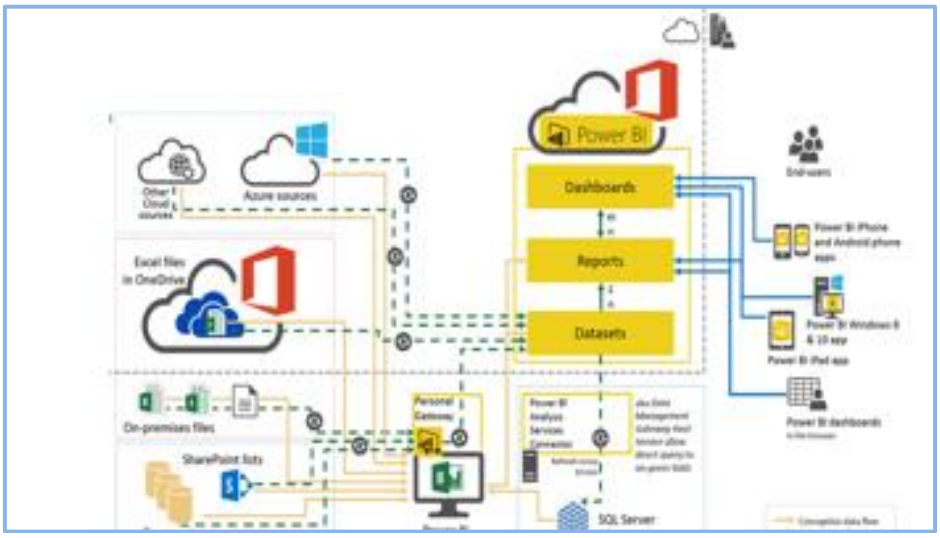


Figure 3. Architecture of Power BI

Power BI uses Azure Active Directory (AAD) to manage users, in addition it manages the data storage and metadata process via Azure BLOB and Azure SQL Database.

Tableau and Power BI provide powerful tools for extracting, processing, and displaying data. Both products offer a wide variety of connectors to different data sources such as relational databases (MS SQL Server, MySQL, PostgreSQL, Oracle, etc.), spreadsheets (MS Excel), Web analytics services, data Warehouses, Big Data Solutions.

Both products support programming languages by which users can create their own graphs or apply specific methods for deeper statistical analysis of data in order to retrieve business-friendly information. These are the R and the Python (Tableau only) languages.

Tableau Desktop and Power BI interfaces are intuitive and easy to use. Through cloud services, users can access their reports from different devices at any time.They are available for Android OS, iOS, Windows OS.

3. APPLICATION OF BI IN THE MINING INDUSTRY

Mining companies cover vast areas, and sometimes the different branches of a large company are located not only in different countries but also on different continents. There is a natural need for the important information resources to be accessible from every point in the company. For this reason many big companies build their own information networks, uniting different types of information systems.

It is typical for the mining industry that the information process is very dynamic, and the data obtained is continuously supplemented with new elements. This leads to the accumulation of huge volumes of data that are difficult to process with classical analytical methods. The so-called "unnecessary data" that further complicates the processing and extraction of knowledge that will quickly support or justify a particular management solution is another problem.

Nowadays, on a global scale, the mining industry faces many different challenges - both operational and because of the changing regulatory requirements.

One of the most important challenges is the constant change in the prices of metals, which requires the use of innovative technologies to maintain or reduce their costs. At the same time, they must also comply with the country's legal, regulatory and environmental standards, which are an important requirement when expanding into new regions and countries.

To deal effectively with all these challenges and to be competitive in the metal market, many mining companies around the world and in Bulgaria already use the BI tools. These tools provide them flexibility in making management decisions maintain job efficiency, while at the same time reducing costs.

BI systems in the mining industry can be successfully applied in:

- Monitoring the quality and grain size of the raw material extraction,
- Tracing the parameters in the mineral processing factories;
- Monitoring the characteristics of groundwater and surface water;
- Optimization of the mining and transport mechanization used;
- Optimization of both the individual processes and the mine work as a whole;
- Optimization of the administrative processes inside and outside the mining company;
- Tracking and optimization of critical manufacturing processes;
- Compliance with the established standards;
- Human resource planning and others.

The main advantage of BI systems is that they enable fast (real time) trend analysis, which is particularly valuable in making strategic decisions for the development of mining companies, where the return on investment is over a relatively long period of time. [5, 6]

BI solutions can find new business directions for companies. An example in this regard is Pulse Mining Systems, which provides ERP solutions for the mining industry. Since the ERP system database used an older technology, this challenged the company when it needed to retrieve and visualize information. Using the Tableau BI system, they were able to create clearer and better visualizations of the processed data. [7]

Rob Parvin, Manager of Visualization and Analytics, claims that thanks to BI, Pulse Mining Systems managed to create a new business segment that is a precedent for the company. With the help of Tableau, they managed to create a single solution for water quality reporting across the overall mining organization. This stimulates the emergence of further initiatives for which the company receives new funding.

4. CONCLUSION

Nowadays, we are witnessing the continuous development of information technology. New sources of raw data are emerging that need to be processed.

The mining companies already have high-tech management systems, resource allocation and high security standards that generate huge amounts of data.

The choice of a particular BI product depends on the specifics of the work and security policies.

If the data is strictly confidential, the user should choose the BI solution of the Tableau. Although the initial costs are relatively high, the data is stored locally (on-premises) in the company and the security level is higher, and the prepared reports and dynamic dashboards can be viewed with Tableau Reader.

If a company uses Microsoft products and platforms, the best choice would be Power BI as one of its key advantages is that it integrates easily into your ERP and CRM systems of Microsoft.

BI systems give us the ability to understand what "the data tells us" through detailed and interactive data visualizations. Information processed through the BI system helps in making the right decisions, strategies and forecasts that drive businesses forward, ensure competitiveness and stable market positions.

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DOI: 10.7251/BMC170701219D

SEISMIC ACTIVITY IN SURFACE MINE OF ARCHITECTURAL BUILDING STONE AT THE TIME OF BLASTING

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ABSTRACT

Exploitation of architectural - building stone in the form of blocks for further processing as a separate segment of the mining industry,is one of the first human activities that, with the development of human civilization, despite the creation of new materials, not only dying out, but gaining momentum. The exploitation of these stone blocks (marbles, granites, travertines, onyx and others) used as architectural-building stones, explosives and mining methods are applied in a completely different way than in the usual methods of blasting..

Key words: blasting, seismic effects, elastic waves, marble, measurement

1. INTRODUCTION

These blasting methods are used exclusively in combination with the production of limited surfaces of a healthy marble mass by performing side cuts and cutouts with classic machines such as diamond woven saws, diamond chain saw, etc.These limitations are made with the primary task during these mining operations not to damage the surrounding healthy rock masses while breaking down or collapsing the block itself or the crushed mass.The greatest attention is paid to the impact on the surrounding benchess and blocks where the healthy rock mass can not be damaged.

The explosive as a means in mining of the surface pits for the exploitation of stone blocks is generally applied in many technological phases of exploitation and with several different purposes.

Drilling and blasting in the exploitation phase is applied at:

- sprinkling of cracked zones (primarily cracked zones),
- shrinkage of low-quality primary zones,
- making cuts and channels for the formation of a work shaft,
- separation of lamellas from the solid rock masses,
- cutting of dropped lamellas with drilling.



Figure 1. Part of the surface mine for white marble "Sivec", near Prilep

2. GENERAL FOR SEISMIC ACTIVITY AT THE TIME OF BLASTING

The elastic deformations caused by the dynamic effect of the explosion represent an oscillatory process, that is, the seismic effect of the explosion**.**

The formed elastic deformations extend in the form of elastic waves radially from the site of the explosion.With the action of an explosion in the working environment simultaneously, all types of elastic waves are formed, whereby the change in the distance from the blasting site also changes their intensity.The intensity of the seismic waves can be determined by measuring one of the basic dynamic parameters of the moved environment, and this:

oscillation speed (V),

acceleration(a)and movement on the ground (x).

The connection between these parameters is obtained by analytical calculations.

The parameter which is used most often for assessing the seismic intensity is:the speed of seismic oscillations on the moved ground (v).

The maximum speed resultant of ground oscillation (vmax) is obtained as the intensity of vectors with components in the X, Y and Z axis according to formula:

$$V_{max}= \sqrt{V_v^2 + V_l^2 + V_t^2}\ , \text{(mm/s)}$$

Where are:

- V_v - a vertical component of the oscillation speed of the ground, (mm/s);
- V_l - longitudinal speed component of the ground oscillation, (mm/s);
- V_t - transverse speed component of groundoscillation, (mm/s).

The registration instruments are placed on hard ground, dug into the ground or the rock. They can also be installed in building structures on a concrete base or hardly compacted earthen ground. The installation of the instrument is always directed with the arrival of seismic waves.

2.1. Registering of the vibration

At points of observation, the waves formed as a consequence of the blasting effect arrive almost simultaneously, causing the movement of the soil particles in different directions. The material particles of the ground oscillate in space and time along very complex paths.

The vibrations are registered in three directions, through the:

1. A horizontal component, perpendicular to the direction of wave propagation, the so-called transverse component, V_m;
2. Horizontal component, oscillation in the direction of wave propagation, so-called. longitudinal component, V_l;
3. A vertical component, V_v.

As the most influential and appropriate parameter for the evaluation of the seismic action during blasting, usually is taken the speed oscillation of the ground.

It is considered that it best correlates and describes the danger of vibrations and damage that can be caused, so all standards and criteria for protection against vibrations are based on the speed oscillation of the ground. By reading from the instrument, we can calculate the maximum oscillation speed (Vmax) and the actual (real) maximum oscillation speed (Vstv).
The real maximum oscillation speed is obtained by reading the amplitudes from the recorded velocity in the same time interval for all three components, and then calculating the resulting real speed using the same formula.By using the maximum oscillation speed (Vmax) to evaluate the seismic action from the blasting, the results are fairly reliable, since the maximum oscillation speed is always greater than the actual, and the real one is that which acts on the object.

2.2. Criteria for assessment of seismic action during blasting

Regulations are regulated in several countries that regulate the level of vibrations caused by blasting that can load objects depending on their significance, condition and dynamic resistance.Such regulations have not yet been adopted in our country in order to solve such problems, we are using regulations and norms from abroad, that is, usually according to Russian, German and American standards.

The intensity of the quake-caused from blasting is sometimes estimated through a scale used in seismology, most often with the Merkali-Kankani-Siberge scale, this scale is known as the MCS scale.Although there are differences between quakes from blasting and quakes from earthquakes, the differences are significant especially during the duration of the quakes, the oscillation period, etc.In determining the permissible deformations for objects that are located on a certain type of rock mass, the deformation characteristics and physical and mechanical characteristics of the corresponding rocky massif are of great importance.

It can be concluded that the vibration, if it is of high intensity, causes such pressure in the rock material, to deformations cross the boundary of elasticity and the appearance of additional mechanical deformations, with these deformations, especially in the marble mass, there are cracks and lasses that make the production of commercial blocks of good quality difficult. For the stability of an appropriate object or rock mass, take into account the calculated permissible deformation, in which the elastic behavior of the rocks must not pass the value of 0,0002-0,0005.

Table 1. Description of the appear in the rock massif

Description of rock mass deformations caused by seismic waves	Speed of oscillations, cm/s
There are no deformations	<20
Occurrence of insignificant development of cracks from previous blasting, a mystic fall of individual pieces of slopes from previous damages	20-50
Intensive development of permanent cracks, individual crumbling of smaller pieces of rocks, occurrence of rocks in tectonically weak rocks and collapse of the slopes after tectonic deformations	50-100
Development of tectonic cracks and collapse of pieces with dimensions 0,5x0,5x0,5m	100-150
Collapse from the sides and roof parts of the underground rooms, after tectonic cracks, formation of new cracks, demolition of protective pillars and parts of the bench	150-300
Fully damaging the sides of the slopes on larger sides 1x1x1m and filling the underground rooms to half the height and breaking down the slopes from the solid rocks	300-400
Full decay of the rocky massif, the collapse of large blocks larger than 1x1x1m and the burial of the underground rooms more than half of the entire height	>400

Table 2: Allowed oscillation speeds by rock type

Characteristics of the rocky massif	Coef. of hardness, f	Velocity of long wave, C_p(km/s)	Allowed oscillation rate of the object by category, cm/s			
			I	II	III	IV
Unbound alluvium	0,5-1	1-2	4,08	8,2	12,2	20,4
Strongly cracked clay with clay and high porosity	1-3	2-3	6,80	13,6	20,3	34,0
Layered rocks meaningfully cracked	3-5	3-4	9,50	19,0	28,4	47,5
Significantly homogeneous rocks with individual cracks and cavities	5-9	4-5	12,2	24,4	36,7	60,0
Faint cracked and monolithic rocks	9-14	5-6	14,9	29,8	44,6	74,5
Very solid and monolithic rocks	14-20	6-7	17,8	35,6	53,3	89,0

3. CONCLUSION

Miner works have a detrimental effect on the surrounding benches in the mines for architectural-building stone, and the most damaging influence are the seismic waves that are unavoidable and extend and activate the already existing cracks and create new ones, and in these mines, the best quality of the commercial block its set on.

However, the blasting cannot be excluded as one of the most important primary mining operations.

So we should try to find ways to minimize these harmful effects.Therefore, only those masses that are already damaged, ie that do not have a good quality for sale, are usually mined and are contoured so as not to transfer the waves from the blasting of a healthy marble mass.

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DOI: 10.7251/BMC170701223M

SAFETY MEAUSERS AT COAL EXTRACTION IN VELENJE COAL MINE IN ORDER TO PREVENT THE EFFECTS OF SUDDEN INRUSHES AND OUTBREAKS OF COAL GASES

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ABSTRACT

Velenje Coal Mine developed, with respect to the natural forces of fracturing and crushing of the coal layer, one of the most effective longwall top coal caving mining method for thick coal layers. "Velenje mining method" with collapsing of overburden strata in excavated area is developed since 1952. The process of controlled caving by pouring crushed coal in the front of flexible canopies and on the armored face conveyor allows very high extraction capacity with provided safety. This procedure, however, requires a good knowledge of caving processes "in situ", advanced excavation equipment and in particular, knowledge and experienced miners. Successful control of caving and continuous collapsing of excavated area is one of the key stages of the technological process for the efficient and above all safe extraction of coal. From the safety point of view, this means in particular the prevention of sudden inrush of coal gases in the work area of coalface as consequence of sudden collapse of overburden strata in the extracted area. Also, due to the impact of mining especially when the coalface is crossing the coal layer Faults or old works, the potential danger exists of an outbreak of coal gases in the area of coalface and gateways. This paper presents measures for safe excavation that are integrated in the Velenje Coal Mine's "Prevention and protection plan" in the case of potential risks and at the occurrence of inrush or outbreak of coal gases in the area of coal face and gateways.

Key words: Velenje mining method, inrush and outburst of coal mine gases, prevention and protection plan

1. INTRODUCTION

Velenje Coal Mine is mining one of the thickest lignite layer known in the world, with a maximum thickness of more than 160 m. Velenje Coal Mine developed, with respect to the natural forces of fracturing and crushing of the coal layer, one of the most effective longwall top coal caving mining method for thick coal layers. The "Velenje mining method" with collapsing of overburden strata in excavated area is developing since 1952. In the present system, the extraction from top to bottom of a thick lignite layer is delimitated on pillars (horizontal delimitation) and horizons (vertical delimitation). In Velenje the longwall extraction principe was developed in to major variants: horizontal concentration and vertical concentration. Whit introduction of "safe extraction criteria" under aquifer layers, which

allow continuous change in the height of the extraction in dependence on the thickness of the insulation layer above the coal layer, the pressure of the water in the aquifer layer above and the depth of the longwall face, the difference between the horizontal and the vertical concentration is lost. We are only talking about the longwall method with increased or variable extraction height [3].

The process of controlled caving by pouring crushed coal in the front of flexible canopies and on the armored face conveyor allows very high extraction capacity with provided safety. This procedure, however, requires a good knowledge of caving processes "in situ", advanced excavation equipment and in particular, knowledge and experienced miners. Successful control of caving and continuous collapsing of excavated area is one of the key stages of the technological process for the efficient and above all safe extraction of coal.

Velenje Coal Mine developed and upgraded longwall equipment to a large extent, themselves. Manufacturers of equipment like to cooperate, because of our experience and continuous development. We also produce certified hydraulic supports and in the greater part the armored face conveyor (AFC) and beam stage loader (BSL). The current standard set of longwall equipment for the lengths up to a ca 220 m consists of:

- Hydraulic support with a width of 1.75 m and a height of up to 4.7 m. The first four and the last three, i.e. drive sections, are located at crossroads with extended canopy to cover the front and rear drives of AFC. Working pressure is 310 bar. Section control is PMC-R and PM4 type.
- AFC JOY: conveyor: length of pan 1.75 m, width 0.9 m, chain 2 x 38 mm x 126 mm, drive 2 x 400 kW VSD motors, Q = max 1250 t/h, v = 1 m/s, planetary gearbox L 700 BP).
- Shearer Eickhoff SL 300: weight 50 t, total power of 845 kW, 3300 V).
- BSL JOY whit crusher (160 kW) and hydraulic breaker: (1 x 400 kW VSD motor, Q = max 1250 t/h, v = 1.08 m/s, lenght = 37.5 m).

In parallel with technological advance and increasing productivity, focus is mostly on safe coal extraction in terms of prevention and timely action in the event of a hazard. Identified potential hazards that accompany the coal mining process in Velenje Coal Mine and threaten the employees and the mine are: outbursts, exhausts, exhalations and inrushes of dangerous coal gasses; outbursts of water, sludge or mud; mine fire; methane explosion; coal dust explosion; explosion of explosives; power and telephony failure; rock bursts. Prevention and protection measures are integrated into the "Prevention and Protection Plan" of Velenje Coal Mine.

For many years in Velenje Coal Mine, we are working on measures that would eliminate as much as possible the dangers associated with sudden outbursts and inrushes of dangerous coal gases, methane (CH_4) and carbon dioxide (CO_2). The inrush of coal gases occurs most often at the technological process of coal extraction at the coal face. This means the sudden inrush of coal gases in the work area of coal face as consequence of sudden collapse of overburden strata in the extracted area (goaf). The gases are accumulating in the unventilated goaf area due gases exhalations from the coal. Also, due to the impact of mining especially when the coal face is crossing the coal layer faults or old works, the potential danger exists of an outbreak of coal gases in the area of coal face and gateways.

In case of outbursts or inrushes the coal gases permissible concentration limits can be exceeded up to several times and pose a great threat to the life of the workers present longwall area. An example of a sudden increase in CO_2 concentration is shown in Figure 1.

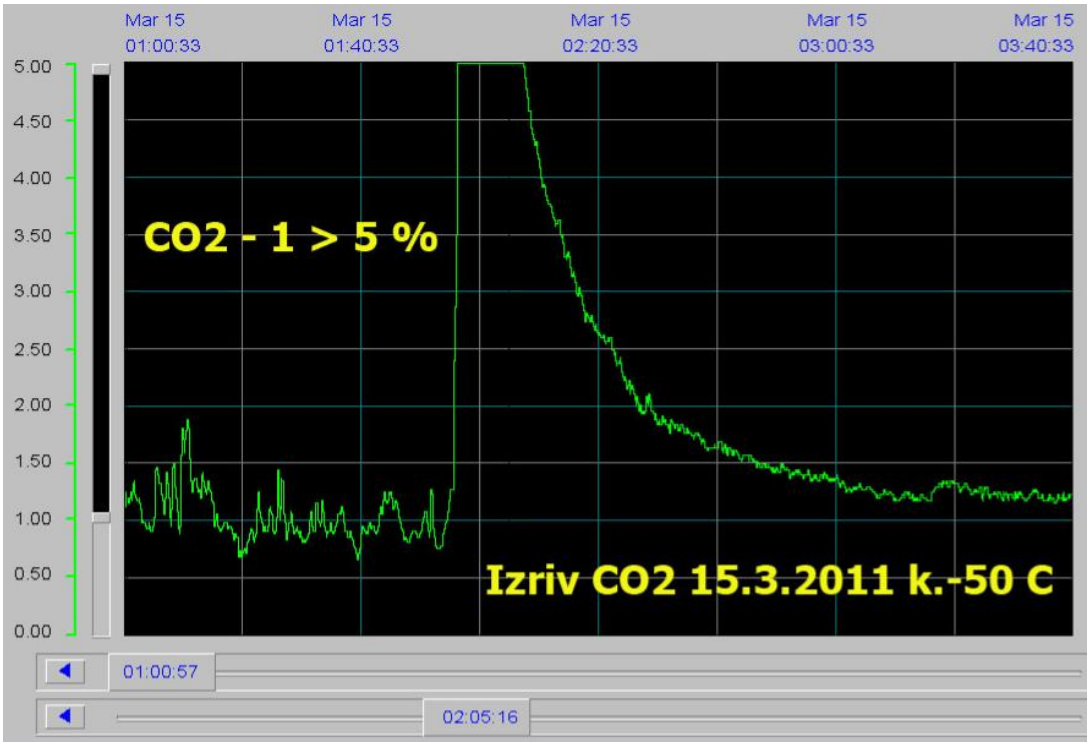


Figure 1: An example of a sudden increase in the CO_2 concentration at the face [4].

2. DEFENSE AGAINST THE THREAT OF INRUSHES AND OUTBURSTS OF DANEGROUS COAL GASES

Among the more successful prevention measures on CMV against the risk of inrushes and outbursts of CO_2 and CH_4 at the coal extraction, which are applied:

- regularly drilling in potentially endangered areas,
- excavation strictly after a technological cycle,
- consistently closing the excavated area (goaf area),
- consistently closing the old works after completed coal extraction,
- consistent maintenance of longwall and safety equipment,
- proper ventilation of longwall area,
- intensive control of gases concentrations in endangered areas by the mining supervisor of longwall face,
- continuous (real-time) monitoring of gases concentrations.

In the event of inrushes or outbursts of CO_2 and CH_4 (retreat from an endangered area), the protection measures are:

- Use of self-rescuing insulating apparatuses (every employee in the mine carries one on the belt),
- use of respirators that are directly connected to the compressed air network,
- alarming the inrushes or outbursts of CO_2 and CH_4 with the flash light system on the longwall and development faces and in return airways,
- auxiliary ventilation on the longwall face,
- automatic shutdown of electrical devices at increased methane concentration above the permitted limit (ATEX).

3. RESPIRATORS

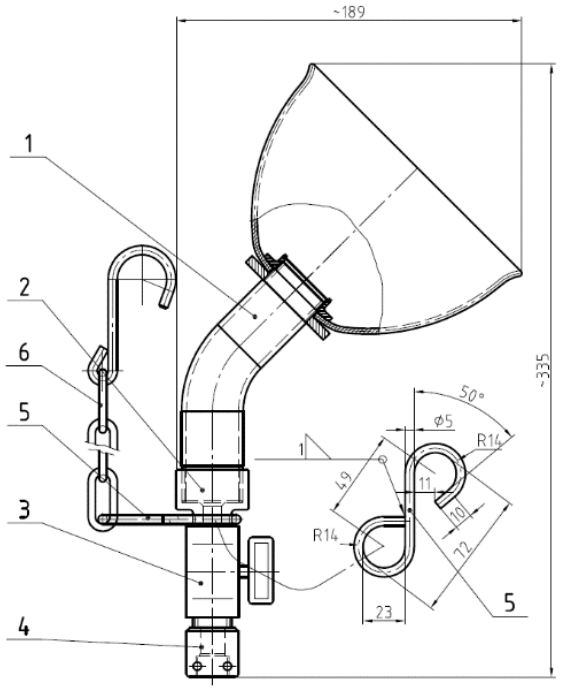
In order to achieve greater safety of people in mining operations, in 2003 we started using respirators.

The phenomenon of inrushes or outbursts of CO_2 and CH_4 on the worksite the occurrence of outbursts or outbreaks of dangerous coal gases is accompanied by an increased concentration of dangerous coal gases. This may also result in a change in the direction of the airflow and the possible temporary suspension of ventilation. The normal airflow direction and speed can be restored from a few seconds to a few minutes. However, the CO_2 and CH_4 concentrations can be lowered even within a few hours. With this phenomenon, workers have to retreat to a safe area as defined in the "Prevention and Protection Plan". At the event miners first approach and use the respirators whit inrush of fresh air, which allows miners to month the self-rescuing insulating apparatuses in a calm manner and retreat to a safe area. By the term respirator we mean a mechanical device that is connected to the compressed air network with a flexible tube and is installed on accessible places at the height of the head (Figure 2). In this way, we provide an independent source of fresh air from a compromised atmosphere.

The respirators are supplied from the compressor station on the surface. The compressor station is composed of six screw compressors manufactured by KAESER with a total capacity Q = 120 m^3/min, at a pressure of 7 bar. Compressors are commissioned in terms of air consumption. The compressor incorporation is automatic depending on the pressure in the pressure vessel. The mine network is connected with pipeline DN 200 through the shaft. The main mine pipelines are DN 125 and DN 100. Pipelines to the worksites are DN 80 [1]. Before connecting the respirators, a standby group is installed, where air condensate and oil droplets are eliminated. Pressure in the compressed air network is from 5 to 7 bar.

In the respirators are installed flow regulators in order to avoid excessive discharge. The respirator is designed so that the outlet air stream in the breathing mask creates an overpressure. Thus, the user is constantly using the uncontaminated air.

In case of inrushes or outbursts of hazardous gases, the supervisor, group leader or supervisor in the main control room of CMV issues a command "apparatus, retreat, gas" [2]. The miners keep their breath, they approach the respirators, open the airflow controller, and wait for a second to two to exhale settled dust and condensate in the air bag. They then approach the cheek and begin to breathe normally. With normal breathing, the self-rescuing insulating apparatuses can be installed in peace and without panic. This is followed by the retreat to a safe area as defined in the "Prevention and Protection Plan".



Legend:
1. Single respirator
2. Airflow regulator
3. Ball valve 1/2″´
4. Connection of flexible tube on network
5. Hanger of respirator
6. Chain with hook

Figure 2: Single respirator.

Locations of respirators

On the long wall faces, a single respirator is installed on each section of the hydraulic support. In the entering airway before the entering the face, four triplets of respirators are installed. In the return airway just after the face, three triplets of respirators are installed. The triplets of respirators are also located along the whole length of entry and return airways at every 18 m.

At the development faces, the triplets of respirators are installed:
- From the face to a distance of 10 m, two triplets of respirators.
- Followed by, at a distance of three times 6 m, 3 times one triplet of respirators and then
- along whole length of roadway, on triplet of respirators on each 18 m.

At drilling worksites, an additional set of triple respirators is installed on each side of the drill set. In the main exit roadway (main return airways), the triplets of respirators (Figure 3) are positioned at each 24 m.

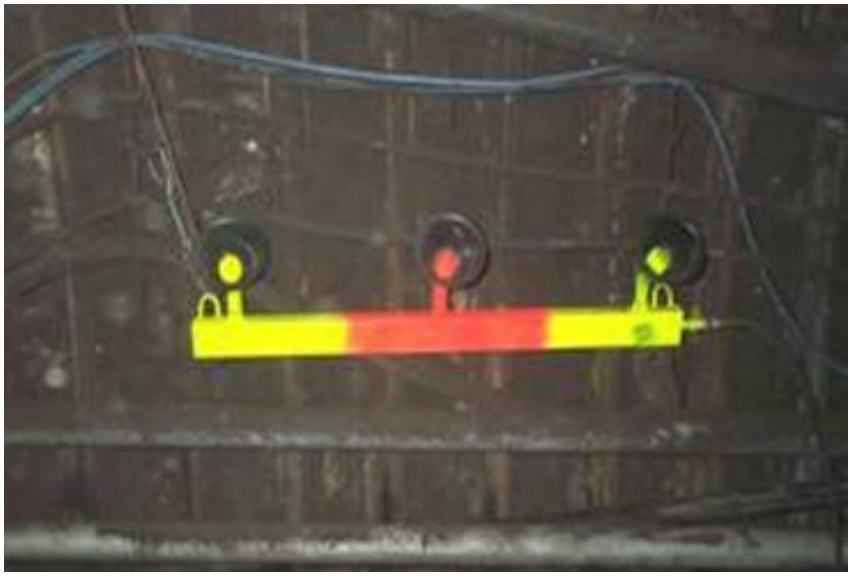


Figure 3: Example of installation of the triplets of respirators in the side of the roadway.

4. FLASHLIGHTS ALARM SYSTEM FOR INRUSHES AND OUTBURSTS OF COAL GASES

Although we have a very well-developed monitoring system for stationary continuous sensors and mobile measurements of dangerous coal gases, information on the occurrence of inrushes and outbursts of CO_2 and CH_4 can be identified too late for timely action.

In order to inform as quickly as possible the entire endangered area, in 2005, we developed the first system for alarming inrushes and outbursts of CO_2 and CH_4 with a flashlights system that draws attention to the danger with an acoustic and light signal.

First, we developed an alarm system with the TX 6831 type flashlights manufactured by Trolex based on a parallel dual system. In recent years, we have developed an even more efficient system with Woelke AGS 01 flash lamps, which is based on two-stage alarm level in one flash lamp [9].

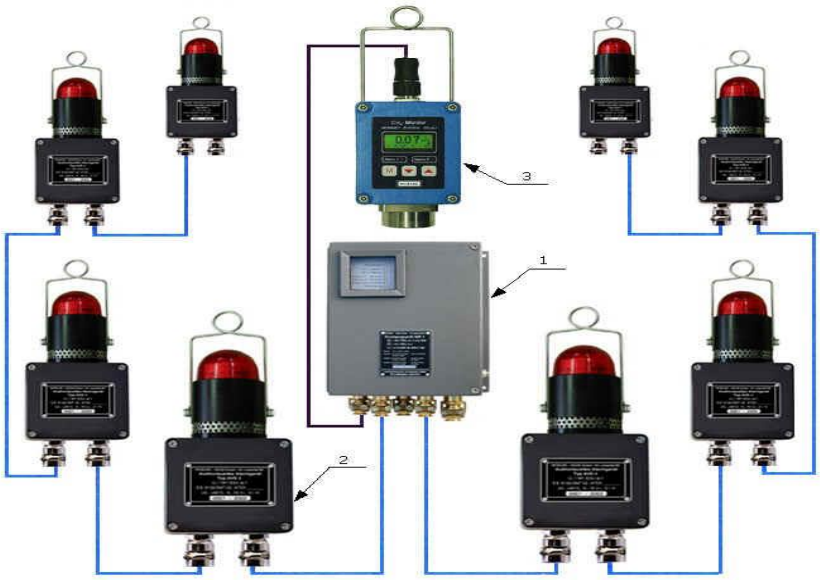
Configuration of flashlights system (Woelke, type AGS 01)

The Woelke flashlights, type AGS 01, has two alarm levels, each level being controlled via a separate input. The first alarm level (exceeded by the concentration of the monitoring gases above 1.5%) activates the red light signal with a flashing interval of 2 s, the second alarm level (exceeded concentration of accompaniment gases exceeding 3%) and triggers a light signal of the same color, but with a flashing interval 1 s, but also a continuous acoustic signal.

The second level of alarm is set so high that it can that we can undoubtedly separate the exhalation of coal gases due to technological process and actual inrushes and outbursts of coal gases that pose a life threat to the present miners. In the event of activating the 2nd level of alarm, the present miners know immediately that this is a dangerous event and they can take action without any prior warning from the supervisors in the Velenje Coal Mine. Action time is vital.

In the event that both levels are activated at the same time, the second level is preferred. In the terminal box, the flashlights also have two keys. The first key allows turning off the active alarm, and with the second key, anybody can check the operation of the device. If it works properly, both acoustic and light alarms are activated when pressed. For the transmission of a light signal (first level) and also an alarm (second level), the flashlights have built-in 10 super bright LEDs, which illuminate in all directions. For the transmission of an acoustic alarm, the flash has a built-in tone generator with a variable frequency from 2400 Hz to 2850 Hz.

Due to the simple design and operation of the two-level system, the employees are focused on lighting and audio signals and there is no need for continuous monitoring of stationary gas sensor. The Woelke AGS 01 configuration is implemented in accordance with the ATEX directive and is in line with the explosion protection Ex I (M1) M2 SYST EEx ib / ia I and is shown in Figure 4.



Legend:
1 Self-powered power supply type USV 4.2,
2 Signal-alarm device (flash lamp) AVS 4.1
3 gas sensor MONIMET type GMM 01.04

Figure 4: Flash lamp system configuration [11,12].

The flashlights are located on the longwall face, return airways of the face and main return airways. The locations are determined in the way that from each part of endangered are can be detected at least one flashlight. Maximum distance between flashlights is 150 m. The system allows separate activation of each flashlight with specific gas sensor in the area, which allows customized automation on a case-by-case basis [10].

5. SAFETY AND TECHNOLOGICAL INFORMATION SYSTEM AND REAL-TIME "PREVENTION AND PROTECTION PLAN"

Safety and technology information system (STIS) was built in 1990, which is considered as the year when the first microprocessor control device was installed in the Velenje Coal Mine . The information system covers the needs of the entire cave. Data logging is at the level of sensors or meters that measure different analogue and digital parameters. Most of these parameters are of a safety character, and some of them are for a technological monitoring and

control. The measured signals are sent to the processing stations, which process the signals and display them on the LCD displays, while at the same time the information is also transferred the surface, where they are further processed in the information centre [5].

The main purpose of transferring data to the surface is to carry out a centralized inspection of all the parameters relevant for solving the crisis situations in the mine. The data are from the processing stations in the mine to the communication computers, which further communicate with the main control computer in the information centre. The data are displayed in real time and archived for a longer period of time. Data are processed and displayed in various forms, such as: alarms, events, diagrams of current and historical values, process diagrams, map of the mine with the displayed sensors locations and real-time values of individual parameters [5,6,7,8].

The most important goal of using STIS data is to maintain the required level of safety in the mine and extraction process. The STIS data are the basis for informing employees about specialties in the mine and of detected dangers and endangered areas because of that danger. After such detection is immediately organizing their retreats from endangered areas to a safe area in the mine or directly to the surface. For a faster, timely and effective action in accordance with the "Prevention and Protection Plan" based on real-time CH_4, CO_2 and CO values and ventilation mine maps. A real-time display (example on Figure 5) based on processed increased gases values enables the display and alarming of endangered areas. Based on two-level arming system with flashlights is possible to detect and confirm separately gases exhalation and inrushes or outbursts of CH_4, CO_2 with display of endangered area. Also based on carbon monoxide (CO) sensors a display of potentially hazardous areas by a mine fire is possible [6,8].

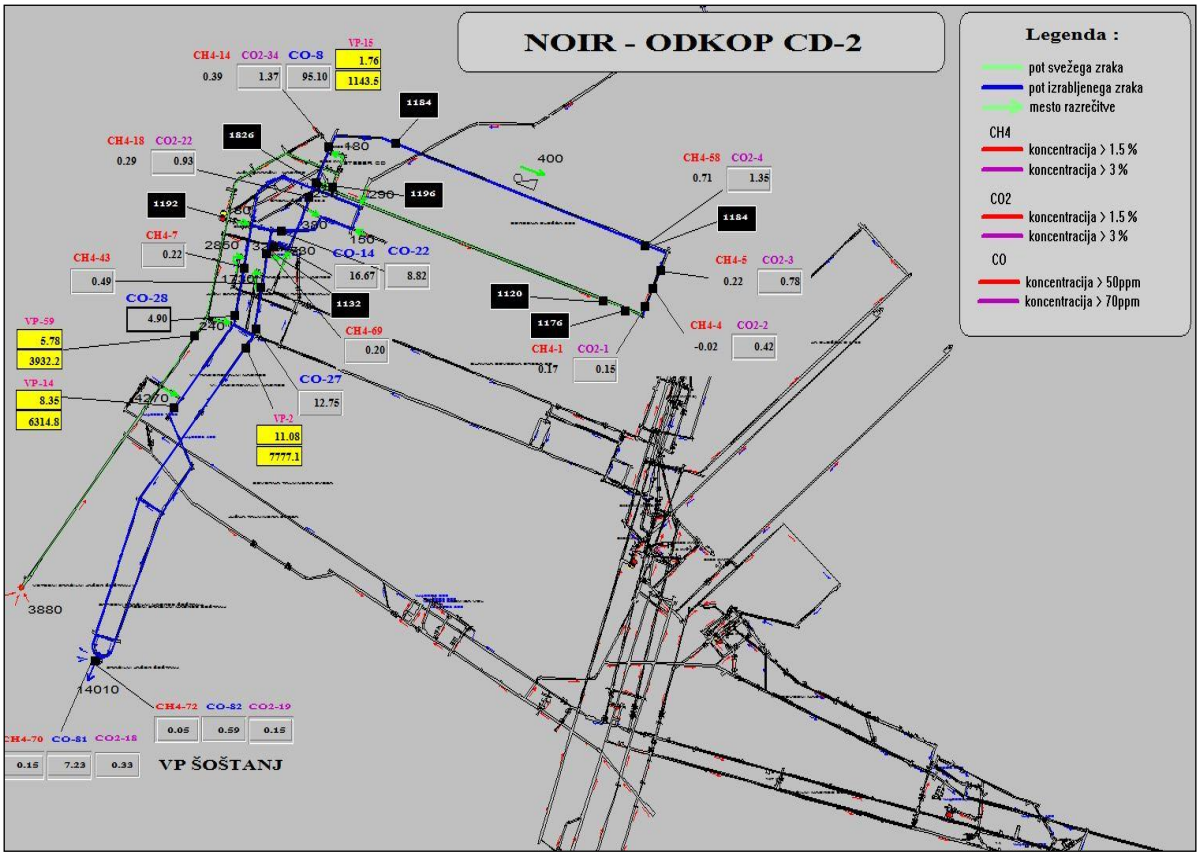


Figure 5: Display of real-time Prevention and Protection Plan [11, 12].

CONCLUSION

Ensuring safety and health of miners is a main priority in Velenje Coal Mine . Sudden inrushes and outbursts of methane and carbon dioxide at coal extraction represent extremely hazardous events that cannot be foreseen or predicted in advance.

An important role in ensuring occupational safety and health at sudden inrushes and outbursts is additional measure, the implementation and use of the respirators in Velenje Coal Mine . These allow the safe breathing of fresh air in the event of sudden inrushes and outbursts of dangerous coal gases and a peaceful and without panic mounting of an self-rescuing insulating apparatus. This is essential for safe retreat in to the safe area as defined in the Prevention and Protection Plan. The supply of the air is directly from the compressed air network which is supplied from the compressor station on the surface.

In order to protect the miners effectively, timely information on the increase of coal gases concentration and the alarming of danger in case of inrushes an outbursts is decisive and vital. Successful detection, reporting and alarming of these dangerous phenomena with the described systems gives timely possibilities for implementing the necessary technical and security measures.

In Velenje Coal Mine, a two-level system with flashlights for the alarming of dangerous concentrations of methane in carbon dioxide allows the immediate detection of increased concentrations at inrushes and outbursts at the coal extraction process.

Real-time data acquisition via sensors and meters are processed through a safety and technology information system that enables the preparation of graphical bases for displaying and remote monitoring of technological processes and the most important, real-time display of endangered areas, which enable immediate action in the coal mine in accordance with the procedures defined in the Prevention and Protection Plan of Velenje Coal Mine .

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DOI: 10.7251/BMC170701233G

USE OF EQUILIBRIUM WATER QUALITY MODELING FOR ARD/ML RISK REDUCTION

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ABSTRACT

Equilibrium water quality models have been used extensively for regulatory purposes, however, mine operators have tended to see such models as primarily related to permit aquisition. A well constructed and calibrated equilibrium water quality model is, however, invaluable for risk management, both during mining operations and following mine closure. Case historys are presented from mines in Canada and Asia. The methodology of utilizing equilibium water quality modeling for gold, copper, nickel and coal (lignite) mines, for waste rock dumps, tailings storage facilities and pit sump/pit lakes is presented. The initial equilibrium water quality model provides a forward-looking estimate of expected water quality at various locations in the mining process. As mining commences, divergence from the predicted water quality may be due to either an inadequate/incomplete model or to unexpected geological or geochemical conditions. Ongoing updating of the water quality model may therefore be invaluable in risk management.

Key words: Risk; water quality; modeling; geochemistry; management

1. INTRODUCTION

Mine water quality models (MWQM) are a requirement of many permitting regulators and funding institutions, in order to provide assurance that the proponent has carefully considered the potential environmental impact of their proposed project [1]. This paper argues, using examples from Canada and Turkey, that MWQM also provide a valuable resource to the proponent, not just in obtaining permitting, but also in risk management during mining operations. Furthermore, MWQM and environmental data may also be used as an equally valuable resource in exploration in similar terrain to the proposed mining project [2].

MINE WATER QUALITY MODELS

Perhaps the most important factor to take into account in mining risk management is that any mine is a complex interactive process, involving engineering, process control, geochemistry, hydrogeology, and environmental issues. Simple statistics often cannot describe the complexity of such interactive processes, thus requiring the use of an MWQM. A comprehensive MWQM should therefore address the following:

- Mine construction and production data;
- Climate (rainfall and evaporation);
- Mineralogy of waste rocks and ore;
- Drainage (surface and groundwater) and extraction (for process water);
- Kinetic and static testing data;
- Mine plan;
- Mitigation strategies;
- Hydrogeology;
- Precipitation and/or dissolution in different pH/Eh environments; and
- Adsorption (if applicable).

In order to achieve this the model is usually split into two portions: mass balance modeling, and; equilibrium modeling. The former may be achieved using spreadsheets (such as EXCEL) while the latter is achieved using specially written software such as PHREEQC or GEOCHEMIST'S WORKBENCH. Each part of the model feeds back into the other in an iterative cycle as the life of the mine is modelled. A flow diagram of a typical MWQM is presented as Figure 1. The MWQM may be further improved by calibration, in which the estimated output data (e.g. parameter concentrations in effluent) is adjusted to account for actual monitoring data. Valuable output from a well- designed MWQM include:

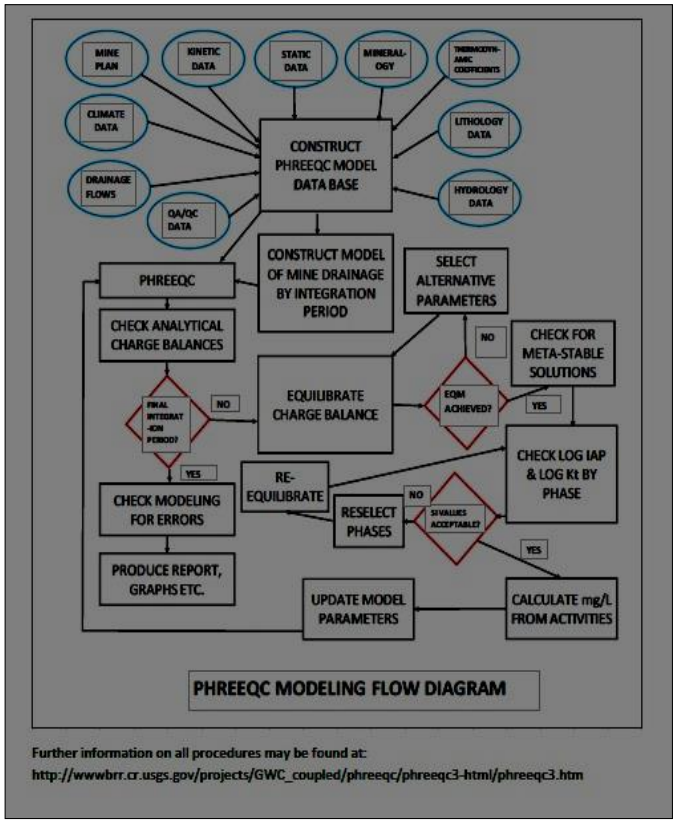


Figure 1: MWQM flow diagram

Parameter concentrations in pore waters (e.g. within a waste rock dump or in overburden);

- Efficiency and optimization of mitigation strategies;
- Flow rates and drainage times;
- Parameter concentrations in mine effluent and comparison with regulator's guidelines;
- Changes to porosity and permeability due to precipitation or dissolution of mineral phases; and
- Interactions between seeping mine water and underlying aquifers.

As the mining operations commence the estimated output data may be compared with data obtained by monitoring and provide the mine operator with an early alert for unexpected outcomes. Changes to the mine plan (e.g. extensions to waste rock dumps, extension to an open pit) can be modelled using the MWQM and provide a risk assessment as the mining process continues. MWQM are equally valuable in both surface and underground operations. The use of MWQMs in three Canadian (Fox Lake Mine, Kemess South Mine, and Bucko Lake Nickel Mine) and three Turkish mines (Balya Mine, Hanonu Copper Mine, and Kestanelik Gold Mine) are compared and contrasted below in order to show potential risks incurred without the use of MWQM and the potential benefits of their use.

2. FOX LAKE MINE, MB, CANADA

The Fox lake Mine began operations in 1969 and was operated until 1985, producing Zn, Cu, Au and Ag [3]. The mine is currently under care and maintenance; the principal activity being treatment of acidic and highly contaminated effluent. No MWQM was prepared prior to or during operations, although MWQM were being undertaken by the late 1970's, and the tailings spread indiscriminately across a large area causing extreme acidity and high As, Se and base metal contamination in five lakes. A MWQM was finally prepared in 2009 as a precursor to proposed mine remediation. The total cost of remediation was estimated at between 60 and 80 million $CAD. It was decided by the private owner of a large portion of the mine to not begin remediation. The ongoing cost of effluent treatment ranges from 500,000 to 1,500,000 $CAD per year and the cost of remediation continues to rise.

3. BALYA MINE, TURKEY

The Balya Pb-Zn mine was initially operated by a French company in the late 19th century, although it is now operated in a care and maintenance manner with a limited amount of exploration work. The mine tailings and waste rock dump generated extensive ARD/ML [4], a situation that persists to date. A remediation plan was developed by an EU funded project in 2013, although the results, including estimated remediation costs, were not published. To the authors' knowledge no MQWM was ever carried out. The environmental impact has been exacerbated by the local municipality's use of mine waste for filling road potholes; spreading the contamination even further.

4. KEMESS SOUTH MINE, BC, CANADA

The Kemess open-pit, copper/gold mine closed in 2011, although the mine is projected to re-open as an underground operation in 2018. High Se leaching from waste rock was identified as a design parameter early on in the design of the open pit facilities. Accordingly, a calibrated MWQM was constructed (Klohn-Crippen Berger, pers. comm.) to facilitate dilution of the leached Se below regulatory limits. The MWQM was updated through the life of the mine as operational changes were made, to allow ongoing estimation of the dilution requirements. The MWQM provided an inexpensive tool for Se mitigation (compared to mine water treatment) and was successful in maintaining Se concentrations below regulatory limits.

5. BUCKO LAKE NICKEL MINE, MB, CANADA

The Bucko Lake underground nickel mine was brought into production in 2009 by Crow flight Minerals. A MWQM was constructed to facilitate permitting [5]. A ring dyke was constructed to hold the first several years of tailings, however, early in the life of the impoundment significant seepage was observed at the toe of the impoundment. It was uncertain whether the observed leachate originated from within the tailings storage facility or was due to seepage from exposed rocks travelling along the interface between bedrock and overburden. The MWQM was used to speedily resolve the issue as the signature of the leachate proved that it originated from within the ring dyke. Subsequent investigation showed that cracks had developed in the bentonite seal within the ring dyke. The ring dyke

seal was repaired the following spring and the mine was subsequently brought back into production.

6. HANONU COPPER MINE, KASTAMONU, TURKEY

The Hanonu open pit copper mine commissioned a MWQM at the start-up phase of construction for permitting reasons, Geochemico, 2017. The MWQM included PHREEQC modelling of the open pit and pit lake, waste rock dump (WRD) and the tailings storage facility (TSF). Although the MWQM was commissioned for permitting it was subsequently used to optimize the operational characteristics of the open pit, especially with regard to closure, the WRD cover design and to minimize environmental impact from potential TSF leachate.

7. DISCUSSION

Case histories from two mines, one in Canada, which elected to not commission a MWQM and one in Turkey, which pre-dated the use of MWQM, clearly show that the absence of a MWQM opens the mining operation to extreme risks. In both cases an environmental catastrophe was created that continues to burden stakeholders with a huge financial burden for remediation. It is helpful to enquire why this is the case. The geochemistry of rock/water, soil/water and tailings/water interactions are extremely complex. This complexity makes it very difficult for mining geologists and engineers to relate the complex chemistry to expected mining operations. The MWQM provides a bridge between the complex chemistry and the detailed mining operations, facilitating mining geologists and engineers to understand, in their own fields of expertise, the implications of the geochemistry to operational risk. A MWQM therefore helps the miner to understand and then mitigate operational risk.

Secondly, a well-constructed MWQM provides an objective tool with which design criteria may be optimized. At the Kemess mine facilitated a low-cost and effective method for mitigating high Se leachate. At the Bucko Lake mine a potentially difficult seepage issue was diagnosed rapidly using the MWQM and the unexpected cracks in the TSF liner were dealt with quickly, minimizing the loss of production revenues caused by mine downtime. At the Hanonu mine, the MWQM facilitated objective optimization of the open pit closure scenario, cover design for the WRD and showed that the TSF was extremely unlikely to cause environmental impact as the synthetic liner aged. Lastly, a MWQM may be used to explore under cover, within the mine footprint, for ore body extensions. The exploration potential of MWQMs are discussed below.

8. MWQM IN EXPLORATION

MWQMs usually relate mine facilities such as waste rock dumps, open pits and tailings storage facilities to potential environmental impact (Figure 2). The MWQM is used to predict location and concentration of environmental impact. In exploration modelling the source of elements is the mineral deposit, however, very similar processes of weathering cause geochemical anomalies to form in the water, soils and vegetation in the environment. An MWQM and data collected for environmental purposes may therefore be used for exploration purposes, either in the vicinity of a known deposit, or for a similar type of deposit elsewhere.

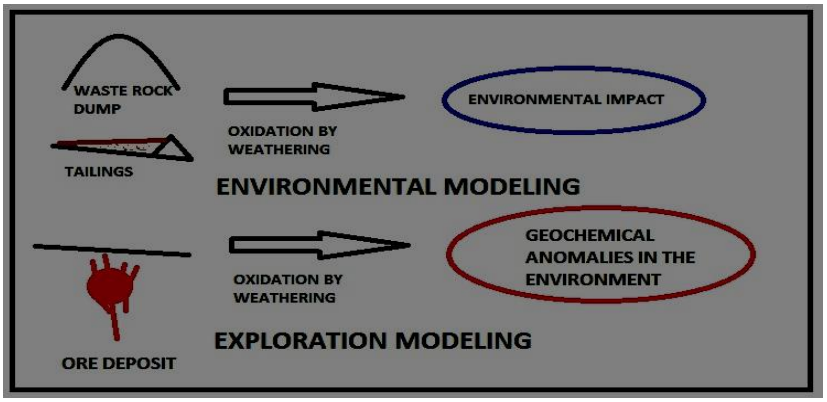


Figure 2. Relationship between MWQMs for enveiromental and Exploration purposes

9. KESTANELIK GOLD MINE, TURKEY

The Kestanelik open pit gold mine, located near Cannakale, Turkey [6] is currently under construction. A MWQM was commissioned, in several stages, with the principal purpose to obtain permitting. The MWQM was subsequently used to examine the groundwater multi-element saturation indices signature [7]. A significant extension of the ore body was subsequently located under thick colluvial deposits, and also, potentially under basalt cover to the south, using only data collected for environmental purposes.

Additional exploration surveys of this nature are currently underway in Serbia (ERM, personal communication).

10. CONCLUSIONS

Consideration of six case histories of mines located in Canada and Turkey that the use of MQWM extends far beyond simply complying with regulatory permitting requirements. MWQM may also be used, to significant effect in:

- Optimizing design of mitigation strategies for mobile elements such as As and Se;
- Forensic studies of unexpected seepage(s);
- Optimization of waste rock cover design, tailings storage facility engineering design etc.;
- Optimization of remediation strategies; and
- Exploration studies within the mine site or for similar mineral deposit types.

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DOI: 10.7251/BMC170701239M

ADOPTIONS OF LEGISLATION AND MINING POLICIES TO FACE THE GLOBAL CHALLENGES IN MINING ACTIVITIES

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ABSTRACT

This article gives a review of the adoption of legislation and mining strategy of Albania towards the global mining policies as a continuous process, evaluation of all its components, territorial planning, re-evaluation of mineral reserves, specific mineral strategies, partnered state-private business, efficiency of legal acts, obligations of mining institutions, national and regional employment policy and environmental issues, as well as the social aspects in mining areas, facing new problems in mining activities in Albania, related it with the mining activity in the Balkan region, European countries and globally, to ensure a sustainable and long term development through effective investments.

The long term strategy of minerals of Albania carried out under political and institutional reforms toward integration of the country in the EC structures will serve the politicians to look at mining industry as a profit source for the country, for employment and sustainable development of the economy.

In the article is presented development in mining activity, new concepts for mining activities connected with the adoption of our strategies with European strategy for minerals, challenges, trade and requests for minerals, ways of information of government organisms, stronger focus on the supervision and monitoring of existing (and future) contracts, transparency to the public, transparency on decision making, involvement of the local authorities, good governance, fight the corruption, perfection of institutional structures of mining service and legislation which will create a clear vision for the role of mining industry in the future with slogan "Minerals and Energy for a sustainable development".In order to offer such mineral resources by creating profit activities, maximum profit to citizens, fight the poverty especially in rural areas and create employment.

1. INTRODUCTION

Mineral deposits are anomalies in the earth crust, geographically fixed and not renewable. Furthermore, mineral deposits have a finite size. In the case of some minerals and in particular metal ores the size of the exploitable deposit is often determined by economic factors such as metal prices and cost of production. Deposits, which are not economic to work under current circumstances, may become economic in future. Protection of mineral deposits is a long-term investment within the context of sustainability. Some universal considerations for sustainable development are: maintaining environmental capital; preventing degradation of land, air and

water quality; efficient production and use of natural resources; decreased rate of emissions of greenhouse gases; increased recycling; and prevention, minimization and safe disposal of wastes.

The mining industry has been a key to the development of civilization, underpinning the iron and bronze ages, the industrial revolution and the infrastructure of today's information age. The mining industry produces over 6 billion tons of raw products valued at several trillion dollars. Downstream beneficiation and minerals processing of these raw materials adds further value as raw materials and products are created to serve all aspects of industry and commerce worldwide. The last decade of the twentieth century saw the creation of mega-commodity corporations that increasingly moved downstream into the beneficiation area, leaving exploration for new mineral deposits increasingly to small junior mining companies. Application of new technology has led to productivity gains across the value chain.

Apart from Antarctica (which has a treaty in place preventing short to medium term exploitation and exploration of minerals), mining takes place in all of the world's continents. Traditional mining countries such as the USA, Canada, Australia, South Africa and Chile dominate the global mining scene. These countries have become the traditional leaders in mining and exploration methods and technology. Exploration and development funding has changed over the past few years with emphasis shifting to areas that have been poorly explored or have had poor access for reasons of politics, infrastructure or legislation. Gold, base metal, diamonds, platinum group elements (PGE's) and rare earth group are the more important commodities explored for and developed globally.

The need to maintain a secure, sustainable supply of minerals makes mining activities one of the important sectors for investments during the last decade. The future mineral supply is a grand challenge. Taking in consideration development in the world mining sector, economic growth in the highly populated countries such as Brazil, Russia, India, China (BRIC phenomena) and others emerging countries over the past two decades, the large quantities of minerals required by the industries, geographic location of mineral deposits, market distortions in mining sector, limitation on exploration and extraction facing increased competition for different land uses and a highly regulated environment, it is foreseeable that the demand for minerals will grow significantly and could put pressure on the supply with minerals especially for metal ores which indeed is already happening, (*shall be mentioned here that the EU is highly dependent on imports of metallic minerals, as its domestic production is limited to about 3% of world production and its consumes is more than 25 %*).

Albania is a country with an important mineral wealth on the Balkan region and is looking at mining industry as a profit source for the country, big source of employment and sustainable development of the economy. Chrome, Copper, Nickel, Titan-magnetite are some of the main riches of Albania. During last 50 years Albania has developed mainly mining activities in the four main minerals, chromite, copper, iron-nickel and coal.

Actually its mining sector is structured in a different way and the traditional minerals covers in quantity only 10-15% of production but still more than 70-80% of the revenues. Adoption of new legislation rules and new policies still need to be improved in order to be comprehensive in time with the global mining development and its challenges.

2. HISTORY OF MINING ACTIVITY IN ALBANIA

Development of mining industry in Albania has passed through four stages: *The first stage* covers ancient times until the beginning of the 20th century. The traces of minerals exploitation are found in Mirdita area written as well in the famous legend of the golden ram skin, which has to do with a technological secret for the benefit of gold, exploitation and processing of copper during the Bronze and Iron Age in the northern provinces (Mirdita, Puka, Porava, Iballë), and in the writings of Aristotle, Herodotus, Plutarch, Strabo, Pliny, for data of mineral activities, as well there are mentioned by Romans for the bitumen in the hills of Selenice, etc.

During Ottoman occupation, our land has shown interest in the use of metal minerals, as evidenced by the decree of the Sultan for the use of the Vllahna site (in Has region). At the end of 19th century the exploitation of mineral resources was introduced into the legal course by Sultan decree (1862) regulating mining activity. Classification of minerals, royalty as percentage of the ore sold value and concept of granting concessions for a period of 40 years up to 99 years was set in this decree, which was amended in 1870. *The second stage* includes the period up to the end of World War II. It is considered as the period when the mining activity begins documented. Many distinguished foreign geologists have compiled different maps, published monographs, scientific articles, etc. In 1903 Vinassa De Regny has published in Vienna the first geological map at 1: 200 000 scale, for the northern regions of Albania. In 1912 were published in Rome the geological map of Vlora region at 1: 200000 by A. Martelli. This period is marked by two important events, in 1922 was compiled the first Geological Map of Albania (J. Bourcart built the Geological Map of Southeast Albania on a scale of 1: 200,000, published in Paris in 1922, with the monograph "Borders of France administered by France"), the first of its kind in the Balkans, whereas year 1929, signs the endorsement of the Law On Mining of the Albanian Kingdom, which paved the way to the exploration and exploitation of mineral resources. *The third stage* covers period 1944-1994, it marks the period when the mining activity was organized in state-owned enterprises characterized from a wide range of development of mining industry. Science of geology, mining and processing was developed with the same rhythm likes the extraction mining activity. Mineral exploration, exploitation and processing constitutes a key component of the Albanian economy, mining became a solid foundation to the economy generating substantial revenues, activities in chrome, copper, ferronickel and coal have been developed in intensive way, Albania in the 1970s and 80s, was the world's 3rd largest producer of chromites. *The fourth stage* covers after 1994, enabled the transition from an economically centralized type of operation, into one based on the free market. Although the production of the mining industry almost collapsed when the country shifted to a market economy in 1990s, the mining industry has been able to continue its road and to realize a sustainable development under privatization and market-oriented economic reform, defining performance goal, critical points (restriction and difficulty which arise in the course of action) taking countermeasures to overcome with priority. Different reforms have been undertaken in restructuring, privatization, legislation, regulatory framework, institutional strengthening, inspection, fiscal and electronic government. The first Mining Law of Albania is approved on 1994. The new legislative package for mining activities is approved during 2010-2011 based on the new mining law Nr.10 304, date 15.7.2010 "For Mining Sector in the Republic of Albania".

3. BASIS OF THE ALBANIAN MINING STRATEGY

The core principle of sustainable development is to improve human well-being and to sustain those improvements over time. There is no one, correct view of what sustainability means or how its principles should be implemented. Sustainable development has four overarching goals: economic prosperity, environmental health, social equity for the present generation and equal opportunities for future generations.

The question of the relationship between economic development and environmental protection is one of the most complex and difficult to resolve in describing the content and scope of the concept of sustainable development. The difficulty arises from the fact that economic development and the impacts on the environment are interlinked.

In this equation adaption of strategies for the mining sector is more complicated due to two important factors: Firstly mining is global and its development trend and prices totally depends from the global market where Albania as well as Balkan countries plays an insignificant role. Secondly still Albania and Balkan countries even that have a wide range of minerals have small mineral potential for big investments due to their size of ore bodies (which varies from small up to medium size, exception are only a few deposits in polimetals, nickel, copper). But Balkan countries can play an important role regionally for Europe, where mining especially for metals is almost at zero level.

The sustainable development on Albanian mining policy is treated in legislation programs and action plans as a balance on three pillars:

- Investment development **through** better legal frame work and regulation, to reduce the administrative barriers, finishing the process of privatization, promotion of industrial business with higher value added, strengthening of the institutions dealing with business, ensuring well-management of natural and human resources**, by** inciting, supporting, encourage the development and increment of domestic production, education and training for entrepreneurship, coordination and cooperation of the developing programs of the country with those regional and more, improvement of the climate for foreign investments, saving the professionalism and transparency **for** a sustainable development.
- Investment development friendly to the environment, **through** enforcing the mitigation measures, rehabilitation process, strengthening of the supervision on remediation, monitoring and post monitoring process; ensure level of work safety, less damages and pollution to environment, **by** application and implementation of advanced technologies, **for** a clean environment and nature, to minimize as much as possible the effects on life of communities.
- Social aspects, **through** the involvement and understanding of community, transparency, civil society involvement in a wider way, taking in consideration profits and damages to the community as an approach to fight poverty, **by** encouraging the creation of new jobs, possibilities for infrastructure improvement, help in health system and education particularly for the areas with poor economic activity, **for** a better life of the citizens.

4. BASIS OF THE ALBANIAN MINING LEGISLATION

The first Mining Law of Albania is approved on 1994, the new legislative package approved in 2010-2011 based on the law Nr.10 304, date 15.7.2010 "For Mining Sector in the Republic of Albania" update the old one in some issues concerning territory planning (including the concept of mining strategy and programming of mining activities by offering mining areas on line through a digital map on GIS and Data Base of Mining Industry, through the competition process and a transparent manner for granting of mining rights, facilitating licensing procedures (concept of one stop shop).

This law includes new provisions on minerals promotion, monitoring, supervision, monitoring of post mining activities, mining closure, closure of abandoned mines, conservation, rehabilitation by strengthening of the responsible institutions, strengthening the rules on health and safety in mining activity, addressing of environment and social issues and the concept of financial surety for environment rehabilitation, realization of the minimum working program for prospecting exploration permits and realization of investment program.

The new legislation intend to adopt mining policies in the direction that extraction activities, cab became a supporting pillar for the economy by attracting big investments in the sector, increasing investor interest. This new policies also intends to:

- Create an investment environment in which appropriate international investorscan start the production and bring the sector to its economic potential,
- Stimulate the transfer of technologies and knowledge,
- Promote and sustain the value added policy for the mining sector on treatment of minerals by implementation of new technologies, increase the value of minerals, profit and employment in the country,
- Promote the development of modern infrastructure and generate valuable foreign exchange earnings,
- Make mining sector a core industry being able to stimulate sustainable economic development, help creation of jobs, reduce poverty, and become a major contributor to the macro-economy and local community development.

Its aim also for:

Legislation and Policy

- Sustainable development of Albanian economy based on short-mid-long terms objectives for maximum profit and a better life of Albanian citizens;
- Realization of a robust legal framework that is essential to the development of the mining sector, with the alignment of national legislation to EU legislation and standards of mining activity, transparency, environmental protection, health and safety management;
- Creation of competitive legal and fiscal conditions to attract and retain private sector investors; attract qualified investors through an appropriate licensing methodology;
- Improving the Investment Climate through Reforming the Policy, Legal and Regulatory Frameworks, including Enhanced Administrative Capacity and Sector Promotion,

guarantee foreign and domestic investors to meet their investment targets profitable and competitive products;
- Encourage treatment activities as a value added policy for the mining sector through fiscal policies;
- Increased investment in exploration and development;
- Prioritize unaddressed environmental / social legacy issues to define a strategic long-range program through the preparation and implementation of a Strategic Environmental and Social Assessment;
- Further improving in the mining licensing requirements to encourage private sector companies to invest in the mining sector. Defining the framework for the tendering of concessions. Implementing of a high standard web site with information on future tenders, on the tendering and licensing procedures;
- Improving productivity and competitiveness, product internationalization, foreign investment promotion and the better use of financial, human and natural resources;
- Increasing the value from mineral resources for the economy from non-renewable resources to sustainable development;
- Offering alternatives based on mining activity diversity, coordinate and cooperate with sectorial strategies and programs of the country with those regional, EC and more;
- Improvement on economical and financial balance of the country by development of exports;
- Maintain a balance of benefits derived from the mining sector among various regions and population groups in the country through increasing revenues of the local authorities by division of royalty to central and local authorities and other taxes;

Strengthening of mining institutions

- Clarifying roles and responsibilities of Government institutions in the mining sector;
- Confirm in the day-to-day operation the new role of government as regulator instead of direct investor and operator of mines;
- Improving sector administration of exploration and development; Clear-cut Terms of Reference to avoid both institutional overlap and gaps appearing in the minerals administration;
- Facilitate access for investors in Albanian Mining sector through "one stop shop"; Strengthening enforcement of mining polices, laws and regulations;

Planning

- Territorial planning, land use strategy;
- Planning integrated systems, particularly for chromites (mining, concentration, smelting), and preparation of further tenders related to chromites mining and processing. Improve the organization of small-scale mining companies, monitoring safety practices, and planning the formation of co-operatives;
- Re-evaluation of mining resources, collecting basic geological data at regional level (e.g. airborne geophysical surveys, etc) that would help to find prospective areas and would

attract private companies to invest in exploration activities (particularly for copper);
- Development of plans for nickel treatment plant, including the optimum type of facility, location and size, so that tendering of build and operation can be planned;
- Developing GIS database and cooperation with relevant institution of the EC, Enhance the scientific knowledge of the basic geology and environment to support detailed exploration by the private sector;
- Transform national geological database into an electronic format and make publicly accessible on an integrated basis via a dedicated website; Other forms of sector promotion (international exhibitions, features in respective magazines, websites);
- Specifying requirements for the minimum activities that companies must carry out under their license conditions and the requirements of data obtained during exploration;

Transparency

- Fulfill the obligation related to implementation of EITI, improving transparency and accountability;
- Transparency to the public, transparency on decision making, Participation of communities;
- Sustainable long-range sector development in which a broad stakeholder group participates in resource development planning and sharing of benefits; this also requires transparency with regard to payments between the companies and the Government;
- Good governance, continuity of reforms, fights the poverty especially in rural areas, fight the corruption, and support a professional and efficient organization of public institutions. Ensure that decisions are made in a transparent and efficacious manner;

Environment

- Development of mining activities using technologies to be environment friendly by evaluation of the environment effects from mining technologies, exploitation system, optimal technologies in treatment of minerals, recycling, control of dust, gas emission and leaching, discharge control for the water qualities, ground contamination form wastes and chemical and hazardous substances;
- Improve the Development Outcome of the Mineral Operations through Addressing Environmental and Social Issues, as well as Improving Sustainability at the Community Level and Benefit Sharing;
- Provide for the environmentally and socially acceptable closure of non-viable operations, progressive rehabilitation;
- Waste management in accordance with EC-directives of mining waste;
- Panorama and Nature protection, protection of national parks, flora and fauna and cultural heritage;
- Sensibilities of the local authorities for mining risk to avoid the construction in the areas influenced from mining activities.

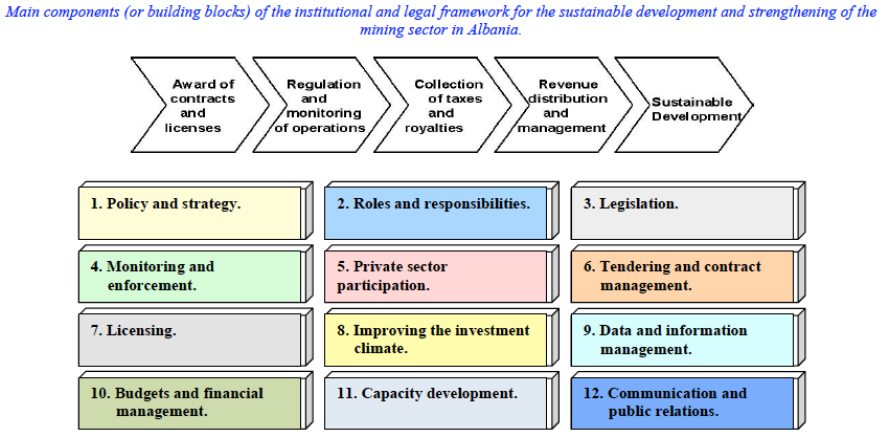


Figure 1.

5. EC POLICIES FOR MINERALS.

European Commission sets out targeted measures to secure and improve access to raw materials for the EU based on three pillars:

- Ensure access to raw materials from international markets under the same conditions as other industrial competitors;
- Set the right framework conditions within the EU in order to foster sustainable supply from European sources;
- And boost overall resource efficiency and promote recycling to reduce the EU's consumption of primary raw materials and decrease the relative import dependence.

The Commission lunched this initiative due to:

- Analysis of the mineral activities and issues rose up from this activity in the European countries,
- Views across Europe considering mineral planning policies and practices as an increasingly critical issue,
- Considerations of the various components of national land use planning priorities,
- Environmental protection and societal benefits issues,
- Needs for minerals and diversified sources,
- Challenges in this sector,
- Increasing transparency.

European Commission in order to develop measures aiming at the sustainable development of the extractive industries and to promote the exchange of best practices with the candidate countries, which have to rely on these sectors for their economic development, with this draft shows that all the candidate countries need to take in consideration such visions on planning for mining sector.

It is important for Albania as a country which inspires to be part of EC, also for the other Balkan countries also inspiring to be part of EU, to adopt our mining policy in accordance with the European policies for mining sector development but at the same time to reach in a win-win situation for both parties.

Apart of that, it is important at this stage that this strategy shall include also the Balkan countries which prospered to be part of the European Union, which in some commodities have half of the production of all European countries and more for the metals.

The EU is self-sufficient in construction minerals**,** in particular aggregates. Non-energy extractive industry of the EU27 generatea turnover of about 50 billion € and provided employment to about 300,000 people. The extractive industry is a vitalsupplier of raw materials to major downstream industries, such as road and construction, chemicals, automotive, aerospace, machinery and equipment sectors, that provide a total added value of 1,5trillion € and employment to about 30 million people.

The future mineral supply is a grand challenge for society at large. However is unthinkable a future society without minerals. Some of the measures that EC evaluate to be taken in consideration on this respect include:

- Define critical raw materials;
- Access to land by promoting sustainable access to raw materials in development policy, and improving the regulatory framework related to land access;
- Promote a sound investment climate and sustainable management of raw materials by improving social and environmental standards, human rights conditions, combating child labor,
- Improving the business environment
- Strengthening the Single Market by Improving market surveillance and Creating a market for business services
- Industrial innovation strategy by more timely development and commercialization of technologies, innovation in traditional manufacturing, E-skills for advanced users, encouraging clusters and better cross-border co-operation
- Management of EU strategic partnerships by initiate policy dialogues with third countries and promote enhanced international cooperation; including provisions on raw materials in trade agreements;
- Identify and challenge trade distortion measures taken by third countries using all available mechanisms and instruments prioritizing those which undermine open international markets to the disadvantage of the EU;
- Assist developing countries with capacity building to manage public finances, sustainable management of natural resources; negotiations with mining companies by developing transparency of mining deals and taxation systems, increase the use of EU budget support on this direction;
- Improved knowledge base and R&D to improve methods for exploration, extraction and recycling;
- Promote the exchange of best practices in the area of land use planning and administrative conditions for exploration and extraction;
- Increase the involvement of national geological surveys in land use planning within the Member States by developing a strategy for integrating sub-surface components into GMES and guidelines on reconciling extraction activities in or near Natura 2000 areas with environmental protection;

- Encourage better networking between national geological surveys to improve the knowledge base of mineral deposits within the EU and to facilitate the exchange of information, improve the interoperability and dissemination of data;
- Promote skills and research, research projects on resource-efficient products and production, development of substitutes;
- Increase resource efficiency and foster substitution of raw materials, promote recycling and facilitate the use of secondary raw materials; encourage recycling markets through Legislation, standards and labeling, Public procurement; Financing, Knowledge sharing and international action;
- Support sustainable supply of minerals needs to be eco-efficient by improved resource and energy efficiency and by increased use of secondary raw materials;
- Ensure sound and harmonized enforcement of the Waste Shipment Regulations.

6. OVERVIEW OF GLOBAL MINING MARKET.

Situation on the mining activities during the last decade and especially in the last five years have been with big price oscillation especially for metals. Many small mining companies have closed activities due to rapid decrease in metal prices up to 40-45 % in some metals such as copper, nickel, etc. Even that merging economies around the globe continue their rapid industrialization; still demand for commodities is low. Yet at the same time, numerous countries are taking steps to safeguard their own supply by curbing the export of natural resources and shutting down some traditional supply markets and this is doing more than affecting commodity prices and hedging phenomena is affecting also. It is changing the way how mining companies is doing business, today's demand drivers are significantly different than they were in the past and mining companies need to change the way they pursue growth if they hope to keep pace.

The global recession that continues with the low increase even in now days had impact on the worldwide production of many minerals. Although there are still concerns surrounding certain national economies, the beginning of the global recovery is reflected in increasing demand for certain key minerals and metals. This is demonstrated by rising prices, particularly for the rare earth elements and metals. Prices are also driven higher by continuing concerns over the long-term security of supply for particular commodities. However that, the developing economies' strong appetite for commodities is sending demand signals to the mining industry, as demand grows from emerging economies, the flow of commodities is increasingly moving to non-Organization for Economic Co-operation and Development (OECD) nations, these are being muffled by the difficulties of obtaining permits for new mines and finding skilled labor.

To help organizations take a forward-looking approach to their business planning in the face of these new market realities, have been identified ten of the top issues they believe will influence the global mining sector most in the coming year, presented in order of priority:

1. The fickle face of financing: International investment fuels the sector;
2. When supply can't match demand: Volatility is the new normal;
3. Securing a social license: Engaging stakeholders takes center stage;
4. New taxes, new regulations and new governments: Political agendas take center stage;
5. How to invest more strategically: Hint, you'll need a long-term plan;

6. The lost generation: The war for talent rages on;
7. At the end of the rainbow: Maintaining the search for that elusive pot of gold;
8. A tough environment: Climate change disclosure and adaptation are getting harder;
9. Working with no backbone: Inadequate infrastructure hampers growth;
10. Rethinking industry fundamentals: Exploring new revenue opportunities;

Market forces today are far from typical. Increased governmental intervention in the form of mining industry nationalization or new tax and royalty regimes, coupled with inadequate infrastructure and a dearth of skilled talent, have made it exceptionally difficult for companies to build new mines or expand existing ones to boost available supplies. These trends are creating a supply shortage that challenges mining companies to rethink their operational strategies. Currently, mining companies continue to face difficulty attracting financing, this ongoing trend dictates a growing imperative to understand their underlying value drivers so they can be positioned to attract fair value.

Business priorities and challenges have dramatically changed, while a number of industry fundamentals have remained unchanged, the relative ranking and focus of key trends have shifted. For instance, companies were most concerned with securing supply, managing commodity price volatility, and ramping back up in response to rising demand. Today, top priorities are attracting financing, finding new supply markets, and engaging local stakeholders in an effort to secure a license to operate. The report also highlights that government intervention around the world has increased and is currently on the rise in the form of new taxes and royalties, more stringent anti-corruption legislation, and rising expectations related to environmental protection.

7. NEEDS FOR ADOPTIONS OF POLICIES TO FACE THESE GLOBAL CHALLENGES

Based on evaluation of the all these three components it is needed that our policy for mining activities in the globalization process of the economy should take in consideration new development of the sector globally.

At the same time we need to work in this context that the policies and strategies of the EC shall be oriented towards involvement of all countries especially Balkan countries rich in mineral resources.

For a specific country mining strategy shall take in consideration policies of its neighbors, region and globally because of a free world market of minerals.
A stable macroeconomic situation with positive growth in mining sector need increase of Foreign Direct Investments intensification of efforts on implementing a number of fiscal and legislative reforms, improving the business climate. Importance of extractive industries requires a new approach in the policies towards the development of mining activities balancing them with the environment and social issues.

As a conclusion our respond on such situation shall have as objective to:

- Develop a dynamic steady growth and sustainable development of the mining sector friendly to environment, certified from community for a maximum profit of Albanian citizens,
- Guarantee an efficient mining structure, and encourage private investments in the sector,

- Offer alternatives based on diversity of mineral resources,
- Be prepared for transformation and restructuring,
- Consequently work for better life and transparency towards the public,
- Coordinate and cooperate with the Mining Programs of the country, with those regional and more,
- Encourage investment in high technology, treatment of minerals and re-evaluation of mineral potential
- Realize promotion, efficient supervision, monitoring, post monitoring, competitiveness, and modernization,

By positioned of the state as inalienable proprietor of mineral resources.

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DOI: 10.7251/BMC170701251M

USING RISK AND OPPORTUNITIES MAPPING FOR MANAGEMENT OF THE REACTIVATION OF THE ABANDONED MINES IN ALBANIA

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ABSTRACT

Management as a system, involves all planning and organizational processes to identify, assess, manage and monitor risks in order to take swiftly measures to reduce or eliminate them and additionally provide advance evaluation and timely delivery of opportunities. In the literature there are plenty of cases where risks and opportunities are seen as the same elements but with different directions. This has provided results of their management not of the expected impact.

The mapping of risk and opportunities management is an important process that is realized, concretized and updated in all the phases of management: identification, assessment, administration and monitoring. The use of maps for managing risks and opportunities in terms of the reactivation of the abandoned mines in Albania, acquires features, the specification of which is addressed in this paper.

Key words: *Risk, opportunities, mapping, management, reactivation, mines*.

1. INTRODUCTION

As abandoned mines are classified all mine layout, development works and mining exploitation workings which, unlikely to be liquidated after the extraction of the mineral up to the limits of technological losses, continue to exist, although the exploitation has stopped.

Their existence is evidenced by a folder containing the all documentation on the activity developed from the beginning until the moment of the termination of mining exploitation as well as all decision-making regarding the perspectives of actions towards them [1, 2].

As such, of course with certain specificities were found in the early 2000s, most of mines in Albania. This includes exploitation of coal, iron ore, copper, chromium, etc. [2]. Focusing on the factors that conditioned this situation we mention:

- Loss of sales markets after collapse in the so-called eastern market, where clearing-out contracts were abolished;

- Lack of ability to invest in modern technology in order to lower the cost of mineral exploitation;
- Unclear ownership relations, as the state, as the sole owner, was no longer able to retain shares in the use of underground assets;
- Inability to support the operating costs as a result of low efficiency in the use of workforce and outdated technology.

As it can be seen, these factors are not related to technological losses, and the qualification of abandoned exploitations in Albania is not a qualification based on certain standards, but in casual circumstances dictated by the level of economic development of the country as well as from the conjuncture of the international market.

This is why, initially, due to the improvement of the sales conjuncture, the reactivation of the exploitation of some chrome, copper and iron ore mines in Albania began. This moment marks the appearance of reactivated mining exploitations. To enable the continuation of this process a functional legal technical environment was created, which includes the following measures:

First, it was decided that the mines would be awarded with a concession on the basis of a concession contract.

Secondly, it was permitted for the exploitation to be carried out in the remaining parts of the ore, which were left as protective columns or as areas with low content of useful mineral components such as Cr_2O_3, Cu and Fe-Ni.

Thirdly, the exploitation of these parts should take into account the conditions of conducting the process within the rock mass affected by the previous exploitation.

Under these circumstances, new owners, i.e. concessionaires, should assess both, the risks and the opportunities of achieving a more productive exploitation. It is precisely this activity that is integrated into the risk and opportunity management system.

2. THE REACTIVATED MINING EXPLOITATIONS AND THE MANAGEMENT SYSTEM

In order to carry out the risk and the opportunities management for reactivated mining exploitations, the following four stages of the system [3, 4, 5] are required:

- Identification;
- Assessment ;
- Administration;
- Monitoring.

The following is a presentation of all four phases, concretizing their use in the chrome Bulqiza mine [6].

2.1 Identification

Risks, involving the entire mining region allowed for exploitation, are classified as georisks. They relate to the probability of simultaneous occurrence of risk (susceptibility) and vulnerability [2, 7].

For the calculation of the risk are taken into consideration [8, 9, 10]:

- Potential factors such as the physical-mechanical properties of rocks, geological conditions, hydrogeological conditions etc.
- Activating factors such as the mining exploitation system, the management of rock pressures, the extent of the mineral deposits etc.
- The mapping of the shape of the impact of the exploitation on the ground surface (funnels, sinkholes, subsidence troughs, landslides etc.);
- The use of land surface;
- Vulnerability of buildings and other threatened objects.

The construction of the assessment model is carried out according to the principal scheme shown in Figure 1.

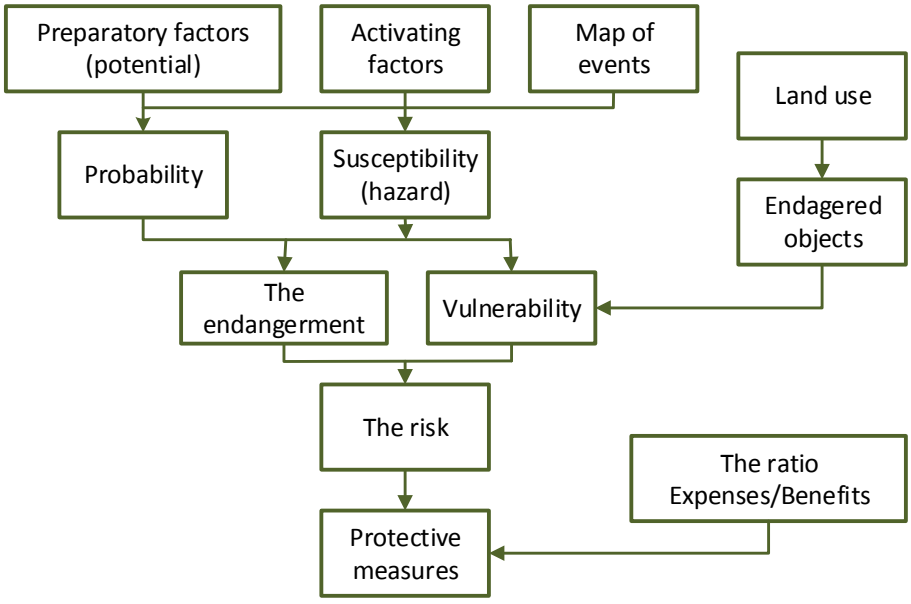


Figure 1. Main scheme for estimating risk

The use of this scheme leads to the construction of susceptibility and risk maps (Figure 2, Figure 3) [2, 7], which identify both their location and intensity.

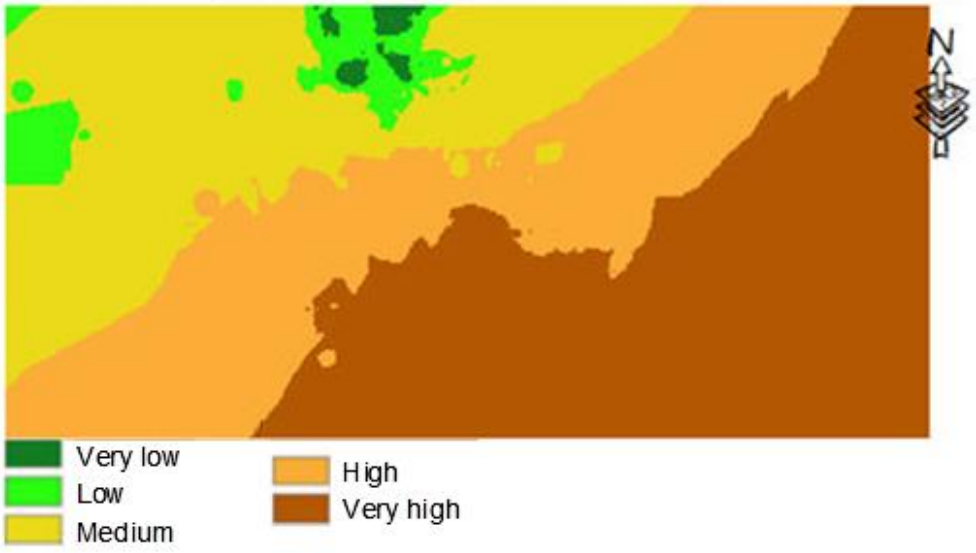


Figure 2. Map of suceptibility (Bulqiza region)

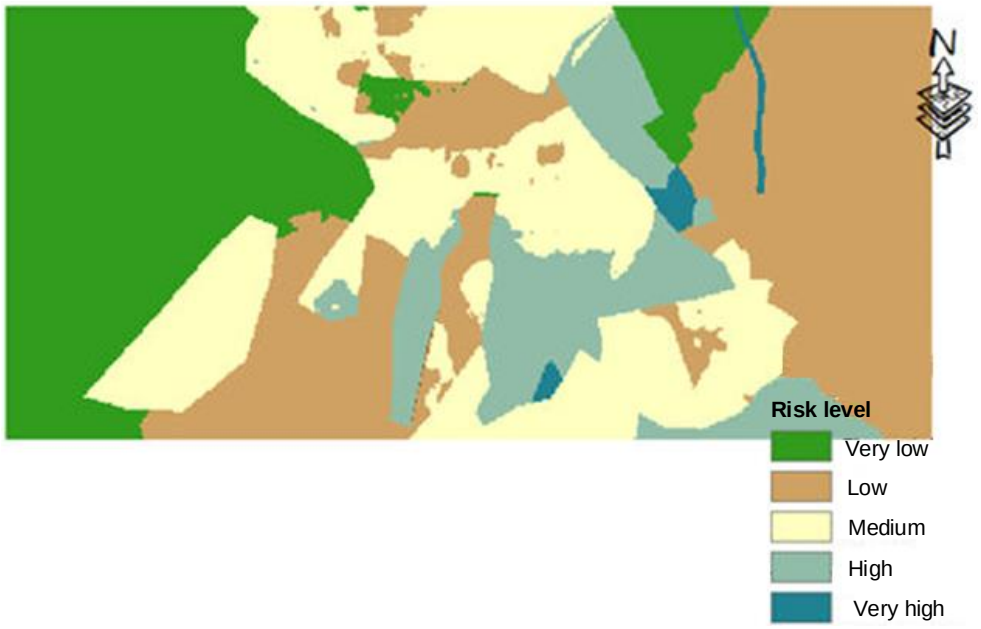


Figure 3. Map of georisk (Bulqiza region)

The map of susceptibility, for the area under consideration, indicates the risk of funnels on the ground surface. The mapping of funnels occurred, over a period of more than 50 years, confirms the validity of the model used. As seen in Figure 4, funnels are located within the area where the susceptibility is very strong, respectively strong.

The risk map is obtained as a superposition of susceptibility to the objects that are at risk by classifying the scale of their vulnerability. The regionalization of risk identifies the areas where the potential of damage caused by funnel appearance takes qualitative qualification, from very weak to very strong.

To evaluate the chances, in the case of reactivation of mining exploitations, their identification should be related to the following cases:

- Avoiding damage by not conducting the mining exploitation;
- Reducing damage using mining exploitation methods that permit this. For example, instead of using the open space method, use the partial fill method;
- Avoiding damage using reinforced construction measures for the protection of endangered objects.

2.2 – Assessment

To evaluate the risks and opportunities are taken into account:

1. For the risks: the frequency of occurrence and the value of the damage;
2. For chances: the probability of occurrence and the potential of impact.

As far as these criteria are concerned, they should be seen in the specific context of reactivated mining exploitations. Although there is more than 50 years of intensive mining exploitation in Albania and studies of the impact of such exploitations both in rock mass and in ground surface [9, 10], it has not been possible to make a statistical estimation of the frequency of funnel appearance, as the development of the process in time so far has not been modeled.

As for the value of the damage, a classification of objects located above the exploited areas [2, 7] is realized which is taken in consideration by concessionaires and responsible state authorities [2] to be used in the risk and opportunities management phase.

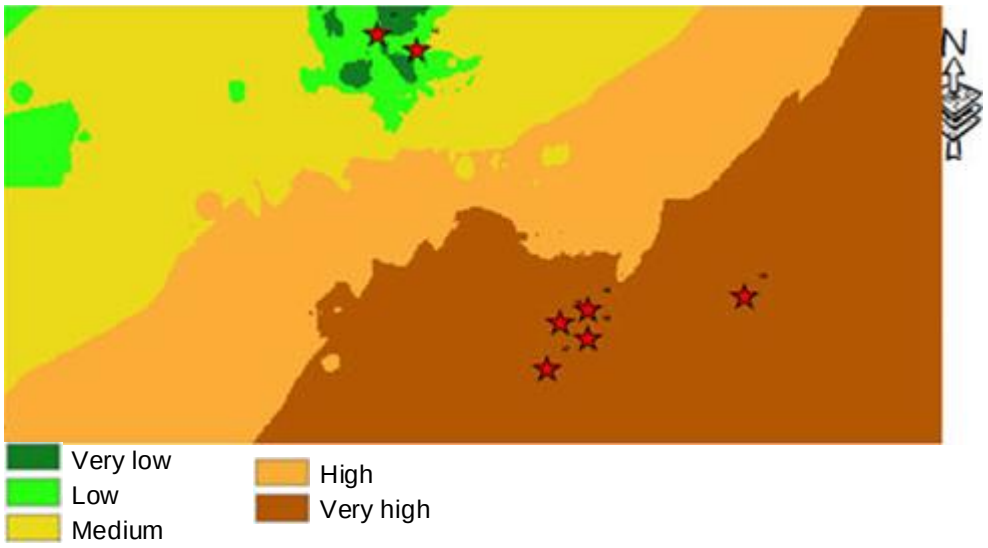


Figure 4. Presentation of sinkholes encountered in reference to the intensity of susceptibility

2.3 – Administration

As a phase of risks and opportunities management, administration includes the whole set of measures that aim at [3]:

- Avoiding risk;
- Reduction of risk;
- Risk transfer;
- Acceptance of risk;
- Detection and use of opportunities.
- In the case of reactivated mining exploitations, using risk assessment maps facilitates decision-making regarding:
- Interruption of mining exploitation to avoid risks;
- Use the most appropriate way for managing rock pressures, resulting in risk reduction;
- Drafting contracts between concessionaires and land users, so that the former have the obligation to undertake the full recovery of the damages;
- Acceptance of the extent of the damage in cases when it affects objects with low degree of vulnerability;
- Taking recovery measures to increase the chance of using objects on the ground even under the conditions of reactivation of underground mining exploitations.
- As concrete measures and with higher usage possibilities in the conditions of Albania are undertaken:
- Fencing of the endangered areas to stop the entry of individuals or other living beings;
- Transfer of facilities such as administrative (office etc.) or technological (enrichment plants, electric cabins, warehouses etc.) outside the areas of dangerous impact.

2.4 – Monitoring

Even in the case of reactivated mining exploitations, the monitoring process includes a set of measures for monitoring and reporting changes in the development of risks and opportunities [3, 4]. The attributes that this process gains are related to conducting observations in a rock mass that is subject to secondary influences of mining exploitation. Observations are carried out within areas of a qualitatively classified risk class. The monitoring results are included in the models of risk and opportunities assessment, which influences the creation of a new situation. This situation then dictates a new procedural review of the identification, assessment and administration phases. Specifically, the results of the monitoring of the mining exploitation of the remaining columns in the Bulqiza mine created the possibility of forecasting the prognosis of the release of a new funnel in August 2017 (Figure 5).



Figure 5. Funnel appearance in Bulqiza chrome mine in August 2017.

Regarding the experience created in Albania concerning the monitoring process, it should be stated that it is not carried out regularly. This is related to the fact that objects (buildings etc.) on exploited mining areas were once classified as abandoned, as well as the lack of legal provisions and standards. Such a practice also weakens efforts to develop and implement ambitious projects related to land use.

Recommendations

- The process of managing risks and opportunities in terms of reactivation of exploitation in abandoned mines should be based on the respective mapping;
- The mapping of geo risk and opportunities enables a qualitative improvement of the management even under the conditions of reactivated underground mining exploitations;
- Under the existing conditions, monitoring is still the most non-stabilized chain in the process. Its improvement should be based on the adoption of legal provisions and the use of unified standards;
- In addition to proceeding in the four management stages, it should be coordinated with the activities of central and local government. In this context, the prepared maps realize a unique understanding of the process and help to better reconcile the measures for risk management and the use of opportunities.

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DOI: 10.7251/BMC170701259M

MINING PRODUCTION WITH ENGAGEMENT OF OTHER COMPANIES EQUIPMENT AND MASHINERY - PRAXIS AND PESRPECTIVE IN THE WORLD AND THE REPUBLIC SRPSKA

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ABSTRACT

This paper presents an analysis and display conditions whereby the mine owner or concessionaire and the holder of mining rights can think about the effectiveness of the mining production organization by equipment hired by another contractor so-called Contract mining. In addition, the paper provides a some review of the exploitation of mineral resources in such a way in the world practice and examples of such mining of production in the Republic Srpska and the neighboring countries

Organizing of mining production by engagement other subcontracting companies can be made with the performance of the overall production process/exploitation or individual- technological stages of production, while in our environment and practice mostly used the second option.

This paper presents also an analysis when it should be considered and situation in which to think about the expediency of production with its own equipment in the new working conditions or the production within these conditions should realize by engagement other specialized companies for individual technological stages or the whole mining production process.

Key words: evaluating mining options, owner mining, contract mining

1. INTRODUCTION

The Contract Mining is a term related to the mining practice whereby the mine owners or concessionaire and the holder of mining rights engage other specialized companies in the field of mining or construction on the performance of the overall production or certain technological exploitation phase - blasting operation, the loading and transport of the material/ore or waste, processing and preparation of useful mineral raw materials, construction and maintenance infaratructural and mining facilities (dams on sludge landfills, external waste dump, roads, depots, buildings, etc.).

Most Australian mines, as for example, use out-sourced services in some way for mining ore and waste. The main distinction between Owner mining and Contract mining is therefore normally based on whom has the most control over the mining process. This is most clearly demonstrated by which party owns (or leases) the mining equipment and operates that

equipment. In Australia, as we said the most precise example, over the last 15 to 20 years there has been a significant increase in the number of mining operations, both open pit and underground that use independent contractors to carry out mining activities. There have been a number of conferences, plus individual papers written, on the subject of contract mining, mostly from the contractor's perspective, or mine owner and concessionaire and the holder of mining rights where in arguments of relations owner / contractor of mining operations and they have different access depending on what aspect is observed (1).

This paper will cover some of the better known arguments about contract versus owner mining in our region but also provide an overview of current practice in the World and some examples of good practice in the Republic Srpska and Serbia with some different approaches for both mining options to help the mine owner evaluate them.

The extended recession in the mining industry at the beginning of this century and the prolonged period which the industry has suffered depressed product prices has had the effect of reducing investment in new and replacement production capacity or exploration of new areas and deposits. . This has in turn caused the cancellation and deferment of new capital projects and as a result, we have on the traditional mining market the mining companies and mining contractors operated suddenly evaporated.. The recession has had other effects. The industry has rationalised and in the process undergone significant retrenchments. Marginal mines have been pushed closer to the edge of insolvency and this has brought about shaft shutdowns, mine closures and the sale of aging and unprofitable mines to nontraditional operators (2.)

Some mine owners had to hire as contractors / contractors in other mines because they had to find alternative sources of work and revenue to ensure their survival as viable businesses. They searched for and found work in their traditional market places the mines. The sort of work changed from that of a capital nature to more and more working cost type of employment. The transition has been fairly gradual. Shaft sinking phased into development, rock handling system construction and rehabilitation, chutes, tips and rock raises, ore body access development, sweeping, vamping and finally stoping. This was a fairly natural progression and was accepted by the industry especially on mines that contractors had begun as green field sites. The South African Mining Industry is well served by several contractors who have access to expertise, skilled manpower and plant pools that are unparalleled in the rest of the world.

This represents a tremendous resource that the industry can tap into or withdraw from at very short notice and virtually as required. (2).

The surface mining process for example consists of a number or interrelated functions, including:

- Mine design;
- Scheduling & budgeting;
- Drilling & blasting;
- Loading and hauling of ore and waste,
- Waste disposal and the landfill maintenance,
- Equipment maintenance
- *Infractructural facilities maintance and other

All of these functions can have a supply and/or service component, which is typically defined by a formal contract. Supply contracts are normally based on a specific material or product where the focus is on quality, product specification, delivery and price.

Service contracts are typically more complex and which party is responsible for what function is important. How risk is shared and the duration of the service may also be significant issues. There are often a number of choices as to who has the prime responsibility for each function or part of a function, ranging from most functions being under the direct control of one contracting company (Contract Mining) to the other extreme of the mine owner controlling the majority of functions (Owner Mining) (1).

Agreements / contracts after out-turn - Contracting mining industrie offer various types of agreements and arrangements for contract administration. The most widespread type of contract is *the firm price measure* and *value agreement*. This arrangement promotes productivity because the price estimation process requires that production targets are set for the tasks being tendered for. This is a most important part of the discipline because it demands the timing of all tasks and so enables them to be rated for incentive bonuses.

Practically all contractors work on bonus systems. This provides motivation for their crews and enables them to attract labour to what must be rather precarious employment when compared to the steady work traditionally offered by the mines. Resources required to achieve these aims are assessed, costed and allocated to the tasks. So the greater the production for a given resource levels the higher the profitability and therefore productivity. (2)

The contractor is so provided with the most powerful of incentives to maximize production and fears or suspicions on the part of the client regarding excessive profit taking by the contractor can be allayed by the introduction of *risk sharing arrangements*. On this way profits or losses outside a pre-agreed tolerance level are shared (for example profits or losses which exceed 15 % of the measured value of work done could be shared on a 50:50 basis). There are some pre-requisites for this type of contract:

1) The scope of the works must be clearly definable.
2) The work to be done should be easily measurable.
3) The responsibilities of one party to the other must be defined (2).

The process of choosing the best contactor relies on market forces to ensure that competitive prices are obtained and provided those who are invited to tender are qualified to execute the job, award to the most competent tenderer will invariably yield a satisfactory result. Long term relationships between client and contractor often develop and can become the subject of continuous negotiation and adjustment.

There have also been significant changes with how mining contracts have been structured and managed. Contracts have become more detailed, due to additional experience on both sides, and in some cases the traditional Schedule of Rates form of contract has moved towards a cost plus profit margin type of contract. However there has also been an increase in litigation between some contractors and mine owners, which highlights the need to have contracts carefully drawn up and then well managed by both parties thereafter. The other change in contract mining is a consolidation of the contracting companies. There are now a small number of main contractors, a few medium sized contractors and a number of smaller companies, down from a total several tenth contract miners only decade ago. The move to increasing economies of scale and reducing overhead costs by merging reflects the mining industry as a whole (2).

Long term relationships between client and contractor often develop and can become the subject of continuous negotiation and adjustment. Another often used method of ensuring even handed treatment is to put these open ended contracts up for periodic retender.

There are various other arrangements have been developed and applied, all with varying degrees of success and satisfaction. The one type of agreement which should be contemplated in certain cases is "cost plus", this in its pure form the least productive of possible arrangements, and can be applied when:

- The scope of the work cannot be defined and time pressures or other circumstances prevent its ascertainment or;
- There is only one contractor with unique skills qualified to do the job and is unwilling to work on any other basis
- If the cost portion cannot be estimated with any degree of certainty it is recommend some limit or performance linkage be applied on this way.
- There are other "conservative" rules / concepts: Own customer responsibility for service and quality control or "Buyer should beware" applied in every part of the time at the recent commercial transactions.

2. ACTUAL TRENDS IN WORLD MINING

This chapter presents a practical some information from the text published on the Internet site: PwC Australia www.pwc.com. entitled "An annual review of global mining trends, mine Global 2107" (3).

„Mine is an analysis of trends based on the financial performance and position of the global mining industry as represented by the top 40 global mining companies by market capitalisation. 2016 was a year of recovery for the world's largest Top 40 mining companies, with profitability returning and balance sheet repair well underway. Rapidly rising commodities prices sparked renewed market optimism and improved credit ratings across the Top 40 firms. Valuations also climbed, especially for the traditional miners, with the trend continuing into early 2017 even as commodity prices remained flat.

Through the 2016 is observed the industry move past the danger and decline of 2015, to draw breath and consider where to from here. The longer term question, as how the mining industry should 'act' with apply the lessons learned from the last cycle, how to reap the benefits from investments they made and take a more countercyclical approach to acquisitions and disposals.

Recovering from 2015's hang, the members of the Top 40 paused in 2016. Rapidly rising commodities prices promised a way forward and the Top 40 the responded with new valuations.

But valuations do not bring news and the impression is that the companies ares still determining further guidelines, since the previous recession left traces and limitations still exist. Rapidly rising commodities prices has been welcomed, but with cautious optimism and warnings to use the lessons of the past. Thus began to use a mine site safety mantra "Stop, think…….act" and is now considering "where to go"!? Some members of the Top 40 mining companies were presenting their intentions in 2016 but 2016 was not the year of action. Now waiting for the reaction and in which direction the mining industry to move forward.

In 2016, traditional players continued balance sheet bolstering to calm the market and stop the angst associated with financial distress. A heavy emphasis was placed on shedding debt. The brakes were firmly applied to exploration activities which continued to shrink, and what little was undertaken was generally allocated to "safe" jurisdictions. Capex fell dramatically again, by a further 41 percent, to a new record low of just $50 billion, and there was a lack of significant greenfield projects announced or commenced. Production was generally flat.

While oil prices increase in the 2016 , prudent cost control measures ensured operating expenditure was constrained. Traditional mining companies were rewarded with a strong upswing in their market cap, and earned some room for maneuver. Many planned disposals were called off in response to better market conditions The exception to this was the 11 Chinese companies within the Top 40. China defied conventional industry behaviour and invested at the bottom of the cycle. The most significant asset buyers among the Top 40 were Chinese companies

PWC Australia is asking what is the next and where to go in the mining industry and is the defensive strategy to simply repaying debt, preserving cash, sustaining existing assets and waiting for a period with sustained increase in prices only solution? In the short term, shareholders appreciate the strengthening of balance sheets and increases in share prices, but the mining industry will need to execute a longerterm vision or it will remain at the mercy of commodities speculators. Shareholders will demand performance from the existing asset base, dividend growth, or they will simply reallocate their capital if the mining sector cannot provide a long-term growth vision. There is clearly a divergence in thinking between Chinese companies and the rest of the Top 40 as their goals are different and Chinese capital is more patient. China aside,is ready that praise should be given for the efforts to repay debt, innovate and adopt new efficiency measures – all of which have helped to curb costs and restore credit ratings and investor trust. But wuestion is where will this thinking take the mining industry if a “playing it safe” attitude to investment prevails in the future. Some statements are that will lead back to old habits of lavish spending in a boom followed by a wave of write-offs during the bust that inevitably follows.

New opportunities and hazards are on the horizon. Do we take it seriously when poses the question "Can we one day stop mining the Earth altogether?" or when Elon Musk puts forward a 100-day guarantee to fix a state’s energy crisis with battery technology?

The industry must carefully consider how it responds. Many in the Top 40 have reflected on the qualitative aspects of their license to operate and additional increasingly demands of the communities for exceptional corporate social responsibility. In terms of safety standards and broader economic contributions, the mining industry has long done some heavy lifting. However, the story often fails to resonate with governments and the broader community

Healthier price-to-earnings (P/E) multiples returned. And, even as price growth slowed early this year, valuations continued to rise until April. This provides a platform for the industry to act into the future. What is happening today does not seen significant action on the future direction of the Top 40, at least by the traditional players. Taking into account the long-term planning such as Rio Tinto takes into account the cyclical nature of the mining industry, and makes a decision that their CEO has a "10-year mandate" The future may be about integration and emerging market companies, who are also focused on new world minerals, are increasingly integrated. In the traditional markets, we are seeing new players seeking to secure supply. It remains to be seen whether the major companies to turn in that direction and whether there will be new investments in new projects, M & A and technology.

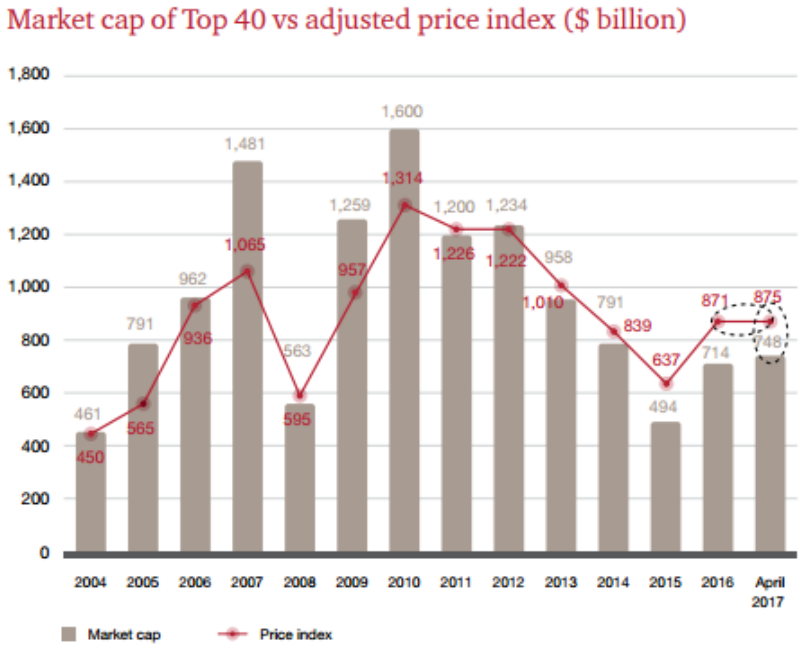
It is noticeable lack of plans and intentions for further development of large mining companies but it is the result of the situation, when early 2017 has heralded further volatility in prices and later further reduction in profit compared to the year 2016. Few things are certain in this industry, but western mining companies know that China is unwavering in its strategy, shareholder activism is rising, government interventions are becoming more commonplace and new players are disruptive.

Industry in perspective -It has been observed the dust settling in the mining industry during 2016 after its long virtual halt. Today, after years of pulling back on investment, exploration and human resources, the world's largest mining companies are ready to move ahead. They have cut debt, strengthened balance sheets and taken necessary impairments. In the process, the mining companies have found themselves in step with an awakening global demand for most commodities, and they have watched their credit ratings rise and valuations grow. This year will be all about assessing options and making the right corporate decisions to sustain the market optimism that these events are not short-term.

The first quarter of 2016 was a turning point as mining industry fundamentals started to improve. Through the year, it was observed a rise in both spot commodity prices and the market capitalization of the Top 40, two markers which have historically been strongly correlated. Though prices have not yet rebounded to the pre-downturn levels reached in 2011, we do see evidence that they have bottomed out. While spot commodity prices remain volatile, long-term analyst consensus price forecasts held relatively stable throughout 2016. The key to a sustained recovery will be to ensure that the mining industry does not repeat the mistakes of the last boom cycle: buying high, pumping up production with marginally profitable and expensive projects, and then recording significant impairments when commodity prices decline.

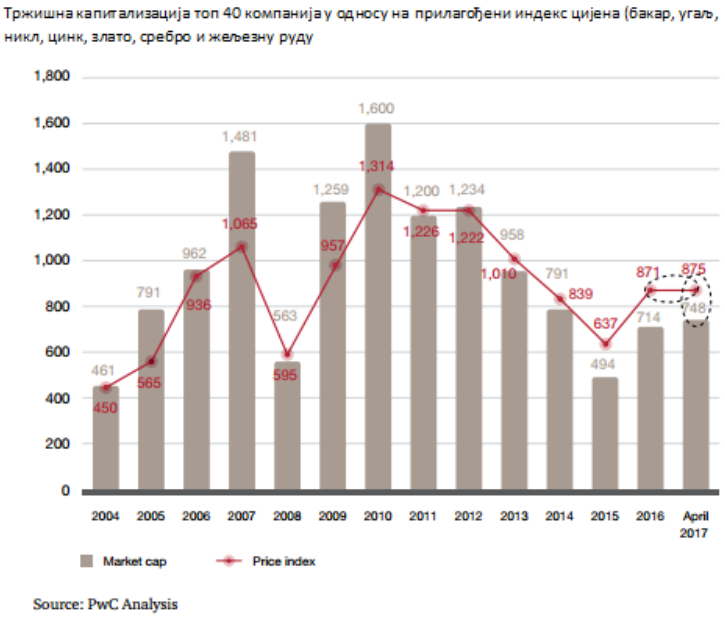
Mining companies need to impose better capital discipline in the decade ahead and, indeed early evidence suggests that they began to do so in 2016. The industry must also consider the potential gain of bolder moves while costs are still relatively low. Last year marked the return to profitability of the Top 40, with an aggregate net profit of $20 billion in 2016 as compared to an aggregate loss of $28 billion in 2015. Valuations also climbed, especially for the traditional miners, with the trend continuing through Q1 2017 even as commodity prices remained flat. The mining industry remains a long way off the peaks of previous cycles, but it has regrouped and begun to rise again.

Movement in Top 40 market capitalization -Overall the market capitalization of the Top 40 increased in 2016 by 45 percent to $714 billion, approaching the 2014 level. Rising commodity prices played a driving role, but there are questions: „Have companies been lucky or good? By the end of April 2017, valuations had gained an additional $34 billion during a period when spot commodity prices were relatively flat. This data suggest that the market is valuing stronger balance sheets and solid management, suggesting that investor trust is on the rise and the recovery is sustainable. It is worth noting, however, that the rise in valuations was distorted by spot iron ore prices.



Source: PwC Analysis
Market capitalization of the Top 40 companies against an adjusted price index for a basket of commodities including copper, coal, nickel, zinc, gold, silver and iron ore.

Figure 1.



Source: PwC Analysis

Figure 2.

Among the traditional companies, four companies represented almost 50 percent of the increase in overall market capitalizations, and each of them has exposure to iron ore:

- BHP Billiton Limited (BHP)
- Rio Tinto Limited (Rio Tinto)
- Glencore Plc (Glencore)
- Vale S.A. (Vale).

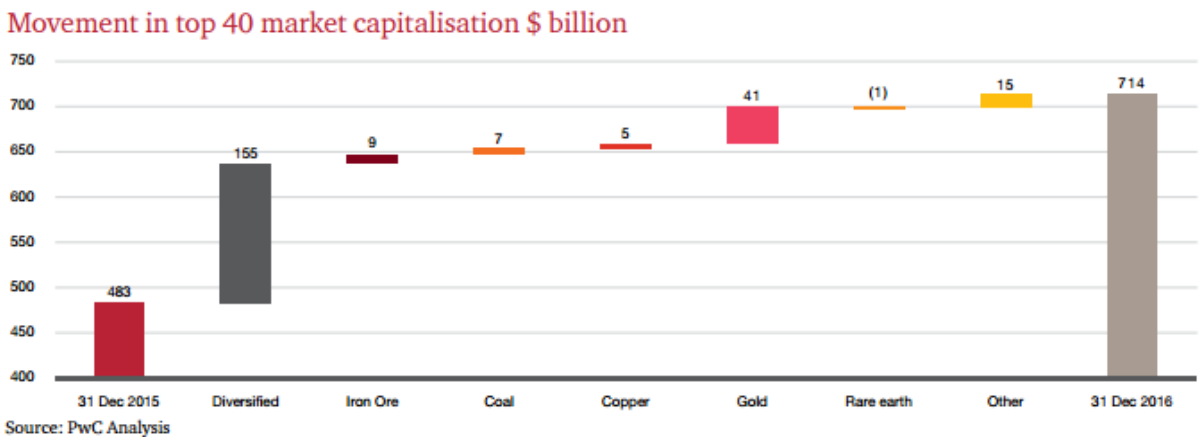


Figure 3.

The three largest increases as a percentage of 2015 market capitalization were Anglo American Plc (Anglo), Fortescue Metals Group Limited (Fortescue) and Teck Resources Limited (Teck). Anglo and Fortescue hold major iron ore assets. Teck, meanwhile, has significant exposure to steelmaking, coal and copper. Traditional companies had larger gains in value, representing 86 percent of the total increase in market capitalization. Chinese companies did not receive much of a lift from rising commodity prices; this may be because they have less sensitivity to price changes and their investor base has fewer liquidity options (and limited investment alternatives) as the ability to invest outside of their country is limited

Impairments significantly reduced - After hitting a near-record in 2015, impairment charges tumbled last year to a less-alarming $19 billion. Although the impairment charges tumbled in 2016, miners also scaled down on capital expenditure in 2016. Hence, impairments taken by miners were still almost 40% of the capital expenditure incurred in 2016. This percentage is close to the average of the impairment as a percentage of capital expenditures. As part of the focus on the capital allocation and the under-pinning of their balance sheets, the Top 40 reduced the outflow related with capex. Closer inspection of the Top 40's 2016 capex revealed that approximately 50 percent of capex was related to sustaining activities, implying that only half of the $49 billion was growth capital, with the remainder used to maintain operations.

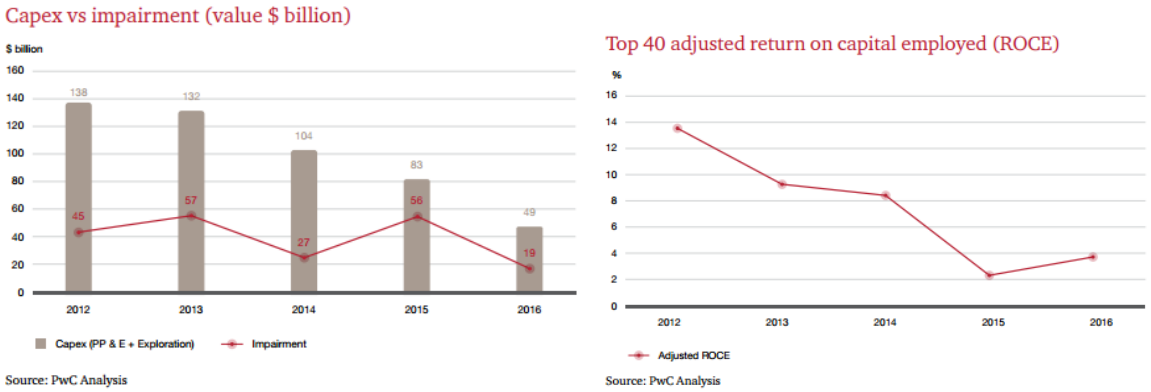


Figure 4.

Impairment losses also were reduced, mainly due to the significant impairments recognized in the prior year and more stable business conditions. Although both indicators were below the prior year's level, it is important to highlight that the proportion of impairments/capex (2016: 39%) has reduced to a level that is similar to 2012 (33%) from the peak of 2015 (77%) which indicates that miners are responding to messages around capital discipline. Iron ore prices doubled to the end of the year, reaching a high of $80 a tonne (CFR spot Australia). This trend continued in early 2017, with prices peaking at a 30-month high of $89 a tonne in mid-February, only to suffer a sharp reversal thereafter.

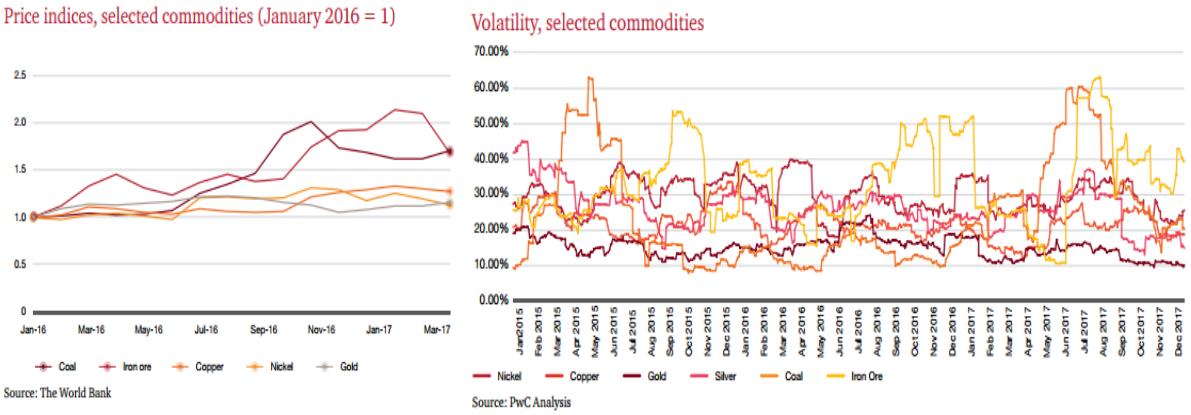


Figure 5.

China in the driving seat - During the downturn, Chinese companies demonstrated one enormous advantage over other miners in both traditional and emerging countries: access to capital. With deeper pockets than their competitors, Chinese players were able to fund more acquisitions than their counterparts, either snapping up assets at premium prices or buying opportunistically. We also witnessed an increase in acquisitions by Chinese private equity firms, and we expect China to continue to be active in acquiring global mining assets as a way to reduce its dependency on imports. One variable worth watching, however, is concerns regarding restrictions on capital outflows by the Chinese government. We have recently seen, for example, tighter approval processes for foreign acquisitions by Chinese companies, although these are not specifically targeting. The Chinese government said in February that the new measures are only directed at reducing suspicious or fraudulent transactions.

3. PRACTICAL EXAMPLES OF CONTRACT MINING IN REEPUBLIC SRPSKA

When talking about the contracting mining practice in the Republic Srpska, Bosnia and Herzegovina and the region it is necessary to give some basic elements of the legal framework. The rules and regulations in mining and geology in the Republic Srpska related to contract mining could be sublimated in a number of articles of the Law on Mining and Law on Geological Research in the Rerublic Srpska.

According to Article 11 of the Law on Mining (Official Gazette of RS 59/12) exploitation of mineral resources should be done in a way that ensure optimum utilization of mineral deposits, the security of people, facilities and properties and environmental protection, in line with current scientific achievements, regulations, standards and technical norms.

According to Article 14 of this Law, exploitation of mineral resources may be performed by a company which is registered in the business register with the competent court registration for this activity. Company has also for management performing of mining operations, surveillance and other activities have employed for an indefinite period of full-time workers who meet the prescribed conditions in terms of the type and level of qualification, experience and authority to perform these tasks. Company shuold meet the fit-out in technical equipment, work safety, protection and improvement of the environment and other conditions prescribed by this law. For the exploitation of mineral raw materials is necessary a concession granted in accordance with the law regulating concessions

In the Article 18. stated the concessionaire may for performing of specific mining operations employ a company (hereinafter referred to as sub-contractor), which meets the requirements referred to in Act. 14, 15 and 16 of the Mining Law and with whom it has a contract to activities perform. Sub-contractor is bound to be comply with the provisions of the law during the performance of tasks for which he is engaged.According to the Law on Mining is practically allowed the possibility that the concessionaire or exploitation rights holder may hire another company to specific works or technological stages of exploitation and production but the formulation of Article 18 leaves dilemma whether it allowed the concessionaire or enterprise with exploitation approval to hire another company or several companies in the organization of the entire mining production. In any case, it may be noted that such examples do not have in the region where is applied ,as yet unwritten rule: " concession right does not grant a company that has no experience in mining" (the last example is the issue of privatization iron ore mine Ljubija, where it was noted that the company could not buy a customer who has no a reputation in the mining industry and who has no experience in the performance or organization of mining production).

Here is an overview of current examples of implementation of contract mining in the mines of the Republic Srspka:

- During the exploitation of iron ore at the open pit Buvač - Omarska mine company "Arcelor Mittal Mines" Prijedor had hired from 2010 the companies with classical discontinuous equipment (systems, excavator-truck) to the excavation and transport of overburden and waste. Period of hiring equipment from other companies should take about 5 years (although that period today and extended), which is defined as planned exploitation and overburden and waste quantities that require involvement of a number of smaller mining equipment and machinery in the defined discontinuous exploitation system. Missing number of dig-loading and transport units the company has not purchased but hired two companies with their equipment, which meet annually waste capacity.
- Supply of diesel fuel and lubricants and oil through the sub-contractor company in "Arcelor Mittal Mines" Prijedor
- The company "Integral" Banja Luka had been hiring a two-year period of exploitation of hisown diabase quarry "Trnava" - *Podgradci* missing equipment on grinding and classifying stone by hired mobile crushing plant and mobile screen plants which meet the increasing needs of the market during the construction of the highway for certain stone size classes and materials for the production of highway roadbed (Banja Luka – Gradiška),
- Company "Boksiti" -Milici has a practice equipment engagement owned by other companies for more than fifteen years, and in cases when it is necessary in a certain period to excavate of increased overburden and waste amounts at open pit mine "Podbraćan" near Milici.

- At about 60 quarries in the Republic Srpska and 40 limestone quarry where it is necessary to prepare the material for loading by drilling and blasting technological stage concessionaire and exploitation rights holder organized by subcontracting in the form of small specialized companies for these jobs. This subcontractor companies have licenses to perform these tasks with its own equipment and personnel and they are more rational and perform these tasks at a lower cost ,
- During the nineties of the last century and the beginning of this century quite a level of engagement of subcontractor companies to excavating,loading and haulage of materials at the coal mines "Bogutovo Selo Sjever" - Ugljevik and "Gračanica" – Gacko,
- Carrying out maintenance work on buildings at the mines is the practice used by most of our mining companies.

Reasons, the terms or contract mining effects of performing specific technological phases of mining operations, or the overall exploitation on the specific mines and certain mineral raw materials can be seen with several aspects:

- hiring equipment from other companies within a specific time period at surface mines has the aim to control the impact of capital expenditures on the mining exploitation economic effects . When defining the mining activities in mine development of such a open pit can be observed a certain period of one year to several years when i necessary additional / higher egagement of number of dig-loading, haulage equipent, additional machinery and devices for the beneficiation and the like. In addition it is possible to observe the situation when the market is experiencing increased demand for specific mineral raw materials, which implies an increase in production capacity. New machines purchase very often may not have positive economic effects, because of the short duration of use of these additional machinery or uncertain length of time in which is necesearry organizing the production with greater capacity,
- the recruitment of other companies equiment mining company like concessionaire and mining rights holders reduces in some way operational costs - the cost of labor, equipment maintenance and others.
- It is necessary to achieve certain pre-conditions for the use of such solutions in the organization of work and entrusting the certain technological stages of exploitation to other companies - subcontractors : a spacious layout placement of sub-contractor equipment where these equipment will not interfere with and jeopardize the operation of the equipment and machinery owned by the concessionaire of the mine. Besides of that companies should provide the harmonization of the regime and the dynamics of their own labor and equipment involved, the establishment of separate production and technological units in which the work of individual ensemble otherwise be independent of the work of others (drilling and blasting operations may be performed independently, operation of the equipment on waste material can be performed independently from the operation on the ore, the operation of crushing / disintegrating and classifying is independent from the operation of the excavating and transport etc.)

4. SOME EXAMPLES OF CONTRACT MINING GOOD PRACTICE IN SERBIA

Names of sub-contractor companies that have performed work in the Serbian underground mines in the period 1998-2010.

Table 1. Works on construction of mining underground rooms

	Name of company	Mines that they have performed work	Number of employees	Total meters of underground rooms performed during the year
1	Argentarija Beograd	RMU Rembas, Ibaarski rudnici, RMU Štavalj, RMU Jasenovac	220 -250	3000-3500
2	Bergman Knjaževac	RMU Rembas, RL Lubnica i RMU Bogovina	55-60	900-1000
3	ODP Kreka Bijeljina	RMU Rembas, RMU Jasenovac	60-65	1100-1200
4	Kreka Input Beograd	RMU Rembas, RMU Jasenovac	60-65	1100-1200
5	Ruding Novi Sad	RMU Rembas	50-55	900-1000
6	Rudeks Vrdnik	RMU Rembas, RMU Jasenovac	35-40	700-800
7	Integral kop Beograd	RMU Rembas, RMU Jasenovac	30-35	700-800
8	Perović i ostali Beograd	RMU Rembas i RMU Soko	45-50	900-1000
9	Resava inženjering Despotovac	RMU Rembas, RMU Štavalj, RMU Jasenovac	45-50	900-1000
10	Morava ineks Niš	RMU Rembas	35-40	800-900
11	Wystavba Doly Острава	MU Rembas i RMU Soko	35-40	1200-1300
12	RGP- Vrdnik	RMU Rembas, RMU Jasenovac	30-35	800-900

Table 2. Work on the excavation and transport of coal

	Name of company	Mines that they have performed work
1	Morava Ćuprija	RL LubnicaIbarski rudnici i RMU Rembas

5. KEY ISSUES IN EVALUATING MINING OPTIONS

Comparison and evaluation of the mining production and organization by subcontractors equipment hiring - contract mining or by exploitation with their own equipment and work of the owners can be performed in several ways, but the main issues and problems can be classified as (L.J. Kirk):

- Corporate;
- Project specific;
- Operational;
- Cost; and
- Risks assessment.

5.1. Corporate issues-issues related to the business policy and the company's work

Corporate issues are those that are normally considered by company directors and group executives, rather than mine-based management. There are many issues that relate to corporate strategies, corporate "culture" or policies, the experience of influential individuals, the outlook for the product being mined and share market perceptions (L.J. Kirk).

Corporate policy on out-sourcing in the mining industry will depend on the strategic focus of themine owner, which can change over time (Dunn 1998).

Companies working in the field of mining can have some other activities such as geological eploration, mine designing, marketing activities besides of the organization of the mineral raw material exploitation. In tough times in the market (e.g., minimising production because of decrease in some mineral raw material demand), those other activities may be their competitive advantage. On the other hand, companies that only deal with exploitation in these difficult business conditions set focus on reducing production costs in certain mineral raw material. The main problem for a number of company / corporation is the availability and usage of available funds, where the return on capital is the key issue. The answer to this question can be positive or negative depends on whether the funds are provided from their own resources or necessary debt. For many smaller companies or companies with poor credit rating, in the case of small deposits or production capacity, for projects with a short working life and exploitation, hiring contractors - Contract mining has advantages. Subcontractors in many cases have better commercial terms for the purchase of such mobile equipment then the mine owners. Therefore, when analyzing the possibilities for their own production organizing or production with subcontracting organizations should consider the following questions:

- Is the solution the purchase of new equipment to debt, leasing on equipment or intended existing equipment utilization? Leasing on eqipment or subcontractors engagement results in a relatively low potential debt in the balance sheet of the company.
- How would this way of production organizing - leasing or other form of involvement of foreign capital impact on future borrowing capacity of the company?
- What investments are required for facilities with infrastructure (eg workshops for equipment maintenance), processing facilities and how should fund this?
- How to use investment capital and impact on company business costs or the unit cost of production? In cases of subcontracting and its equipment costs are treated as ooperating production costs while purchasing their own equipment, especially with the use of loans, such costs are treated as capital rather than operating costs.

Previous issues require detailed financial modeling and assessment like a key component reliability. Another corporate issue that may be important is the company structure of the operating entity. For example both case studies involve joint ventures between major mining companies, and it is reasonable to expect that each joint venture partner will have some different views or priorities on mining options. Where is one owner, or an owner with a clear majority, decision making may be clearer and easier to implement. *Where exist a joint venture arrangement it is recommended that an independent party assist the joint venture parties find common ground and carefully evaluate the differing or conflicting corporate views (L.J. Kirk).*

5.2. Project Specific Issues

Although every mining project is unique, when evaluating mining options the main considerations usually include (1):

- Mine life;
- Is it an existing mine or a "greenfields" project;
- The planned initial mining rate and the variability of the mining rate over the life of the project;
- The availability of trained and experienced personnel;
- The commencement of mining as part of the project schedule. Is mining or pre-stripping of waste material on the project development critical path or is there a commercial advantage in bringing the project on stream as soon as possible;
- Does the project financing require greater confidence in mining costs? Is the feasibility study conservative?
- Is there any government incentive that may affect the evaluation? For example at the beginning of one of the case study projects a government development allowance was available to the mine owner but not to a contractor (1).

5.3. Operational Issues

The key issues revolve around people, equipment and grade or ore mining control. All mining operations are dependent on the people who run them. Identifying the particular skills needed, then locating and retaining the best people for the job is common to contract or owner mining. If the people with the required skills and experience are not already in that organisation the first step is to evaluate how difficult it will be to find, attract and retain these people. Contractors should have an obvious advantage in already having a pool of trained and experienced personnel but the mine owner may also have existing experienced personnel or be confident of recruiting them.

One cost advantage that contractors may have is the ability to work their personnel on a longer roster and more intensive (contractor's employees are more likely to be interested in longer hours for more money) than the mine owner. This could result in reduced employment costs (on-costs and accommodation costs) and subsequently lower mining costs, but only if the labour turnover rate does not become excessive or productivity and efficiency suffers.

The employment of contractors with more flexible labour conditions was part of the solution although it thirty years ago would be infeasible because the industrial relations and union issues хавe previously been a significant consideration to use of contractors in the mining industry (mining industry had a "rigid, inflexible" contracts agreements).

Evident are fewer differences in general employment terms and conditions between contractors and mine owners especially in countries where contractors would still be expected to have better training programs and more experienced trainers.This important item can be addressed by mine owners by utilising the equipment suppliers, trainers and independent specialist training companies, at least in the short term.

Equipment selection and flexibility is a strong point of contractors, especially in the most commonly sized equipment. Contractors have the experience and a current cost and productivity data base on a large range of different mining equipment whereas an owner miner would normally have a much more limited fleet and less direct operating experience. Contractors are also often able to mobilise additional or replacement equipment at short notice, for short periods, or to meet peak demands. An owner miner can still use hired equipment from contractors in the same way but it may require a lot more time and effort to organise and may cost more. A possible counter to the above contractor advantages is that the owner miner may be able to change the mine plan to suit the situation and the available fleet, compared to the risk of having to change the contract scope of work that could lead to increases in contract costs. Also for the largest or more specialised equipment, such as draglines, contractors may not have this equipment available or be able to provide suitable replacements.

Hiring a contractor may be doubtful in cases where ore control during mining, that is the careful separation of the valuable minerals from the surrounding waste, becomes more important in selective, high value mines (shear-zone hosted gold deposits), than in bulk mining operations such as coal or large, low grade metal deposits. The mine owner usually determines the ore mining method although the experienced contractors can often provide practical solutions. The use of backhoes or excavators, compared to face shovels, for selective mining was primarily developed by an Australian contractor, for example (Roche 1996). The issue here is not so much the mining method but the required quality control and quality assurance over ore mining. The contractor has a profitbased focus, which depends on maximising loading productivity, and hence maximising the volume mined. The mine owner may be more concerned about the quality of the ore that is mined, which may reduce loading productivity due to the extra care required in selective mining of ore and adjacent waste. In either case the focus should be on what is best overall in terms of cost and net benefit.

5.4. Cost Issues

It is possible always to ask the question how the costs of exploitation involved in the total operating costs of the mine, and how sensitive is the project in sustainability terms of cost of mining exploitation. The answers to these questions are often the main factors for assessing who should organize and carry out the exploitation itself. In this sense, the question is whether the mine owner should to concentrate on other issues related more to increase the value of the project, such as geological surveys, improving the processing quality or marketing the final product, or production costs are a critical factor in determining whether the project is viable and at the same time a significant factor in determining mine life?

For both case studies mining costs are significant as a percentage of total operating costs and therefore in determining profitability. Both mine production with relatively low grade deposits and the final pit depth and hence mine life is based on economic criteria, not a physical bottom limit of known mineralization.

The issue may then become what cost premium or margin, if any, is justified in employing a contract miner rather than undertaking the mining directly, if there aren't other more important issues such as the corporate and operational issues discussed above or the risk assessment issues. There is always an inter-relationship between cost, benefit and risk.

It should not be assumed that a contractor would cost more than owner mining would. In the depressed times for some resources industry competition between contractors is high. The contractors may also have an existing mining fleet available at a significantly lower ownership cost than a new, replacement fleet, particularly if there has been a significant change in exchange rates that affect new equipment supply prices, for example. It is necessary to carefully analyze the conditions of subcontarctors engagement in terms of cost and in this regard may be mentioned the following situations:

- It is reasonable to expect that the contractor will include at least some replacement equipment costs in calculating the contract prices, as well as including a profit margin no lower than the current cost of capital. Apart from the contractor requiring a profit there is also the cost of duplication of some functions. This includes site-based costs such as some duplication in management and administration and off site costs such as company related overheads that are common to both the mine owner and the contractor. Depending on the economies of scale and the relative efficiencies and competencies of management and administration between the two organizations this cost may not be significant and could even favour the contractor.
- Related to costs the question is who would benefit from any future savings. With changes in mining methods or processes, improvement in technology or continuous improvement of the whole mining operation it is reasonable to expect there would be some savings, or a slowing of cost increases in real terms. For contract mining the majority of any savings would normally go to the contractor and at best the owner would get 50% of any savings. For owner mining the majority of any savings would go to the owner, although suppliers or specialist service providers may earn a share.
- If costs increased rather than decreased the contractor may have to absorb this but only if the contracts scope of work and contract terms and conditions were very clear on the specific issue. It is more likely that the owner will incur the majority of increased costs in any event. However there is a greater risk that an owner miner would be less focused on reducing unit costs than the contractor, who has no other way to improve profitability.
- An mine owner could accept a reasonable margin on the service contracts, such as equipment maintenance and blast hole drilling, but there should be no additional margin on supply contracts such as leasing, fuel, explosives and tyres. The end result is a much lower overall margin but without accepting a major increase in risks, if these contracts are as well written and as well managed as a mining contract would need to be. Managing very specific individual contracts may also be easier than managing one large general contract.

5.5. Risk Assessment

There are significant risks in mining, regardless of who does the actual mining. The mine owner already carries the risk of geological modelling, grade control, mine design, geotechnical stability, environmental and community issues, overall responsibility for health and safety and of course the vagaries of the market for the end product(L.J. Kirk).
In the evaluation of contract or owner mining the main comparative risk areas are:

- Equipment selection;
- Equipment performance (productivity, availability and utilisation);
- Quality control of ore mining;
- Health and safety;
- Human resources management;
- Implementation (new mine) or transition (changes of the system or method of exploitation, or a change of property relations);) risks;
- Contractual and litigation issues; and
- Production or operating costs.

It is often difficult to quantify all the above risks although sensitivity analysis of equipment performance and operating costs can quantify the likely range for both options. There are more subjective judgements required for risk issues than for the other key issues.

6. THE CONDITIONS FOR THE APPLICATION OF CONTRACTUAL MINING

According to a global survey report published by software developer Geosoft in November 2013. the managing data has become a critical concern in the minerals exploration community. There was a growing need for effective data management in the minerals exploration community to attract investors and improve discovery rates (4).

While organisations are getting better at centralising their data on a single platform, more work needs to be done to increase data accessibility, reduce duplication, smooth work-flows and decrease dependence on data experts.Data was collected from 415 organisations around the globe. The 693 respondents represented a cross-section of roles in resource and energy exploration companies, industry service providers, government and educational institutions. More than 60% were from the mineral resources industry, with the remainder coming from energy (11%), government (9%) and educational organisations (5%), among other sectors. When Geosoft conducted a similar survey in 2011, only 18% of respondents identified managing exploration data as a matter of "critical importance". According to the study, this number has risen to 44%, with another 38% of respondents regarding data management as one of their top fives areas of concern (4).

Currently, organisations are taking data out of the hands of individuals and making centralised servers responsible. According to the survey, 40% of respondents had managed their drill-hole and geological data on a centralised server with a folder or file structure, while 51% managed geophysical and other survey data this way. Survey respondents also relayed that they wanted tighter control over their exploration data and a more efficient workflow. Two out of three respondents would prefered a single, commercially available platform or an in-house solution instead of having to outsource their data management, hire consultants, or enable users to manage their own data (4).

The mine owner or mining rights holder should consider the option of contracting mining in cases where:

- Direct costs of production / exploitation represent a significant portion of the total cost of the mine,
- mine life is not more than 5 (five) years and there is a need to maintain a reasonable and constant production capacity,
- A lack of or failure to have trained, experienced workforce and staff in the area where mine works and exists.

Contract mining becomes significant, determined and effective practice in the last decade of the last century. Subcontractors are offering mine owners or concessionaires a number of advantages for the projects that have:

- short mine life,
- widely varying production capacity,
- limited possibilities of borrowing or low credit standing,
- complex joint venture arrangements (joint venture),
- lack of experience in mining production,
- mining policy without tradition or,
- rigid labor market.

S F J Dunlop presents historically overview of the contract mining development of. With the advent of the resources' boom in the late 1960s, most of the larger scale resources projects were commissioned on an 'owner-mining' basis, where the mine owners owned and operated the primary mining equipment. Then, in the 1980s with the arrival of the gold mining boom a change occurred – most of the mining activities were carried out on a contract mining basis. Over the last ten years, however, there has been a marked trend amongst larger open pit mining operations back towards owner-controlled and operated mining operations. The reasons for this are many but centre on the concept of the owner being prepared to assume more of the mining risk in exchange for the advantage of not having to pay a contractor's profit margin (5).

Where mining contracts are entered into, the older 'schedule of rates' format is being replaced with less adversarial and more entrepreneurial contract alternatives involving operating cost and risk sharing in a spirit of 'fair dealing'. Nevertheless leasing and maintenance issues have also been influenced in recent years by the advent of more flexible contract arrangements with equipment suppliers .Group purchasing has also had some influence for project operators. (5).

Dunlop in analyzing the requirements for and against the utilization of contract mining further states that for many mine owners, the optimal risk management profile involves a practical off-setting of the assumption of mining risk on the one hand, against the removal of litigation risk and potentially reduced operating costs on the other. Potential pitfalls may arise where equipment size and type requirements vary with time or where manpower management, mining flexibility, job skills or safety exposure factors are relevant. These issues are commented on in the context of some well-known examples, and an update is provided on contract mining versus owner operation in large scale Australasian open pit mining generally (5).

On the other hand, A Douglas is assessing the perspective of contractual mining and this puts into question the productivity of exploitation. “When we talk about productivity, there are some basic attitudes: "Productivity is the ratio of inputs to outputs". The larger the ratio the greater the productivity. This statement is deceptively simple and the concept and measurement of productivity is more complex. Physical measures such as tons or metres per man month are not necessarily all accounting. "If we use money as a medium of conversion and do not confuse it with cash. Then we can express productivity as the relationship between cost and worth per unit". This still does not encompass the whole concept as safety, reliability, quality and rate of output are also variables in the equation”.

To clarify the matter further productivity is virtually analogous to profitability. Profit is a phenomenon with parallels in nature. All life must make a profit, if the outputs do not exceed the inputs the organism must live off its own reserves or consume its capital and instead of growing will wither and disappear (2).

Historically contractors have been employed during the pre-production phases of projects i.e. prospecting, infrastructure construction, shaft sinking, ore body access development and have disappeared from the picture by the time that stoping is in progress. In producing mines their services have been restricted to specialized activities/tehnological phases: drilling, water control, support functions and raise boring. The existence of a large robust contracting industry is proof in itself that it is productive and provides these specialised services effectively, efficiently and to the satisfaction of their client body, the mines (2).

Overall productivity and profitability largely depend on the manner and intensity of engagement of workers-miners. Workers at the subcontractors are aware that their incomes depend directly on their engagement and performance, and this is basically the main difference in the productivity and profitability of workers engagement with the mine owner or contractor.

7. CONCLUSIONS

There continues to be a major role for contract mining where contractors have demonstrated that they have the necessary skills and expertise, can provide flexibility in the use of differing mining equipment and variable mining rates, are prepared to accept a reasonable amount of technical and operational risk and are competitive in pricing.

Owner mining should be compared to contract mining for larger, longer-life mining operations and a detailed analysis of the corporate, operational, cost and risk issues should be carried out.

Corporate issues include the best use of and return on capital, the company structure of the operating entity and project specific factors. The key operational issues involve people, equipment and ore mining quality control. Confidence in achieving similar production performance and efficiencies to that of contractors is a key risk and the need for detailed transition or implementation planning should not be underestimated. Changing from contract mining to owner managed mining is more often considered where mining costs are a significant portion of the total operating costs.

In analysing what makes up the mining costs there may be an opportunity to reduce margins or fees, without significantly increasing risk. For example the mine owner could directly manage some or all of the major supply contracts, such as equipment leases, fuel supply and explosives supply, while still employing a mining contractor to manage the other mining functions and employ the operating personnel.(1).

The concept of contracting has proved itself with regard to convenience, certainty, competitiveness of cost and client acceptance. The practice has taken root and the barriers to entry for this type of work are low. There is little requirement for specialised and costly plant and equipment and the skills required are fairly readily available or easily acquired.

The mining industry and the satellite contracting community are both fairly mature and the latter exists for the benefit of the former. Adversary situations have occasionally developed in the past and soured relationships between contractor and client. The development of symbiotic relationships which are characterised by cooperation for the mutual benefit of the contracting parties, will I believe become the norm in the future (2).

In terms of mining in the Republic Srpska, Bosnia and Herzegovina and neighboring countries, the utilization of contract mining is still limited to the organization of individual technological phases in this way in the mines. There are still can not encounter cases where a company has a concession rught to a mineral raw materials and the location fully organize production on the basis of contract mining. With the aspect of the concession policy, method of preparation and evaluation of feasibility studies, working modes and fit-out of company engaged in contract mining in this region of the Western Balkans, the question of viability and sustainability of this way of organizing production is present.

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DOI: 10.7251/BMC170701279C

ASSESSMENT OF THE DIGGING FORCE FOR UNDERWATER COAL MINING

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ABSTRACT

Lack of geomechanical parameters for the digging force in underwater mining conditions was noticed during analyzing the available technical documentation at preparatory phase for the revitalization of the vessel bucket wheel excavator "Kovin" vital parts. This fact has caused the need for the implementation of relevant research program that will determine the water pressure effect on the coal mining process. The research program included certain geomechanical laboratory tests which are adapted to the specific conditions and needs of underwater coal mining. The program had included coal laboratory testing of the linear and surface digging force in underwater conditions at different hydrostatic pressures of the water column. Intention of this laboratory testing program was to simulate underwater coal mining conditions at different depths of the excavation – digging according to the "in situ" principle.

Key words: digging force, underwater mining, coal

1. INTRODUCTION

Continuous technology is used for overburden, gravel and coal exploitation on this deposit. Overburden and gravel excavation is carried out with the vessel bucket-chain excavator "5630", the remaining quantities of gravel and sand, which are not within the reach of the vessel bucket-chain excavators, as well as coal are excavated by means of the vessel bucket-wheel excavator model UCW 450, so called "Kovin" [1]. The analysis of the available documentation necessary for the revitalization of the vital parts of the vessel bucket-wheel excavator "Kovin", showed the lack of geomechanical parameters which necessary to determine the influence of the aquatic environment pressure on the work environment - coal, and it promoted the necessity for appropriate research of the water column pressure influence to the digging force value.

2. GEOLOGICAL STRUCTURE OF COAL AND GRAVEL DEPOSITS, KOVIN (COAL DEPTH)

Coal and gravel deposit "Kovin" represents the northern part of the unique sedimentation area Kostolac-Kovin, which is separated by the Danube river. Gravel deposit stratigraphically belongs to the pleistocene, and it was developed in the riverbed facies. Its evolution is being continuously monitored at the entire exploration area. Kovin coal deposit belongs to the stratigraphic unit of the upper miocene or pontian actually his younger part - top pontian. In this area of the substratum section of the miocene part are pontian and pannonian formations, while overlying strata corresponds to quaternary formations [1], Figure 1.

3. WATER PRESSURE ON THE MINING LEVEL

The conditions under which it is necessary to determine the value of digging force should as much as possible to simulate the conditions that are present during the process of the excavation "in-situ" and they are determined on the basis of the depth analysis at which the coal exploitation is carried out. The maximum depth of which is necessary to perform the test is determined on the basis of the fact that the coal layer is characterized by frequent thickness change, and for safety instead of the second coal layer average depth of 50 m, maximum depth of 60 m was employed.

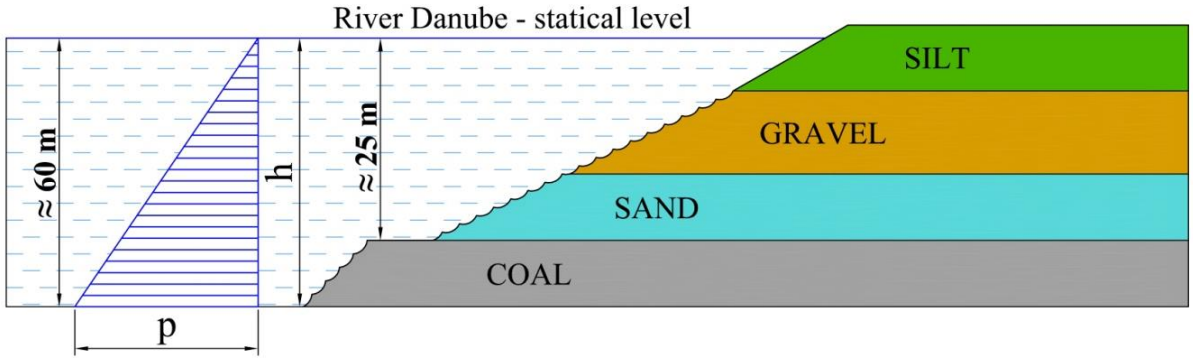


Figure 1. Schematic representation of the geological coal and gravel deposit "Kovin" with showed pressure changes with increasing depth

The conditions that exist in the process of coal excavation in this deposit were analyzed for the steady water (fluid) state, which may be subjected to the basic laws of hydrostatics as part of fluid mechanics. In any liquid that is under influence of the earth's gravity, pressure within itself creates, and it does not depend on the amount of fluid in the vessel, but on the height of the liquid column. The foregoing implicates that with increasing of observed point depth in the fluid there is an increase of liquid weight which is located above this point, and therefore the pressure increases. In order to determine the pressure values at the given point on a particular depth in the liquid it is necessary to determine the hydrostatic pressure using Torricellis formula (Evangelista Torricelli, 1608-1647):

$$P = \rho \cdot g \cdot h$$

where: ρ – liquid density [kg/m^3],
g – gravity acceleration [m/s^2],
h – depth [m].

If product of the liquid density (ρ) and the gravitational acceleration (g) is replaced by a unit weight (γ) the hydrostatic pressure equation will be in the following form:

$$P = \gamma \cdot h$$

where: γ – liquids unit weight (kN/m^3),
h – depth (m).

Hydrostatic pressure change with the observed point depth increase, in this case point at which coal is excavated, is graphically shown in Figure 2.

Hydrostatic pressure in the given point at a certain depth in the fluid acts equally in all directions. Considering the structural and physical properties, particularly fracturing and porosity, in the wider zone of the excavation a coal layer is fully saturated with water. The water influence, in terms of external loads, can be defined as the coal is immersed in the liquid at depth at which excavation is carried out, whereby the pressure of the water column, according to the law of hydrostatics acts uniformly in every direction, Figure 2.

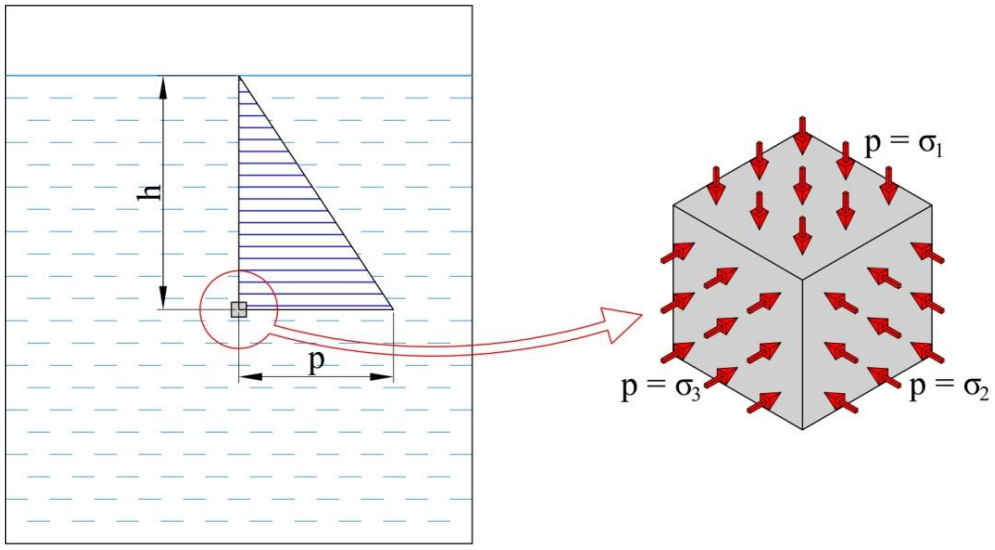


Figure 2. Influence of water on the coal during the excavation

4. METHODOLOGY TESTING OF DIGGING FORCE

Specific digging force for rock material (coal) was determined using the test methods supplemented with a wedge that has been proposed by the company Orenstein und Koppel. For this purpose we used specially designed wedge that was used for wedging the coal specimens. The wedge has lateral sides slopes of 17^o to the longitudinal axis, a blunter tip of width b = 5 mm and the wedge length l = 65 mm, figure 3.

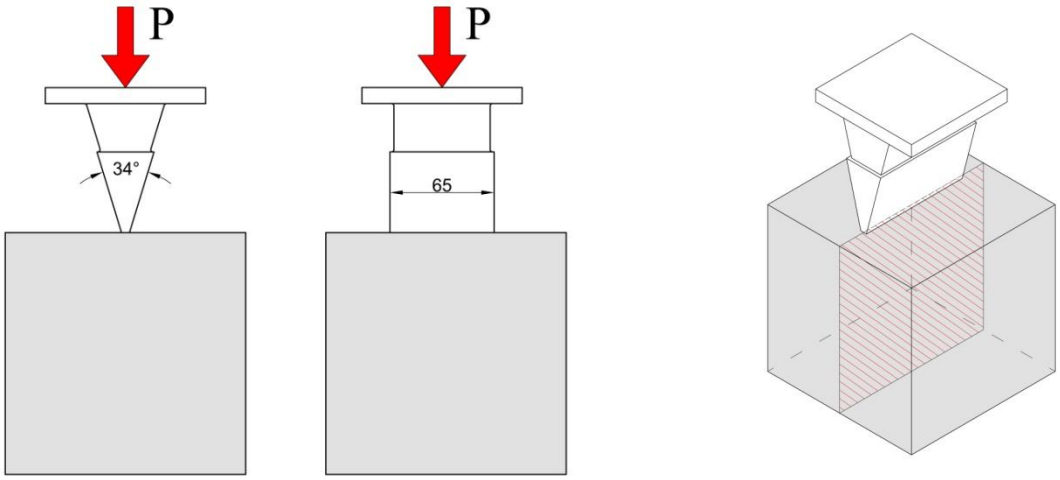


Figure 3. Schematic representation of the wedge and its position while testing specific cut resistance in the primary method

At the original method the wedge for testing is being installed into a suitable hydraulic press which is used for vertical force applying. The vertical force increases constantly until the specimen fracture. During the tests, in addition to determining the value of the force which led to the specimen fracture, it is being determined the size of the wedge penetration into the speimen [2]. Value of specific line cut resistance K_L and surface cutting force, K_F are calculated according to the following equations:

$$K_L = \frac{P}{L} \quad (\text{N/cm}') \quad \text{i} \quad K_F = \frac{P}{F} \quad (\text{N/cm}^2),$$

where: P – breaking off force (N),
L – length of the wedge which was involved in the breaking (cm´),
F – specimens cross-section area (cm^2).

Digging force testings for the underwater excavation purpose were performed with extended wedge testing method which allows the same "in-situ" conditions which exist during the coal excavation in this deposit. The influence of water and excavation depth are defined in terms of external loads so that the sourroundig pressure value is constant and that it acts uniformly in all directions, Figure 4.

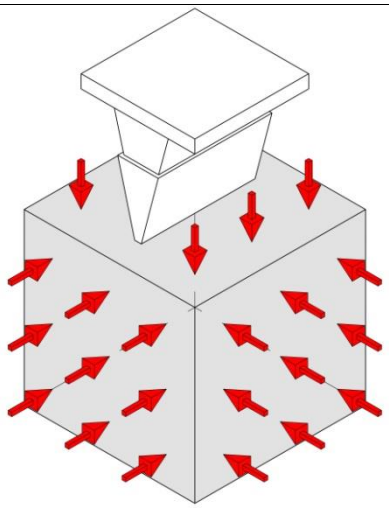


Figure 4. The position of the wedge during the examination of the specific cutting resistance in revised method

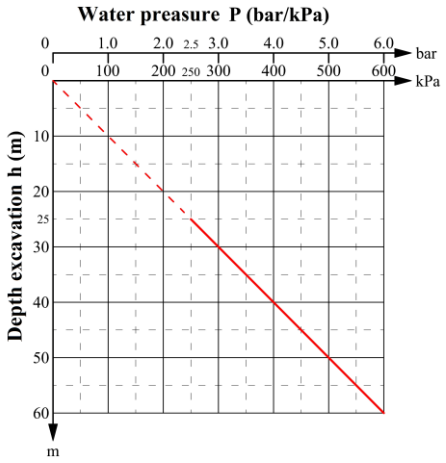


Figure 5. Hydrostatic pressure variation in water with an increase in the depth at which excavation is carried out

The values of water pressure for which it is necessary to carry out tests determined according to equation:

$$P = \gamma \cdot h\,,$$

where: γ – water unit weight, $\gamma \approx 10$ [kN/m^3],
h – excavation depth, h = 25 – 60 [m].

Pressure interval for digging force test is set according to the depths at which coal excavation is performed, and it ranges from 25 to 60 m, and is:

$$P_{\min} = 10 \cdot 25 = 250 \text{ (kPa)} \approx P_{\min} = 2.50 \text{ (bar)},$$

$$P_{\max} = 10 \cdot 60 = 600 \text{ (kPa)} \approx P_{\max} = 6.00 \text{ (bar)}.$$

Diagram of the pressure changes with increasing water depth, as well as the pressure interval in which the digging force tests were conducted are shown in Figure 5.

5. TESTING APPARATUS

During the tests it was necessary to provide a water pressure p which exist at the coal exploitation depth (h = 25 to 60 m), and accordingly the water pressures p during testing were p = 250 to 600 kPa ($p \approx$ 2.50 to 6.00 bar). Given that the excavation is performed in hydrostatic pressure conditions at a certain depth in the water, values of the principal stresses are equal and their value is $\sigma_1 = \sigma_2 = \sigma_3 = p = 250 \div 600$ kPa.

Digging force testings in coal underwater exploitation conditions were carried out using the equipment for triaxial testing of soils that was modified to fulfill needs. The main task of triaxial cell is to provide a sourrounding preasure and to maintain this pressure constant during the entire course of the tests (σ_3 = const), Figure 6. The same pressure conditions are required for performing the digging force test in defined terms. In existing triaxial cell for the

maximum specimen diameter of d = 150 mm is additionally installed wedge for digging force testing according to the method that has been proposed by the company Orenstein und Koppel, in Figure 3. The appearance of modified testing equipment is shown in Figure 7.

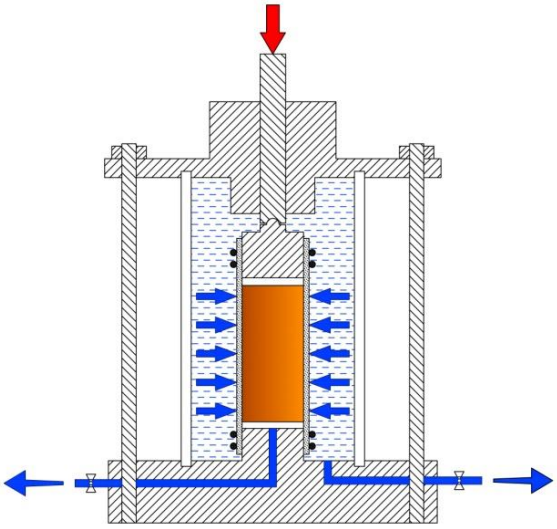


Figure 6. The basic principle and equipment for triaxial test performance

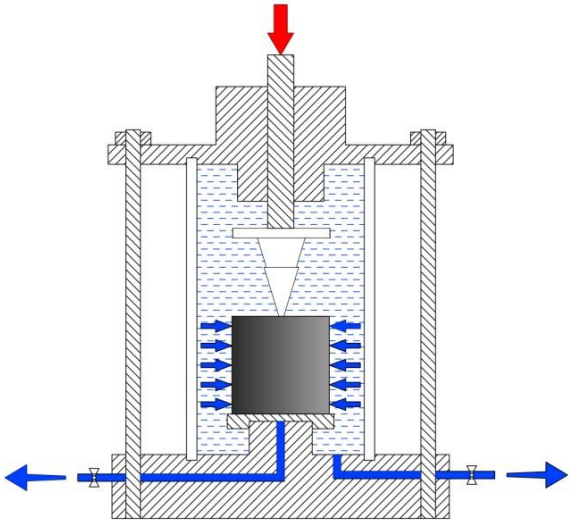
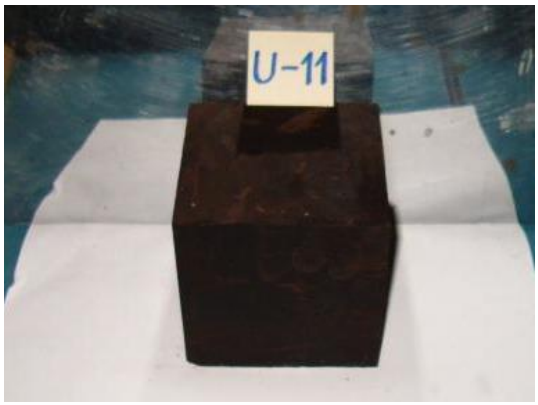


Figure 7. The design of equipment suitable for carrying out digging force testings in terms of the underwater coal excavation

6. TESTING PROCEDURE

Digging force testings in underwater exploitation conditions were carried out on prismatic specimens, which height was h = 100 mm, width a = 100 mm and length b = 100 mm (diagonal of the base specimen was about 141 mm), which are placed at the center of the triaxial cell base. During test the specimen was not isolated with rubber membrane, because the material (coal) in natural conditions is also exposed to the water at a certain depth. After installation of specimen and triaxial cell closure, cell was loaded with distilled water, and then the load application was carried out in two phases. In the first stage sourrounding lateral load σ_3 was applied and it was kept constant during the entire course of the test (σ_3 = const). In the second phase, the axial load σ_1 was gradually increased until fracture occures. The specimen before and after examinations is shown in Figure 8.



Specimen after tests



Specimen before tests

Figure 8. Specimen appearance before and after digging force tests in underwater coal excavation conditions (sample labeled U-11)

The presented digging force testing procedure in underwater coal excavation conditions was used for tests in the aquatic environment with a gradual increase of hydrostatic pressure, regarding to lateral load was σ_3 = 0,0 - 2.5 - 3.5 - 4.5 - 6.0 bar. Results of carried out laboratory tests are shown in the Table 1.

Table 1. Test results of deggining force tests for all defined lateral pressures σ_3 *(SAMPLE LABELED U-11)*

Sample tag	***Specific cutting force depending on the lateral pressure σ_3 (bar)***					
		0.00	***2.50***	***3.50***	***4.50***	***6.00***
U – 11	K_L *(N/cm')*	873.46	936.33	963.52	990.71	1029.80
	K_F *(N/cm^2)*	57.38	61.73	63.48	65.24	67.80

7. CONCLUSION

For the purpose of revitalizing the cutting wheel of vessel bucket wheel excavator "Kovin" (UCW-450), it was necessary to make a appropriate geomechanical laboratory research with an emphasis on cutting resistance testing in underwater exploitation conditions. Using a gradual increase of the hydrostatic pressure during the test, the working conditions of coal exploitation at different depths of the excavation were simulated. According to depths which are used for coal excavation, the pressures of the water column of about 2.5 to 6 bar were provided, and during the actual test lateral loads σ_3 = 0,0; 2.5; 3.5; 4.5; 6.0 bar were applied. Digging force tests in coal underwater exploitation conditions were carried out using the equipment for triaxial testing of soil that was modified for these purposes. With quoted equipment and depicted method it is possible to provide the water pressures which exists at the certain coal excavation depth, so that the results of digging force tests can be considered authentic.

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DOI: 10.7251/BMC170701287S

MATHEMATICAL MODEL OF STACKING OPERATION OF STACKER/RECLAIMERS

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ABSTRACT

This paper presents the methodology defining key parameters of the stacking operation of rail-mounted bucket wheel stacker/reclaimers for raw bulk material handling systems.

Due to gaps in the theory on this topic, the study aims at developing a mathematical model which characterizes the stacking operation of these machines. It integrates the relevant relationship between design characteristics of the equipment and the parameters associated with its work environment.

The chevron stacking method has been analysed and a mathematical model of the process is developed. The model is useful for a blending or homogenizing operation at a stockpile, as well as for automation of a stockpiling process. Furthermore, the model accurately defines operational parameters of this process which would enable an improved utilization of the machine and a better operation management.

Successful validation of the developed model was performed for an operating iron ore mine.

Key words: bucket wheel stacker/reclaimers, bulk material handling systems, stacking operation, mathematical model

1. INTRODUCTION

Handling and shipment of bulk solid materials plays a vital role in the modern economy. It involves a steadily increasing volume of raw materials being transported from often remote mine sites world-wide to treatment facilities in both developed and developing countries. Due to significant costs of those operations bulk solids handling systems ought to be efficient, reliable, and highly productive and should enable a good material flow at the least cost. This requires more efficient reclaiming and stacking machines.

Ground storage is the most economical method of stockpiling solid bulk material, particularly for a large tonnage. This way, one minimizes the investment and operation/maintenance costs of structures and auxiliary equipment. This layout is used in a variety of industries. The role

of large-scale raw bulk materials handling systems using stockyards consists of the following [[4],[4],[5],[7],[8]]:

- Transfer function: Facilitate a transfer between two or more means of transport;
- Buffer function: Even out capacity fluctuations between supply and destination facilities;
- Quality function: Even out quality fluctuation of the same raw bulk material on a stockpile.

The equipment employed to handle raw bulk materials in stockyards is referred to as a reclaimer (reclaiming operation only), and/or stacker/reclaimer (both functions: stacking and reclaiming). The most frequently used ones are the rail-mounted boom-type machines, such as a stacker/reclaimer shown in Figure 1. To simplify, these machines will be referred to as "R&SR".

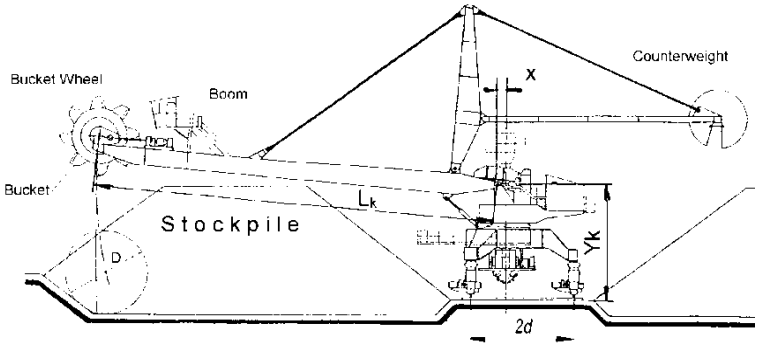


Figure 1. Rail-mounted bucket wheel stacker/reclaimer with a slewing boom [[4],[4],[7],[8]]

During the *reclaiming operation* the bucket of such machines follows a 3D trajectory defined by a combination of three elementary movements: a) rotation of the bucket around its own axis; b) slewing of the boom around a vertical axis, and c) travel of the machine on rails parallel to the pile.

The *stacking (stockpiling)*, as the second operation, can be carried out only by stackers and stacker/reclaimers. The latter are bucket wheel machines in which the wheel boom conveyor is reversible. The stockpile conveyor runs in opposite directions for stockpiling and reclaiming. By this means the belt can convey material from the bucket wheel to the slewing axis, or alternatively from the slewing axis to the bucket wheel (Figure 2).

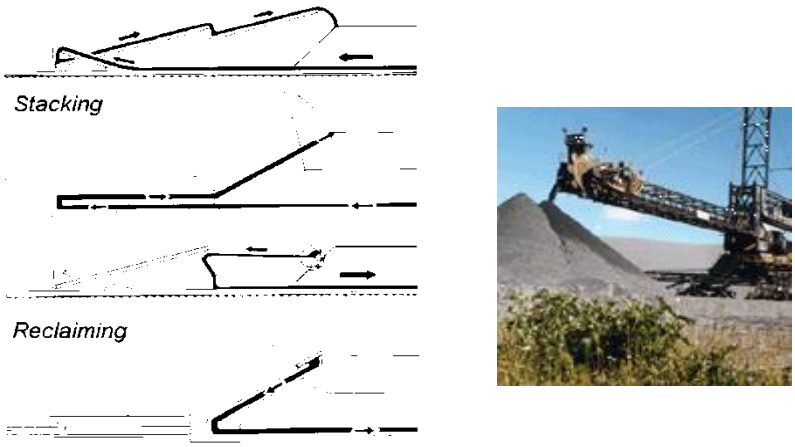


Figure 2. Flow of material when using a combined stacker/reclaimer [[4],[7],[8]]

During a stacking operation performed by a stacker/reclaimer, the material is fed to a belt (as in the case of a typical stacker), and its bucket wheel does not rotate. The machine travels alongside the stockyard where bulk material is stockpiled. In general, combined machines are provided with greater stacking than reclaiming capacity. This way, the time lost in reclaiming from the stockpile is kept at a minimum. It is of great importance to the subsequent production process.

In the scientific/technical literature, there are gaps in the theory regarding the stacking operation of R&S/R. The current paper aims at developing a mathematical model which characterizes the chevron stacking operation of these machines. It integrates the relevant relationship between design characteristics of the equipment and the parameters associated with its work environment. These parameters are quite important for a blending or homogenizing operation at a stockpile, as well as for automation of a stockpiling process. In this case, the boom position in stockpiling operation should also be defined. Furthermore, the paper defines operational parameters of this process which would enable an improved utilization of the machine and a better operation management.

The case study performed at the Omarska iron ore surface mine demonstrates the applicability of the methodology.

2. STACKING METHODS

Various methods are available for stacking bulk materials to build up stockpiles of particular dimensions. In general, three methods of stacking are used in practice for the stockpiling of bulk solids, such as:

- Chevron stacking (Figure 3a)
- Windrow stacking (Figure 3b)
- Combined stacking (various combinations of the two basic stacking methods)

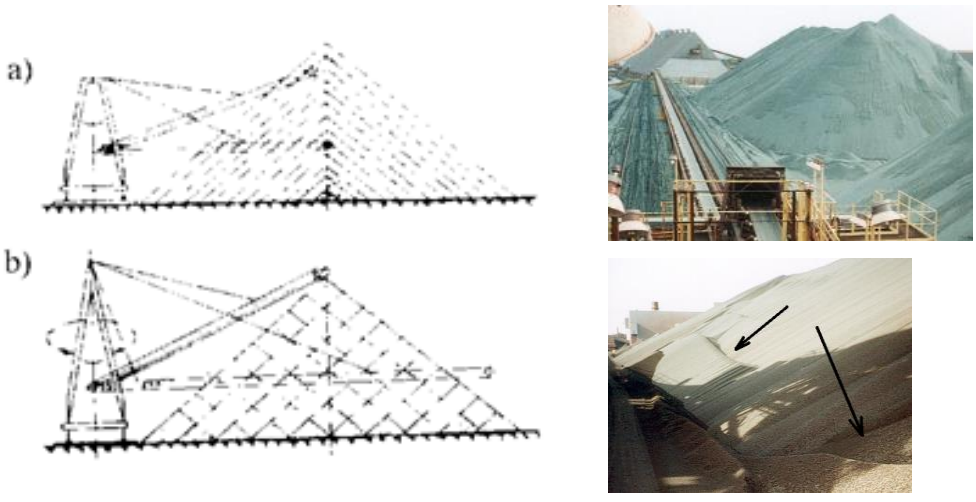


Figure 3. Stacking methods [[4],[7],[8]]
a) chevron; b) windrow

The chevron stacking is suitable if materials of even lump sizes are handled. Otherwise, segregation of lumps would occur (large lumps will roll down the side of the pile and collect at its base). The windrow stacking is preferred to chevron stacking since it provides a better

blending and homogenization, even for materials of varying lump sizes, can be achieved with this method. The selection of a particular method depends on the blending and/or efficiency desired and the type of reclaiming to be adopted.

3. GEOMETRICAL PARAMETERS OF THE CHEVRON STOCKPILING OPERATION

In order to define operational parameters of a chevron stacking operation, it is necessary to define the main geometrical parameters of a stockpile cross-section as a function of a stacking method. The approach is based on previously conducted works in this area [[4],[7],[8]].

3.1 Geometrical parameters of the chevron stacking method – triangular cross-section of the stockpile

The chevron method stacks material at the center discharge peak. Each subsequent layer completely covers the previous layer thus forming a chevron cross-section (Figure 3a). During the chevron stacking operation, each successive layer has a larger volume (Figure 4).

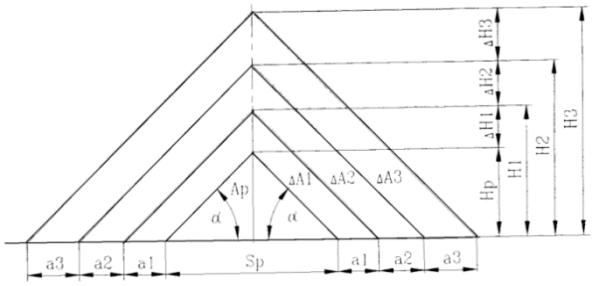


Figure 4. Relationship between main geometrical parameters of the chevron stacking method – triangular cross-section

To maintain a constant layer thickness, the travel speed of the stacker should be variable; it should decrease as the layers build the pile. The chevron method represents the most economical approach to homogenizing material whose lump size is consistent. The single line discharge at the peak of the pile enables the use of a short luffing boom stacker (stacker/reclaimer). In the further analysis, the main geometrical parameters of this stacking method are defined.

- Area of the first stockpiled triangle A_p

According to Figure 4, it may be written:

$$A_p = 0.25 \cdot S_p^2 \cdot \tan\alpha \tag{1}$$

The same parameter may be expressed through the stacking output as follows:

$$A_p = \frac{Q_{st}}{60 \cdot v_{tr(\max)}} \ [m^2] \tag{2}$$

Where:

V_{tr} [m/min] – travelling speed of the machine

Q_{st} [m^3/h] – stacking output of the machine

The first layer is built with a maximal travelling speed of the machine.

From equations (1) and (2), one obtains:

$$S_p = \sqrt{\frac{Q_{st} \cdot \cot\alpha}{15 \cdot v_{tr(\max)}}} \ [m] \tag{3}$$

- Cross-section areas of subsequent triangles A_n

They are calculated as follows for nl^{th} triangle (Figure 4):

$$A_{nl} = 0.25 \cdot \left(S_p + 2 \cdot \sum_{i=1}^{nl} a_i \right)^2 \cdot \tan\alpha \ [m^2] \tag{4}$$

Where:

α [^{0}] – angle of repose of the material

- Cross-section area increase ΔA_{nl}

The area increase of each stockpiled triangle (layer) represents the difference between the current triangle and the previous one. It may be written as follows:

$$\Delta A_{nl} = a_n \cdot \left(S_p + a_{nl} + 2 \cdot \sum_{i=1}^{nl-1} a_i \right) \cdot \tan\alpha \ [m^2] \tag{5}$$

The travelling speed in each layer may be determined as follows:

$$v_{tr} = \frac{Q_{st}}{60 \cdot \Delta A_{nl}} \left[\frac{m}{min}\right] \tag{6}$$

After defining the width of each stockpiled triangle, their height should also be determined. This is necessary for a whole determination of parameters for this stacking mode.

- Height of the first stockpiled triangle H_p

According to Figure 4, its value is calculated as follows:

$$H_p = 0.5 \cdot S_p \cdot \tan\alpha \ [m] \tag{7}$$

- Height of n_{lth} layer

$$H_{nl} = \left(0.5 \cdot S_p + \sum_{i=1}^{nl} a_i \right) \cdot \tan\alpha \ [m] \tag{8}$$

If the thickness of all layers is constant (a), H_{nl} is calculated in the following manner:

$$H_{nl} = \left(0.5 \cdot S_p + nl \cdot a_{nl}\right) \cdot \tan\alpha \ [m] \tag{9}$$

- Height increase ΔH_{nl}

According to Figure 4, this value is calculated as the difference between two neighboring layers. At the same time this increase represents the necessary boom-end hoisting to stack a new layer. It is calculated as follows:

$$\Delta H_{nl} = H_{nl} - H_{(nl-1)} = a_{nl} \cdot \tan\alpha \ [m] \tag{10}$$

In the case where, due to technical and operational requirements, it is necessary to stack under condition $\Delta A_{nl} = A_p$, the value of (a_{nl}) is calculated as follows:

$$a_{nl} = 0.5 \cdot \left(\sqrt{nl+1} - \sqrt{nl}\right) \cdot S_p \ \ [m] \tag{11}$$

After determining the geometrical parameters of this stacking mode, it is necessary to define parameters of the boom-positioning during this operation. For this purpose, Figure 5 is analyzed.

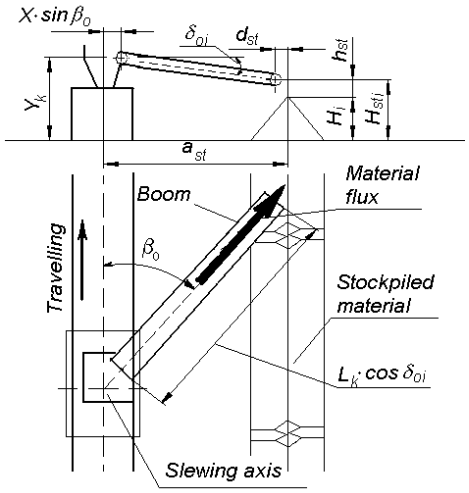


Figure 5. Parameters of the boom-positioning during chevron stacking method [[4],[7]]

- Height position of the boom-end during the stacking H_{sti}

During the stacking operation the boom-end shall not touch stockpiled material in order to ensure an interrupted and safe operation. Therefore, it should be positioned at a safety distance h_{st} over the stockpile peak. The height of the boom – end is calculated as follows (Figure 5):

$$H_{sti} = H_i + h_{st} \ \ [m] \tag{12}$$

This parameter should be calculated for each stockpiled layer.

- Angle of the boom-inclination δ_{oi}

In a stacking operation, the boom is inclined at a certain angle. It changes with each stockpiled layer and is calculated in the following manner (Figure 5):

$$\delta_{oi} = \arcsin \frac{H_{sti} - Y_k}{L_k} \ \ [^0] \tag{13}$$

For the first stockpiled triangle, and lower layers, this value is usually negative.

- Distance between the slewing axis of the machine and the stockpile center a_{st}

According to Figure 5, it is a distance which remains constant in the chevron stacking method. Its value is calculated as follows:

$$a_{st} = \sin \beta_0 \cdot \left(X + L_k \cdot \cos \delta_{0i}\right) + d_{st} \ \ [m] \tag{14}$$

For a stacking operation, the distance (a_{st}) is usually known. The value to be calculated is the boom slewing angle β_0. After the transformation of equation (14), one obtains:

$$\beta_0 = \arcsin \frac{a_{st} - d_{st}}{X + L_k \cdot \cos \delta_{0i}} \ [^0] \tag{15}$$

The angle β_0 changes slightly according to the change in the stockpile height.

By determining these geometrical and design parameters, all operational elements concerning the chevron stacking method are defined for the triangular cross-section stockpile.

3.2 Geometrical parameters of the chevron stacking method – trapezoidal cross-section of the stockpile

The previous analysis was carried out for a triangular cross-section of a stockpile. In the case where the allowed stockpile height is reached, and one has at the disposal an additional surface remains available, new layers can be added on the existing stockpile side (Figure 6).

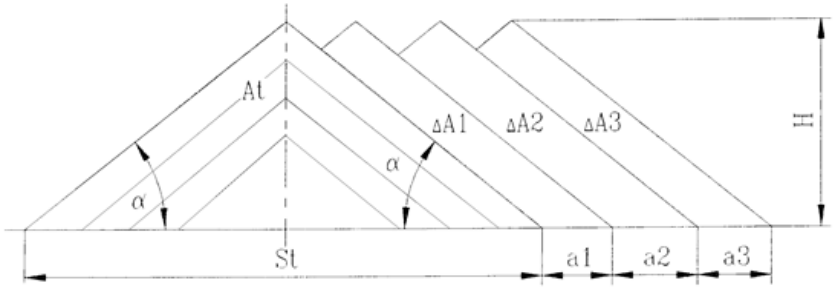


Figure 6. Relationship between main geometrical parameters of the chevron stacking method – trapezoidal cross-section

That stockpiling results in a trapezoidal cross-section shape. Mathematical relationships obtained above are still valid for this cross-section shape. However, it is necessary to add some particular clarifications with regard to this working mode.

The following parameters ought to be determined:

- Area of a triangular cross section A_t

Total area of stockpiled triangle is calculated as follows:

$$A_t = 0.25 \cdot S_t^2 \cdot \tan\alpha \ [m^2] \tag{16}$$

- Extension area increase in a trapezoidal cross-section ΔA_{ntr}

$$\Delta A_{ntr} = a_{ntr} \cdot \left(H - 0.25 \cdot a_{ntr} \cdot \tan\alpha\right) \ [m^2] \tag{17}$$

The parameter (a) represents the thickness of additional layers in a trapezoidal cross-section in the chevron stacking method. Its value depends on operational and design parameters of this operation. If the value of an area ΔA is previously known, the value of (a) is calculated by solving formula (17):

$$a = 2 \cdot \left(H - \sqrt{H^2 - \Delta A \cdot \tan\alpha}\right) \cdot \cot\alpha \ [m] \tag{18}$$

The travelling speed of the machine is calculated as per equation (6).

- Boom slewing angle regarding additional layers in trapezoidal cross-section β_{0i}

For each additional layer in a trapezoidal cross-section, the boom slewing angle also changes. The same relationships as those shown in Figure 6 remain valid with a slight modification regarding these additional layers. The slewing angle for each additional layer is calculated as follows:

$$\beta_{0i} = \arcsin \frac{a_{st} + \sum_{j=1}^{i} a_j - d_{st}}{X + L_k \cdot \cos \delta_{0i}} \; [^0] \tag{19}$$

By determining these geometrical and design parameters, all working elements concerning the chevron stacking method for a trapezoidal cross-section are defined. As in the previous case, they may serve as a basis for the planning and automating of this operation.

4. CASE STUDY

The case study has been carried out at the "Omarska" surface iron ore mine. The iron ore bed "Omarska“ is located in northeast part of Ljubija metal-genetic area in northwest Bosnia. It consists of two mutually separated Surface pits, SP "Jezero" and SP "Buvac" [[1],[2],[3]]. The ore preparation technology is based on gravity – magnetic separation of limonite iron ore with average quality of 50.50% Fe, 12.50% SiO_2, 12.00% H_2O. Within process of washing, crushing, multiple screening, classifying on spiral classifiers and hydro-cyclones as well as treatment of fine fractions on high intensity electromagnetic separators two commercial products are prepared:

- Lump ore concentrate -APR, fraction size - 40+10 mm, average quality 54,00% Fe, 7,50 % SiO_2,7,00% H_2O; angle of repose: 37^0; material bulk density loose: 1.5 t/m^3
- Fine ore concentrate -BPR fraction size - 10+0 mm, average quality 52,00% Fe, 11,00 % SiO_2,12,00% H_2O; angle of repose: 37^0; material bulk density loose: 1.7 t/m^3

The ratio Lumps: Fines = 45:55 with mass recovery of approximately 80 % and Fe recovery of 83 % of the total quantity of around 1.5 million tones yearly [[1],[2],[3]].

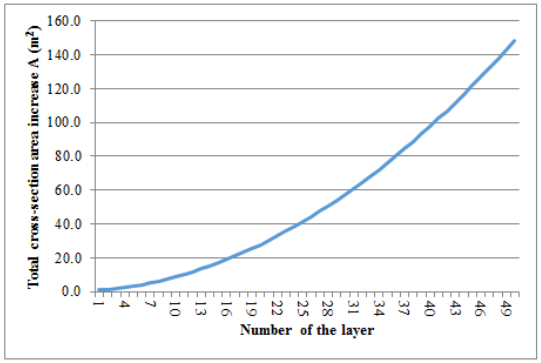
The mine uses a stacker/reclaimer "DELATTRE LEVIVIER" which main design and operational characteristics are presented below:

- Boom length: 28.0 m; Stacking output: 1000 t/h; Maximum travelling speed: 30 m/min;
- Wheel diameter: 6.4 m; Number of buckets: 8; Volume of a bucket: 0.4 m^3 (cell-less buckets);
- Bucket wheel rotation speed: 5.0 rpm; Bucket wheel installed motor power: 75 kW
- Conveyer width: 1200 mm

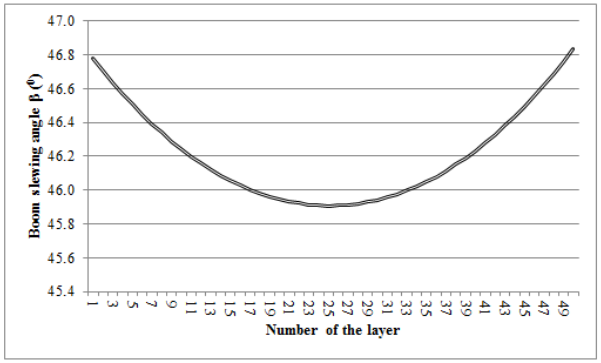
The stacking is typically performed using chevron method with layers of 15-25 cm in a normal operation and 8-12 cm in the final shaping phase of the stockpile. The calculated operational parameter of the stacking operation in 50 layers during a chevron stacking at the Omarska iron mine are presented in Table 1. Figure 7 graphically depicts the variation of some key operational parameters (cross-section area increase, boom slewing angle variation and travelling speed of the machine).

Table 1: Parameters of the chevron stacking operation

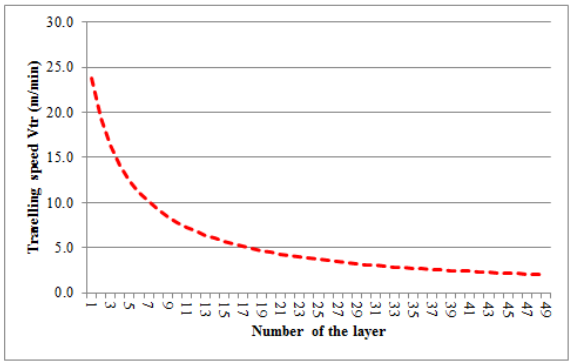
	Ap =	0,44	m²																								
	Sp =	1,54	m																								
Layer	Hp =	0,58	m							Width of the stockpile base												Boom inclination angle			Boom slewing angle		
1	ΔH1	0,2	m	H1	0,78	m	a1	0,27	m	2,07	m	A1	0,80	m^2	ΔA1	0,36	m^2	V_{tr1}	25,00	m/min	δ_{o1}	-9,92	°	β_{o1}	46,78	°
2	ΔH2	0,2	m	H2	0,98	m	a2	0,27	m	2,60	m	A2	1,27	m^2	ΔA2	0,47	m^2	V_{tr2}	23,82	m/min	δ_{o2}	-9,50	°	β_{o2}	46,70	°
3	ΔH3	0,2	m	H3	1,18	m	a3	0,27	m	3,13	m	A3	1,84	m^2	ΔA3	0,57	m^2	V_{tr3}	19,40	m/min	δ_{o3}	-9,09	°	β_{o3}	46,63	°
4	ΔH4	0,2	m	H4	1,38	m	a4	0,27	m	3,66	m	A4	2,52	m^2	ΔA4	0,68	m^2	V_{tr4}	16,37	m/min	δ_{o4}	-8,67	°	β_{o4}	46,57	°
5	ΔH5	0,2	m	H5	1,58	m	a5	0,27	m	4,19	m	A5	3,31	m^2	ΔA5	0,78	m^2	V_{tr5}	14,16	m/min	δ_{o5}	-8,26	°	β_{o5}	46,51	°
6	ΔH6	0,2	m	H6	1,78	m	a6	0,27	m	4,72	m	A6	4,20	m^2	ΔA6	0,89	m^2	V_{tr6}	12,47	m/min	δ_{o6}	-7,84	°	β_{o6}	46,45	°
7	ΔH7	0,2	m	H7	1,98	m	a7	0,27	m	5,25	m	A7	5,20	m^2	ΔA7	1,00	m^2	V_{tr7}	11,14	m/min	δ_{o7}	-7,43	°	β_{o7}	46,39	°
8	ΔH8	0,2	m	H8	2,18	m	a8	0,27	m	5,78	m	A8	6,30	m^2	ΔA8	1,10	m^2	V_{tr8}	10,07	m/min	δ_{o8}	-7,02	°	β_{o8}	46,34	°
9	ΔH9	0,2	m	H9	2,38	m	a9	0,27	m	6,31	m	A9	7,51	m^2	ΔA9	1,21	m^2	V_{tr9}	9,19	m/min	δ_{o9}	-6,61	°	β_{o9}	46,29	°
10	ΔH10	0,2	m	H10	2,58	m	a10	0,27	m	6,84	m	A10	8,82	m^2	ΔA10	1,32	m^2	V_{tr10}	8,44	m/min	δ_{o10}	-6,19	°	β_{o10}	46,24	°
11	ΔH11	0,2	m	H11	2,78	m	a11	0,27	m	7,37	m	A11	10,25	m^2	ΔA11	1,42	m^2	V_{tr11}	7,81	m/min	δ_{o11}	-5,78	°	β_{o11}	46,20	°
40	ΔH40	0,2	m	H40	8,58	m	a40	0,27	m	22,77	m	A40	97,66	m^2	ΔA40	4,50	m^2	V_{tr40}	2,47	m/min	δ_{o40}	6,11	°	β_{o40}	46,23	°
41	ΔH41	0,2	m	H41	8,78	m	a41	0,27	m	23,30	m	A41	102,27	m^2	ΔA41	4,61	m^2	V_{tr41}	2,41	m/min	δ_{o41}	6,52	°	β_{o41}	46,28	°
42	ΔH42	0,2	m	H42	8,98	m	a42	0,27	m	23,83	m	A42	106,98	m^2	ΔA42	4,71	m^2	V_{tr42}	2,36	m/min	δ_{o42}	6,93	°	β_{o42}	46,33	°
43	ΔH43	0,2	m	H43	9,18	m	a43	0,27	m	24,36	m	A43	111,80	m^2	ΔA43	4,82	m^2	V_{tr43}	2,31	m/min	δ_{o43}	7,34	°	β_{o43}	46,38	°
44	ΔH44	0,2	m	H44	9,38	m	a44	0,27	m	24,89	m	A44	116,73	m^2	ΔA44	4,93	m^2	V_{tr44}	2,26	m/min	δ_{o44}	7,76	°	β_{o44}	46,43	°
45	ΔH45	0,2	m	H45	9,58	m	a45	0,27	m	25,42	m	A45	121,76	m^2	ΔA45	5,03	m^2	V_{tr45}	2,21	m/min	δ_{o45}	8,17	°	β_{o45}	46,49	°
46	ΔH46	0,2	m	H46	9,78	m	a46	0,27	m	25,95	m	A46	126,90	m^2	ΔA46	5,14	m^2	V_{tr46}	2,16	m/min	δ_{o46}	8,58	°	β_{o46}	46,56	°
47	ΔH47	0,2	m	H47	9,98	m	a47	0,27	m	26,48	m	A47	132,14	m^2	ΔA47	5,24	m^2	V_{tr47}	2,12	m/min	δ_{o47}	9,00	°	β_{o47}	46,62	°
48	ΔH48	0,2	m	H48	10,18	m	a48	0,27	m	27,02	m	A48	137,49	m^2	ΔA48	5,35	m^2	V_{tr48}	2,08	m/min	δ_{o48}	9,41	°	β_{o48}	46,69	°
49	ΔH49	0,2	m	H49	10,38	m	a49	0,27	m	27,55	m	A49	142,95	m^2	ΔA49	5,46	m^2	V_{tr49}	2,04	m/min	δ_{o49}	9,83	°	β_{o49}	46,76	°
50	ΔH50	0,2	m	H50	10,58	m	a50	0,27	m	28,08	m	A50	148,51	m^2	ΔA50	5,56	m^2	V_{tr50}	2,00	m/min	δ_{o50}	10,24	°	β_{o50}	46,84	°



a) Total stockpile cross-section are increase



b) Variation of the boom slewing angle



c) Variation of the travelling speed of the machine

Figure 7. Variations of some operational parameters during the stacking operation

5. CONCLUSION

The paper presents a detailed mathematical model of the chevron stacking method. It integrates the applicable relationship between design characteristics of the machine and the parameters associated with its work environment (geomechanical characteristics of the bulk raw material and geometrical characteristics of the stockpile).

The case study has been performed at the iron mine Omarska. The obtained results demonstrate the applicability of the proposed model. It could be used in refining and optimizing stacking operation at a specific mine site.

The future research work should aim at developing models for other types of stacking methods (windrow and combined) as well as their impact on the blending/homogenizing efficiency with regard to the raw material quality and lump size.

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DOI: 10.7251/BMC170701297B

RISK MANAGEMENT ON THE INNER LANDFILL OF SURFACE MINE "TAMNAVA – WEST FIELD"

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ABSTRACT

The greatest engineering challenges in the process of exploitation of lignite began after the flooding at the SM "Tamnava – West field" in May 2014. Specific geotechnical conditions required changes in technology on excavation and landfill, and the task in order to continue with production of coal was solved with methods which exceeded former theoretical knowledge. All solutions are checked with geotechnical investigation works, geostatic calculations and with geodetic monitoring of problematic field. The main objective of this paper is to show ways to overcome the shortcomings of previously designed solutions. At this moment, mullock material is used for ensuring the stability of temporary slopes and for extrusion of watery and muddy material in landfill toe area.

Key words: risk management, landfill, remediation

1. INTRODUCTION

Formation of the inner landfill in SM ”Tamnava-West Field” started 18 years ago. The site covers an area of 8.50 km^2 and is shown in the figure 1 between the internal lines: OE and PO (west – east direction) and 172.5 to 142.5 (north – south direction). With the effected height of 142 masl, the height difference of 132 meters in relation to the deepest open level within the coal exploitation area is achieved (minimum level on the coal system is 10.50 masl). Gradient of working slopes system is generally mild, amounting to about 4°. The disposed material – soft soil (lessoid and coaly clay, gravel, sand, alevrolite, and related varieties) is filled to thickness of up to 90.00 meters. The lenght of contour of the system of the northern finishing slopes is about 2.00 km. The southern, advancing front of working slopes is inclined from 142.50 to 145.00 line, and is up to 3.00 km long. There are plana of interlaminar sediments developing to the east-west direction, which strenghten the landfill’s footing and extrude the surface mud from the pre-footing area (figure 5). The entire site overlays the watered, medium-grained, quartz – micaceous sand, water from which is firstly drained into collection basins of SM ”Tamnava-West Field” and then re-pumped into collection basins of inactive SM ”Tamnava-East Field”. The re-pumping process ends into a flowing lake – an extended part of the artificial flow of the Kolubara river. The figure 1 is a layout of the inner landfill, flowing lake and the Kolubara, inclusive of surveying benchmarks and area covered by sludge- watered deposits.

2. METHODS USED FOR CONTROL OVER INSTABILITIES ON WORKING SLOPES DURING PRE-FLOODING PERIOD

In the pre-flooding period, there was no technically acceptable method for providing realistic prognosis on design slope stability of II BTO (alevrolite and related varieties) and I BTO systems (made of the Quaternary sediments). The unique solution that the author of this paper considered for control over the sudden landsliding toward lower parts of the open-pit mine was formation of sandy counterforts in front of the then lable working slopes of II BTO system. This solution was based on the fact that the quartz – micaceous sand was chemically viable soil, not prone to degradation under abrupt temperature fluctuations. In addition to this, their grain-size distribution allowed the surface water drain easily from pores and voids of fresh disposed, impermeable masses of working slopes (figure 2). Risk controlling by the proposed method was also acceptable in terms of financial validity, for sand was one of the lithologic members of the exploitation field.

This implementation of the method started in 2013, and the most important testing was carried out during the pumping of the mine (2014), when the landslide in lenght 65.00 m and width 1170.00 m was activated. On that occassion, the sandy cones managed to mitigate the blows and save the submersed machinery from the mechanical damage. The position of the diffused cones and moved ground is documented in the two underwater imageries of ”Plov puta”. The photo of the landslide area (28 November, 2014) is provided in the figure 3.

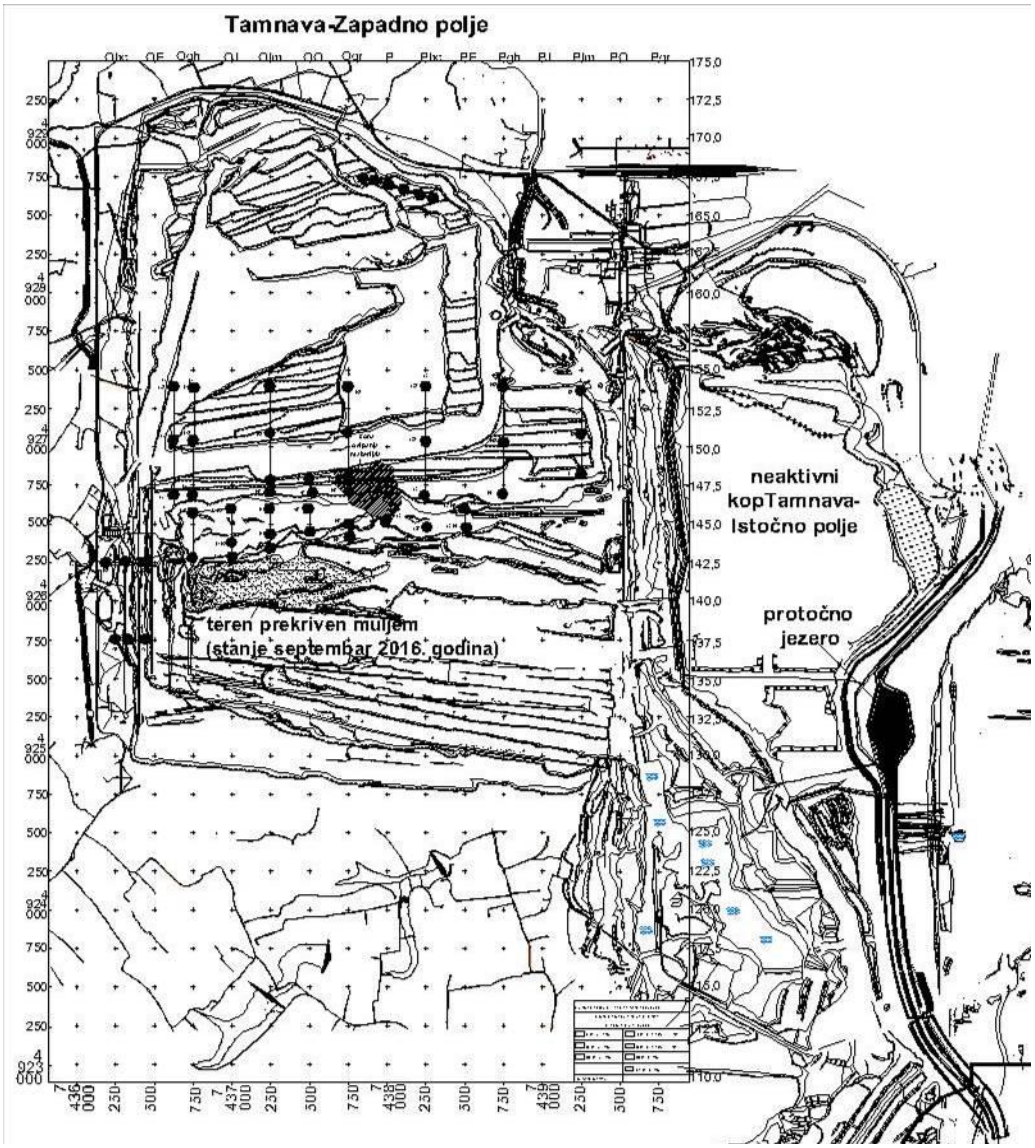


Figure 1. Skecth of SM ”Tamnava-West Field”, with benchmarks observed, and area covered by sludge and water remained after the pumping of working figure



Figure 2. Protective sandy objects,, in front of the footing of the inner landfill of SM”TWF”, pre-flooding period



Figure 3. Activated landslide of watered, poorly consolidated masses of II BTO system, during decrease of level in the flooding lake for 20.00 m-28.11.2014

3. ENGINEERING CHALLENGES IN FORMING THE LANDFILL, DURING AND AFTER THE PUMPING OF THE MINE

In course and after the pumping specific circumstances were generated in the inner landslide. The following issues should have been resolved:

- overtopping the ground beyond the design maximum levels (+110 masl) in area of the north, finishing slope, and securing the continuos pumping of the mine by possible rescue of submersed machinery,
- rehabilitating the pre-footing watered and sludged area of the landfill, in order to secure further disposal (figure 4. and 4a),
- securing the slope stability along the corridor in order to move the machinery to the next mine ”Radljevo”, and
- aligning the landfilling technology with the requirement for increased manufacture of lignite to 2 000 000 t/annum.

The said assignments required the progressive and cautious analyzing of all available data, given the solutions had to be complied with the rulebooks on exploitation of mineral raws and law on mining anf geology.

4. METHODS, INVESTIGATION RESULTS AND CONCLUSIONS ON GEOTECHNICAL CONDITIONS FOR CONSTRUCTION OF LANDFILL, DURING AND AFTER PUMPING OF THE MINE

4.1 Methods and results of investigation performed in the inner landslide during and after pumping of the mine

Doubts about the general stability of the inner landslide were intensified during and after the mine pumping. In order to ensure the real assessment of possible rescue of submersed machinery and continuous pumping of the mine, the geotechnical investigations were designed. The investigation results (two-staged investigation) were a basis for creating the field model and deploying geostatic calculations. The tables 1 and 2 provide calculation parameters for the recognized geotechnical zones. Geotechnical investigations were based on the strength parameters of alevrolite which was a predominant soil in the landslide composition. The figure 4 shows nonconsolidated blocks of alevrolite, grain-size 300 – 1000 mm. All mathematical analysis of slope stability were verified by geodetic tracking of 33 benchmarks within seventeen cycles (from June 2014 to May 2015) as well as by underwater recording. During the pumping of the mine, no response on groundwater from piezometers was received, while during the hydrogeological monitoring it was impossible to define a unique aquifier and conclude whether the slope stability was affected by action of filtration forces. Pumping tests showed the soil was poorly permeable to impermeable, and that, by simplification principle for such type of mining activity, it could have been considered as an earth fill relying upon permeable sand. The sandy subsoil of the landfill is seismic resistant and with good bearing capacity, with thickness exceeding 100.00 m.

Table 1. Engineering geological properties of the field for area - 1 poorly consolidated soil

Geotechnical environment	Density of naturally humid soil : ϒ (kN/m^3)	Density of absolutely arid soil: ϒd (kN/m^3)	cohesion: c_u (kN/m^2)	Angle of internal friction: $\varphi_u(^0)$	Compressibility modulus: σ=100-200 (kN/m^2)
Disposed material 1	18.90	14.80	50	0	1550-7600

Note: *The presented strength parameters are applicable to non-drained conditions for disposed alevrolite, depth from 8.00 to 10.00 meters.*

Table 2. Engineering geological properties of the field for area – 2 highly consolidated soil

Geotechnical environment	Density of naturally humid soil : ϒ (kN/m^3)	Density of absolutely arid soil: ϒd (kN/m^3)	cohesion: c_u (kN/m^2)	Angle of internal friction: $\varphi_u(^0)$	Compressibility modulus: σ=100-200 (kN/m^2)
Disposed material 2	20.80	17.78	32	39.40	1450-6100
Disposed material 3	It has properties of bedrock when positioned at depth exceeding 35.00 meters.				

Note: *The presented strength parameters are applicable to drained conditions for disposed alevrolite, depth from 10.00 to 35.00 meters. The given parameters can also be adopted by calculating the limit bearing capacity of the planum on which the planned overtops would be built.*



Figure 4. A detail from the slope of nonconsolidated blocks of alevrolite, grain size 300-1000 mm (II BTO system)

4.2 Conclusions on geotechnical conditions for forming the landfill during and after pumping

The reahabilitation of the flooded mine and execution of geotechnical works was performed in regular consultation with the Jaroslav Černi Institute for the Development of Water Resources from Belgrade. The related results were documented in seven reports prepared by members of the team in charge of monitoring the pumping process.

It was concluded that the slope security of II BTO system depended on adhesion of earth blocks due to asperity of great contact area, fill hight and subsoil inclination. The joining of cones resulted in formation of a working slope made of chaotically filled blocks, grain size ranging between 300 mm and 1000 mm. Filling the gaps between the cones with material to

designed levels in the hinterlands of the working slopes resulted in plana in lenght over 100.00 m. The positive impact of so chaotically distributed blocks was the plunging and natural draining of surface water within the slopes of II BTO system. However, behaviour of the ground when the earth blocks were highly leached by surface water and voids were filled with water, what was the case during the flood and long-lasting and heavy rainfall, was slightly different from behaviour of the monolithes of those sediments in the mine. Simulating of similar conditions was done through triaxial tests in undrained, non-consolidated conditions onto samples extracted from 5.00 m depth, from boreholes positioned nearby the flooding lake. The method resulted in strenght parameters shown in the table 1. For depths greater than 10.00 m it was assumed that the earth blocks were crushed under the weight of an overburden of recently filled masses, and that better settlement of materials was reached. This theory was confirmed by standard penetration testing. The properties of this environment and measuring methods are presented in the table 2. The uneven values of compressibilty modulus ($Ms_{(100\text{-}200\ (kN/m2)}$=1450-6100 kN/m^2) indicated that the soil was not consolidated at the moment, and that roaming water might occur through voids of loose material. The author of geostatic calculations for slope stability decided not to model the roaming water as ground water, for they were not part of a unique aquifier (verified by hydrogeological monitoring) nor could pore pressures develop in them due to diffused motion and occassional oocurrence (during rainfall). After the pumping of the mine was finished, the described geotechnical model was used for preparation of monthly operative plans, and calculated results were verified in seven cycles of geodetic measurements of 53 benchmarks. It should be noted that deformations on the working slopes of II BTO system of the landslide occured once, when water in the flooding lake dropped for 20 meters. The instability exested even before the flood, when the ground was affected by landslides due to rapid landfilling in thickness greater than 50 m across the slanted base (figures 3, 8 and 9). The method of risk control against sudden sliding, before the flood and the works, is described in the chapter 2 (figure 2).

4.3 Overtopping the landfill beyond earlier design maximum levels

The author of this paper analized the space after the pumping (the space was extending from the muddy area to the finishing slopes on north (2.00 km long)) and obtained positive results in terms of capacity of the field for safe execution of intervention works inclusive of landfilling onto the tight area. This portion of the landfill was formed of alevrolite that had been depositing 15 years before the flood. Stability was checked by geostatic calculations by Spencer method, including the strenght parameters under tables 1 and 2. As the analysis showed no existence of sliding plane through a base that was 2.00 km away from the flooded lake, standard penetration testing and direct determination of capacity were therefore cancelled. That decision was also supported by the fact that fresh alevrolite on the bank of the flooded lake had already been tested for standard penetration and that better results in terms of capacity might have been expected in the area outside impact of the lake. Field overtopping above the design maximum levels was regularly followed by visual inspection. Observations on soil behaviour were compared with the instrumental surveying in grid of 33 benchmarks. The analysis excluded the impacts of the highest onto the lowest, saturated parts of the field and forming of sliding body in lenght of about 2.00 km. However, due to expressed suspicion of colleagues into the validity of all previous works and conclusions on geotechnical conditions in the landfill, it was decided to pay more attention to the given assignment. The author of this paper tried to mitigate the possibility that certain instability factor was overlooked in the analysis by proposing that tailings should be filled in steps of 10.00 m high

in wide fronts. This technology was expected to ensure more even consolidation of loose material in any weather circumastances. The first overtop above the design maximum levels was carried out in February 2015 to level +120 masl. The figure 6 shows the state of affairs in the site for level of +142.00 masl, in June 2017.

5. EXPLORING POSSIBILITIES FOR DISPOSPING SOFT SOIL INTO MUDDY AREA

Once the pumping of the mine was completed, the watered ground with properties of a marsh, in an area of around 22 ha, remained to overlay the coal and sand in the pre-footing area of the working slopes of the landfill (figures 5, 5a and 5b). This geodynamic phenomenon is a low capacity base for stable fills of tailings soil. The impact of lable soils onto the stability of higher systems of the landslide and possibilities of their co-action on security of coal systems were explored. It was also required to determine the degree of risk from sliding down of disposed, earth masses from higher zones for needs of moving the machinery into the mine ”Radljevo”.

5.1 Proposal for rehabilitation of the muddy field, and implementation

A rough solution to rehabilitation of the watered part of the field was presented by the author of this paper in June 2016, via means of electronic correspondence, to all members of the expert team of MB ”Kolubara”. The proposed sandy rehabilitation of the lable field was documented in the ***Report on sludge (deposit) at SM ”TWF”*** no. 04.07-363408/1-16 of 12.09.2016. The rehabilitation measure included the extrusion of the lable, fluid materials by interlaminar sand to the deepest parts of the field. Sand-made plana in the east-west direction would be formed by this technology. The filling would last until the contact with a solid base – coal or flooring sand is achieved, with simultaneous extruding of lable soil into the space from which water could be injected into the surrounding granular ground or removed by pumps. At the same time, the sandy plana would serve as counterforts against higher disposed masses and drains for remaining water masses. The figures 5, 5a and 5b confirm the started implementation of the solution.



Figure 5. Sandy masses extruding the mud to deepest,, western part of land (September 1, 2016*)*



Figure 5a. Marshy part of the mine bordered by sandy planum to south (September 2, 2016)



Figure 5b. Marshy part of the mine confined by sandy masses to the western, deepest space (June, 2017)

6. COMPLETION OF PROPOSED GEOTECHNICAL SOLUTIONS FOR RISK CONTROL IN THE INNER LANDFILL

Due to thightness of the disposal area, the overtops were performed just during the pumping of the mine. Currently, they are 32.00 meters above the design maximum heights. Slopes were not demolished in this area, nor the impact of increased strains reflected to the lower, before-footing and watered slopes. Land slides or failures were not projected in the calculations, nor were observed during the new geodetic monitoring of seven cycles on 53 points (the measurements took place from September to March 2017). The state of affairs in the performed working slopes of I BTO system (level 142 masl) and wide planum (disposed clay and terraced gravel) in the area of landfill where the earlier design maximum level was exceeded are shown in the figure 6.



Figure 6. Planum and slope at the I BTO system of inner landfill OP "TWF" with no deformation and landslide - 25.06.2017

The instabilities occured in lower parts of the landfill, just after the monitoring stopped in April 2017, when an earth block happened to slide down the working slopes of II BTO system in lenght of 450 m and width to 380 m (figures 1, 8 and 9). This caused no damages to the machinery nor delays in landfilling works. The figure 1 shows the sliding body contoured between the profile lines 150 and 145 (north-south), OO and P (west – east). The influcence of the ground in motion onto the machinery operating at lower levels was prevented by plana of sand and interlaminar tailings- the materials which also extruded the mud and water toward deeper, western part of the mine, as in accordance with the proposal presented in the Report on sludge (figure 5). The analysis showed that the essential factor for instability was the landfilling onto the subsoil with morphological anomalies in form of amphitheatre recessions, within which poorly consolidated masses were repacking due to gravity action. The secondary factor of instability was dewatering of soil due to abrupt rise of temperature and reduction of apparent cohesion. The sliding or dispersing of materials are expected to take place in the ground until local depressions become filled. The risk of landslide actions against the coal systems is reduced by designing slopes with general gradient of 7° and plana, at the level of +66,00 masl in width of 165 m. The deformed flooring of second coal complex is illustrated in the figure 9. The block diagram shows ground in motion bordered with lines Oqr-P-Pbc (west – east) and 152-135 (north – south).

7. CONLCUSION

Assessing the same issue in mupltiple manners is a consequence of adjustments in number of variable data and understanding of conditions in the site that is a dynamic-static system, in area of 8.50 km^2. The investigations performed in 2014 and 2015 enabled new approaches in defining the instability factors in the area of inner landfill SM ”TWF”.

The main objective of this paper is to minimize the bias impact when assessing the instabilities, with adjusting the all human values to nature-ruled principles. The accurate interpretation of instability factors ensures the proper risk control in landfills.

ACKNOWLEDGMENT

I wish to thank my colleagues in SM ”TWF” for suggesting me to improve the interpretation of available data, to devise geotechnical exploratory works and provide proposals for rehabilitation of specific issues.



Figure 8. Landslide on II BTO system of inner landfill, 400 m wide (April 5, 2017). Contours of the sliding body follow the shape of deformed plane of slope of II coal complex

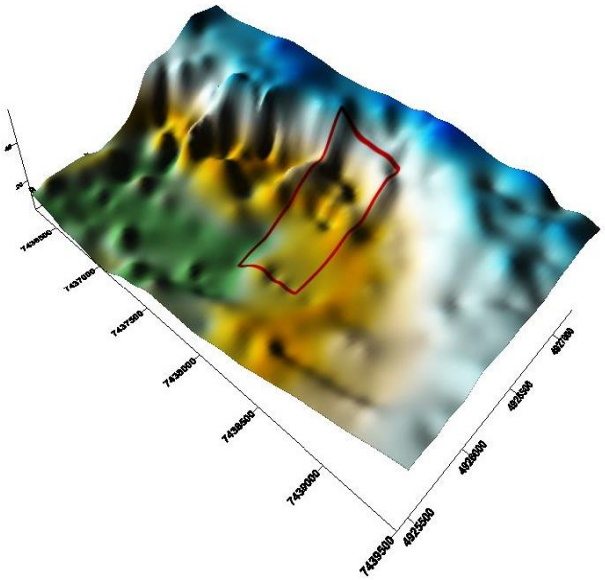


Figure 9. Block diagram with anomalies within the flooring of II coal complex, disposed material is sliding and dispersing along them - (Z.Kitanović, geological engineer)

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DOI: 10.7251/BMC170701307J

SPECIFIC FEATURES OF SAND EXPLOITATION TECHNOLOGY IN DEPOSIT JAKOVAČKA KUMŠA WITH HIGH GROUNDWATER

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ABSTRACT

Topic of this article is technological solution of construction sand exploitation in existing excavation zone and in extension of the exploitation field in sand deposit Jakovačka kumša in coastal environment of river Sava, aquifers in sediments supplied from the river and the system of melioration channels which maintain the level of ground water.

Key words: sand, exploitation, hydraulic and float dredger

1. INTRODUCTION

Territory of the sand deposit Jakovačka kumša belongs to local municipality Surčin in Belgrade, 30 km far away from the city centre.

In morphological sense the narrow area of the sand deposit represents the alluvial plane of the river Sava with altitudes of +72- +77 m, in average +74 m. Building of defensive embankment of river Sava and network of the melioration channels it was prevented flood from the river bed of Sava and ground water level was regulated maintained at the level of +71 m.

Geological material on the narrow territory of deposit is represented with Quaternary sediments: clay, yellow fine-grained sands, gray sands, gravelly and sandy clays.

Productive series of Jakovačka kumša deposit is made of yellow fine-grains and gray sands. The top layers of the productive series are made of sandy clays and humus and in the bottom is sandy-gravely series.

Based on geological material typology of some sediment porosity and hydrodynamic conditions in the frame of Jakovačka kumša deposit, it is possible to select 3 types of aquifers:

1) Compacted type of higher abundance within (bottom) sandy-gravely sediments with altitudes of +58.55 - +37.24 m.
2) Compacted type of lower abundance within gray sands (lower layer of productive series) which are above sandy-gravely sediments with altitudes of +62.70 - +54.74 m.

3) Conditionally anhydrous sections of terrain connected with yellow fine-grained sands (upper layer of productive series) with altitudes +68.22 - + 57.30 m and the overburden of productive series, clayey sandy humus.

   Supply of aquifers of type 1) and 2) isprovided from the river Sava, from old fossil bed and by infiltration of rainfalls and drainage during low water level toward the river and artificially via tubular and artesian wells.

In the former period of 5 years in the area of Jakovačka kumša deposit, the public company (JP) Surčin exploited sand by discontinued system, hydraulic dredge-trucks. Excavation of send at deeper levels (up to 66 m) resulted in excavated zones backfilled by water. On such these surfaces, in extension of the sand exploitation, will be exclusively executed underwater exploitation by the system float dredger - pipeline - precipitators.

JP Surčin is presently in the procedure of widening the exploitation field the purpose of which the Mining Institute doo Beograd made the Main mining project on construction sand exploitation in Jakovačka kumša deposit.

The project defines two periods of exploitation:

1) Up to 10th year of exploitation with system of underwater exploitation implementation, and
2) From 11th until the end of exploitation (48thyear)with combination of classical discontinued system and underwater exploitation system.

2. PRESENT SITUATION OF WORKS ON SAND EXPLOITATION AND CONTINUATION OF EXPLITAITON UP TO 10TH YEAR

The figure 1 illustrates a satellite record where is visible present situation on the surface mine Jakovačka kumša. The sand is excavated by the floated dredger- suction (1), hydraulically transported together with water by float and ground pipeline (2), toward prepared cassettes or precipitators, (3) and (4).

The same figure illustrates water surface (5) formed in the zone of exploitation in the frame of approved exploitation field. On the eastern side, in the frame of approved exploitation field is so called inner landfill (6), actually utility landfill which has narrowed exploitation area. Also on the eastern side of the exploitation zone, is municipal asphalt road (7) that goes to Jakovo on the north side of the surface mine, toward river Sava embankment on the south side of the surface mine.



Figure 1: Present condition of work on the surface mine Jakovačka kumša

Due to already excavated top layers, excavated zones backfilled with water and lack of the condition for excavation by hydraulic dredgers, continuation of excavations up to 10th year of exploitation will be performed by underwater procedures.

The sand, excavated and sucked in by float suction dredger will be transported together with water by dredger's float and ground pipelines with diameter of (Ø) 300 mm to precipitators (figures 1 and 2). Maximum length of the float pipeline in next 10 years will be of 400 m. hydraulic transported sand, together with water, will be placed in precipitators to sieve and dry, and then it will be loaded in trucks by hydraulic dredger, directly or indirectly. The figure 1 illustrates 2 precipitators which are filled and emptied in alternating mode, (3) and (4).



Figure 2: Underwater exploitation system on the surface mine
Jakovačka kumša, float suction dredger – pipeline – precipitator

Excavations of sand will be performed in the existing exploitation zones up to designed excavation plane with altitudes +60 - + 63 m, and on depth of 8-11 m in relation of the water level, by work of the floatsuction dredger in 3 sub-floors with altitudes +66, +63 and +60 m.

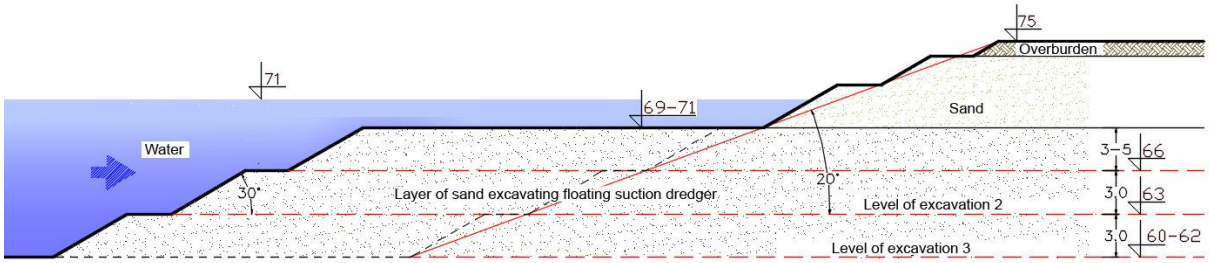


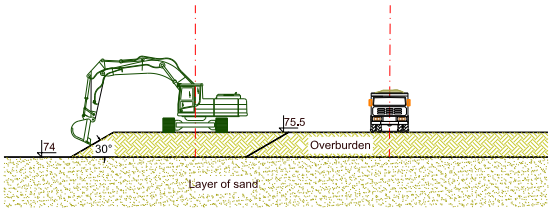
Figure 3: Technological profile of surface mine
Jakovačka kumša, I phase to 10th year of exploitation

3. SAND EXPLOITATION AND OVERBURDEN IN EXTENDED SECTION OF THE SURFACE MINE AND EXPLITATION FIELD

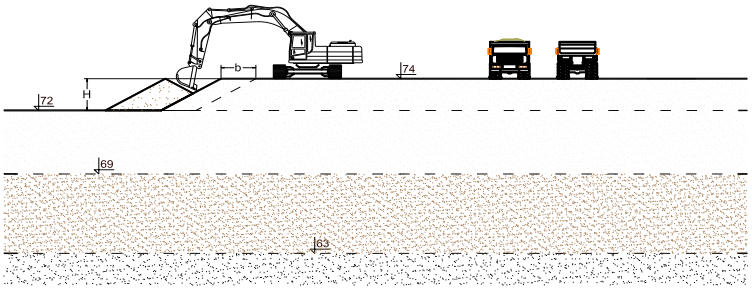
Exploitation of construction sand and overburden on the Jakovačka kumša deposit, in the extension of surface mine, from slopes of present excavation zones (Kumša 1 and Kumša 2) to final slopes in the contour of new extended exploitation field designed as combination of discontinued surface exploitation and continued underwater exploitation.

Sediments of Jakovačka kumša deposit inside of designed surface mine will be excavated in 3 floors.

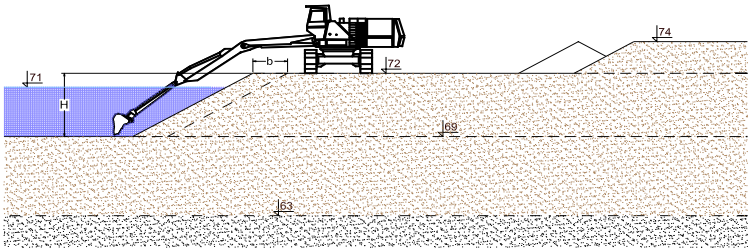
1)



2a)



2б)



3)

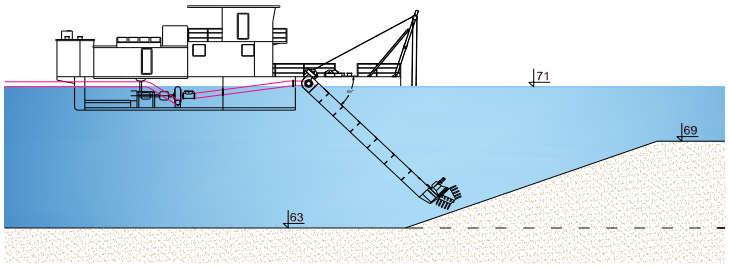


Figure 4: Excavations in contour surface mine Jakovačka kumša

On the floor 1 will be excavated materials from overburden, humus and clayey 0-3.1 m, by hydraulic dredger with direct loading in trucks.

On the floor 2 will be excavated sand in 2 sub-floors with two different hydraulic dredgers:

- Upper sub-floor (2a) excavates hydraulic dredger which excavates overburden; the dredger is placed on the surface of the layer-sand, it excavates 1st sub-floor to altitude +72 m, and at the same time takes the sand placed by the dredger work on the lower sub-floor and loads it directly to trucks;
- Lower sub-floor (2b) excavates hydraulic dredger with longer mast which is placed on altitude +72 m, excavates sand partially below the water level to altitude +69 m and temporarily place it on the standing level with working slope of higher sub-floor and in zone of 1st dredger.

On the 3rd floor sand will be excavated by float suction dredger from altitude +69 m to +63 m in two sub-floors of 3 m high and transported through pipeline to precipitators.

Sand excavated in 3rd floor by float sucking in dredger (Figure 4-3) will be transported together with water by the pipeline with diameter of 300 mm to precipitators where it will drained and then loaded in trucks and transported to consumers.

For continual underwater exploitation it is necessary to predict, in each period of exploitation, two precipitators (or cassettes) for sucked and hidraulic transported sand. Simultaneously, in one cassette will be performed hidraulic filling and in other loading of dried sand in trucks.

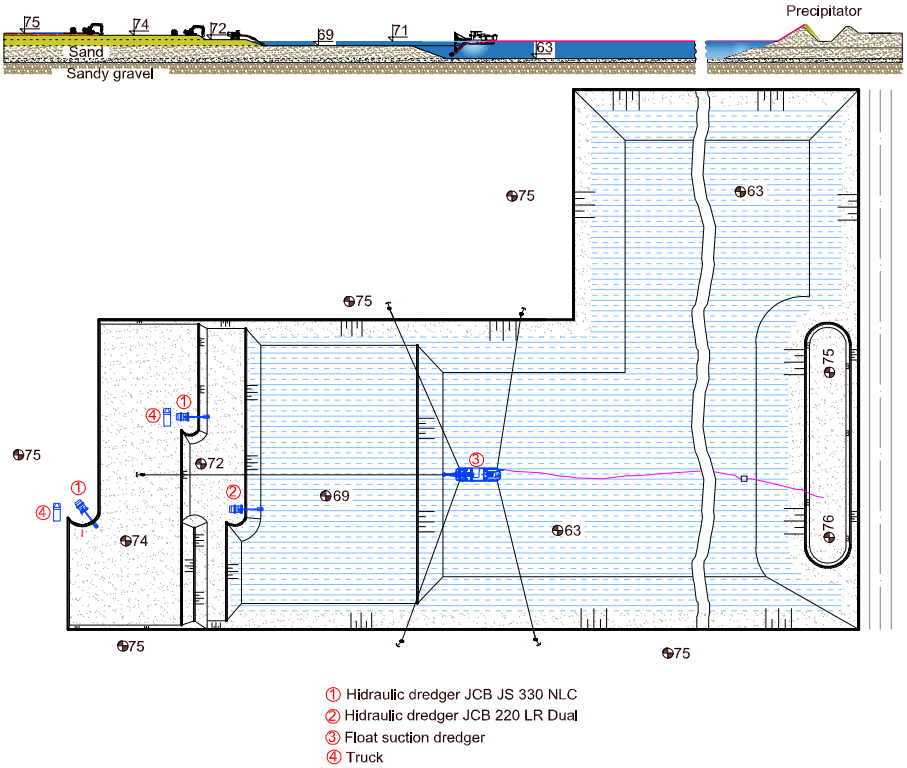


Figure 5: Consistent technological scheme of sand excavations and overburden in extension of surface mine Jakovačka kumša.

Figure 5 illustrates consistent technological scheme of excavations of overburden and construction sand on surface mine Jakovačka kumša, in plan and cross-section, in full development of the surface mine, in the period from 11th to final 48th year of exploitation.

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DOI: 10.7251/BMC170701313K

STABILITY MONITORING OF THE EXISTING COLLECTOR UNDER THE FLOTATION TAILINGS DUMP VELIKI KRIVELJ (SERBIA)

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ABSTRACT

Flotation tailings dumps, in terms of environmental protection, represent a real risk on humans and the environment, regardless of whether they are in operating mode, or the process of disposal has been completed. In eastern Serbia, on the territory of Bor Municipality, there are two active flotation tailings dumps: Flotation tailings dump RTH and Flotation tailings dump Veliki Krivelj. Risk of environmental hazard can be high in one or more projected structures within the tailings dumps.

The aim of this paper is to answer the question whether the collector of the flotation tailings dump Velki Krivelj resistant to accident situations. Potential accidental situation can lead to extreme pollution of Borska reka, subsequently Timok and Kriveljska reka, and finaly Danube. Visual observation of entrance and exit portal of collector, show that the concrete structure is stable and safe in terms of functionality. Noticeably higher damages exist on spherical sections of metal coatings, and breakthrough of contaminated drainage water is present at 70th and 80th meter of the collector (chainage 0 + 200 m). The level of water is significantly increased at the exit portal of collector as well as blurring of Kriveljska Reka is observed.

Key words: monitoring of stability, existing collector under flotation tailings dump Veliki Krivelj, water of Borska Reka, Danube

1. INTRODUCTION

Copper-sulphuric-pyrite ores are formed in veins, smaller ore bodies of irregular shape with stockwork-impregnation type of mineralization. The area around the massive-sulphide mineralization is filled with the stockwork-impregnation and wire type of mineralization [4]. The common minerals found are quartz, pyrite, chalcopyrite, galenite, enargite, and gold. These ores also contain a number of minor elements such as rhenium, cadmium, bismuth, gallium, thallium, germanium, indium, selenium, tellurium, gold, silver, ytterbium, and

yttrium. Considering the chemical compositions of flotation tailings, continual monitoring of dump structures is necessery.

High risk of environmental hazard is noticed for one or more projected structures of flotation tailings dump Veliki Krivelj. Oxidation of sulfate minerals leads to release of many toxic metals from the flotation tailings into solution (Fe, Cu, Cd, As, Mo, and others). The resulting solutions (acid mine drainage - AMD) is characterized by low pH (typically less than 5) and a high concentration of sulfate ions and metals [2]. Potential Health/Environmental Effects from base metals found in mine tailings and flotation tailings Veliki Krivelj are: (1) Copper (Cu) - Small amounts are considered non-toxic and necessary for human metabolism. However, large doses may induce vomiting or liver damage. Toxic to fish and aquatic life at low levels; (2) Cadmium (Cd) - Cadmium is concentrated in tissue and humans can be poisoned by contaminated food, especially fish. Cd may be linked to renal arterial hypertension and can cause violent nausea. Cd accumulates in liver and kidney tissue. It depresses growth of some crops and is accumulated in plant tissue; (3) Arsenic (As) - Highly poisonous and possibly carcinogenic in humans. Arsenic poisoning can range from chronic to severe and may be cumulative and lethal; (4) Chromium (Cr) - Cr^{6+} is toxic to humans and can induce skin sensitisations. Human tolerance of Cr^{3+} has not been determined; (5) Mercury (Hg) - Hg and its compounds are highly toxic, especially to the developing nervous system. The toxicity to humans and other organisms depends on the chemical form, the amount, the pathways of exposure and the vulnerability of the persons exposed; (6) Iron (Fe) - Essentially non-toxic but causes taste problems in water and (7) Zinc (Zn) - May affect water taste at high levels. Toxic to some plants and fish.

Both surface water and groundwater have the potential to be impacted by tailings disposal sites, either through surface water runoff to watercourses or leaching to groundwater. The nature and significance of such impacts will depend upon the characteristics of the waste material and the environmental setting. During the operational life of the dam the discharge of the effluent containing tailings will have been a key source of water based contamination.

2. EXPERIMENTAL

Monitoring the state of stability of collector under the Flotation tailings dump in second half of the 2014, was carried out on the route of 0-200 metres. The picture shows the end chainage 0 + 200 m and the start of the repaired collector (Figure 1) work, that is narrowed in diameter of 1m. When you go inside that part of the collector, further continual monitoring, visit and review of panels is disabled due to the high pressure of water.


a)

b)

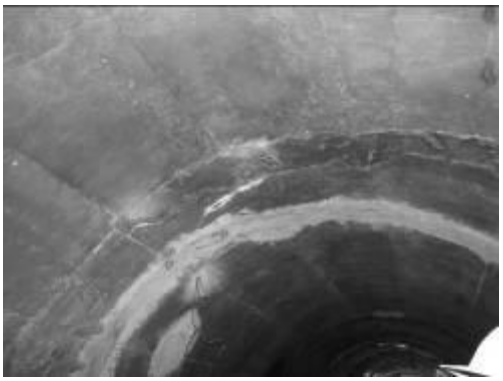
Figure 1**.** a) The start of the repaired collector; b) Exit portal of collector.

Visual observation of concrete exit portal of collector (Figure 2 a, b), indicates that the concrete structure is stable and safe in terms of functionality [3]. Noticeably higher damages exist on spherical sections of metal coatings (Figure 3a, b), and breakthrough of contaminated drainage water is present at 70th and 80th meter of the collector (chainage 0 + 200 m) (Figure 4 a, b). The level of water is significantly increased at the exit portal of collector (Figure 5 a, b) as well as blurring of Kriveljska Reka is observed.


a)

b)

Figure 2. Visual observation of collectors.


a)


b)

Figure 3. Collectors from 4th to 6th meter.



a)


b)

Figure 4. Collectors from 70th to 80th meter.

3. RESULTS AND DISCUSSION

The negative impact [1] on the environment caused by AMD is reflected in the long-term contamination of soils which come into contact with mine waters and accumulation of heavy metal ions in it; in mixing AMD with surface water; AMD mixing with the underground waters and their pollution, and so on. AMD represent one of the most serious threats to the aquatic world and the environment. Water from the dam "3A" of flotation tailings dump

Veliki Krivelj contents pollutant Cu^{2+} ions 53 (mg /dm^3) and flows to the Kriveljska reka (Figure 5a, b). The water quality [4] of the Kriveljska reka is given for the period 2006-2017:

- C_u min. 0.376 (mg/dm^3) in 2006. max. 168.140 (mg/dm^3) in 2010;
- Fe min. 0.200 (mg/dm^3) in 2007. max. 455.40 (mg/dm^3) in 2010;
- Cd min. 0.000 (mg/dm^3) in 2006, 2007 and 2008. max. 0.035 (mg/dm^3) in 2011;
- Zn min. 0.170 (mg /dm^3) in 2007. max. 13.040 (mg /dm^3) in 2010;
- Ni min. 0.000 (mg /dm^3) in 2006 and 2008. max. 0.250 (mg /dm^3) in 2010;
- pH min. 5.62 in 2009. max. 7.22 in 2012;
- Suspended solids in 105^0 C min. 172.40 (mg /dm^3) in 2009. max. 460.90 (mg /dm^3) in 2012.


a)

b)

Figure 5. Exit portal of collector (high level of water, as well as blurring of Kriveljska reka).

4. CONCLUSION

The high degree of water pollution of Kriveljska reka at the exit portal of collector is established by visual observation. Breakthrough of contaminated drainage water (AMD) is noticed at 70th and 80th meter (chainage 0 +200 m). Is permanent monitoring of the collector of Kriveljska reka necessary in order to prevent environmental disasters? The answer is YES. Visual observation of concrete exit portal of collector show that the concrete structure is stable and safe in terms of functionality. Increasing number of damages on the cladding sheets of collector's calotte requires permanent monitoring of the collector of Kriveljska reka.

5. ACKNOWLEDGMENTS

This investigation was conducted under Project TR 330021 "Research and Monitoring the Changes of the Stress Strain State in the Rock Mass "In-Situ" around the Underground Rooms with Development of Models with Special Reference to the Tunnel of the Krivelj River and Pit Bor", funded by the Ministry of Education, Science and Technological Development of the Republic of Serbia.

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DOI: 10.7251/BMC170701317S

THE STATE AND THE DEVELOPMENT PERSPECTIVE OF POWER UTILITY OF THE REPUBLIC OF SRPSKA

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ABSTRACT

Despite the facts and information about the continuous decrease in energy potentials year after year, the power utility still represents the foundation of economic growth and development in every country. Accordingly, development of electric power industry is the foundation of modern ways of living, as well as the key factor for economic changes. Development of the energy sector makes technological development, economic growth of the company as an institution, increase of the investments and stronger competitiveness possible. The energy industry is one of the most intense branches of the economy when it comes to the investments, and the power utility is the same within the energy industry.

This paper will give a brief outline of the current state of the Power Utility of the Republic of Srpska (orig.: *ElektroprivredaRepublikeSrpske – ERS*), as well as some future forecast for making the competitiveness in the open market stronger.

Key words: Energetics, Power Utility, competitiveness, investments

1. INTRODUCTION

The Power Utility of the Republic of Srpska is a strategic company performing the common interest business. It is organized in the form of a joint-stock company (JSC). The parent company is 100% owned by theRepublic of Srpska and holds 65% of the subsidiary companies' capital stock.The work of Power Utility of the Republic of Srpska is based on securing enough electricity for the purpose of secure and quality supply to the consumers in the Republic of Srpska.

Apart from continuous and secure energy supply, the main priorities of the Power Utility are aimed towards providing quality service while having environment and well-being of the society in mind.

Power Utility of the Republic of Srpska chooses to perform activities related to responsible investment with the main goal to achieve the security of supply, sustainability and competitiveness, based on the parts of the strategy which needs to ensure production growth, electricity and heat supply security, mineral resources (coal) distribution increase and implementation of EU directives' obligations.

2. ORGANIZATION

Power Utility of the Republic of Srpska consists of 11 subsidiary companies (SC) within one Parent Company. The companies within the group (Holding) have the status of separate legal persons, each managing its own books of accounts, accounting its corporate income taxes and each submitting its annualtax declaration for determining the income taxes.

The business policy is determined by the Republic of Srpska Governmentthrough Shareholders Assembly, and it is implemented by the Supervisory Board, Revision Committee and Management Board. The Management Board of MH "ERS" is in Trebinje and it works by its own administration.

Mixed Holding "Power Utility of the Republic of Srpska" (hereinafter: MH "ERS") consists of 5 electric power production companies which are:

1. Subsidiary Joint Stock Company "Hydro Power Plant on the Trebišnjica River", Trebinje
2. Subsidiary Joint Stock Company "Hydro Power Plant on the Drina River", Visegrad
3. Subsidiary Joint Stock Company "Hydro Power Plant on the Vrbas River", Mrkonjić Grad
4. Subsidiary Joint Stock Company ""Mine and Thermal Power Plant Gacko", Gacko
5. Subsidiary Joint Stock Company "Mine and Thermal Power Plant Ugljevik", Ugljevik

MH ERS also consists of 5 electric power distribution companies, which are:

1. Subsidiary Joint Stock Company "Elektrokrajina", Banja Luka
2. Subsidiary Joint Stock Company "Elektro Doboj", Doboj
3. Subsidiary Joint Stock Company "Elektrodistribucija Pale", Pale
4. Subsidiary Joint Stock Company "Elektrohercegovina", Trebinje

Subsidiary Company "Electronics Research and Development Centre (orig.: ZP IRCE – *Istraživačkorazvojnicentarelektroenergetike*) in East Sarajevo is the eleventh company. Also, two private limited companies (PLC) were founded: PLC "HPP Dabar" (founded by MH "ERS" " JSC ""HPP on the Trebišnjica River") and "Termo Nova" PLC (founded by JSC "Mine and Thermal Power Plant Ugljevik", Ugljevik).

3. CURRENT STATE AND DEVELOPMENT POTENTIAL

Analysis of the current state analysis and characteristics of energy sector in Republic of Srpska are the starting point for identifyingfuture energy needs, possibilities and options for meeting those needs, and defining targetsand measures for achieving goals of energy sector development inRepublic of Srpska.

Due to its natural characteristics, the Republic of Srpska belongs to the group of regionsrich in following:

- Hydropower potential
- Significant coal reservesfor creating bigger electric power generation capacities,
- Significant potential with the unused renewable energy resources and so on.

The total demand for electric power in the Republic of Srpska is satisfied by generating power in domestic power plants while the surpluses of electricity are exported to other markets.

The electric power production in the Republic of Srpska is based on thermal power plants that only use domestic coal and have about 60% share in total electric power generation, and on hydro power plants which generate about 40% of electric energy.

Total installed capacity of the generating plants within "ERS" is 1326,08 MW

The average annual electric power generation of the existing generating plants is approximately 5500 GWh. Generating capacities are mostly located in the southeast part of the Republic of Srpska, that is in the Herzegovina region, where "HPP on the Trebišnjica River", "HPP on the Drina River" and "Mine and Thermal Power Plant Gacko" are located. "Mine and Thermal Power Plant Ugljevik" is located in the Semberija region, and "HPP on the Vrbas River" is in the Krajina region.

The security of supply is very high, considering the fact that domesticsources of primary energy are used. At the moment, the power generation exceeds the consumption with the generated energy surplus of 20 – 30%.

Based on the existing energy capacities, installed and available capacities at power plants gates, performing regular main annual maintenance and 70% possibility of total water inflow at measuring points, uninterrupted electric power generation is ensured.

a. Electric power production

The following chart shows the difference between the planned and realised production in 2014 and 2015.

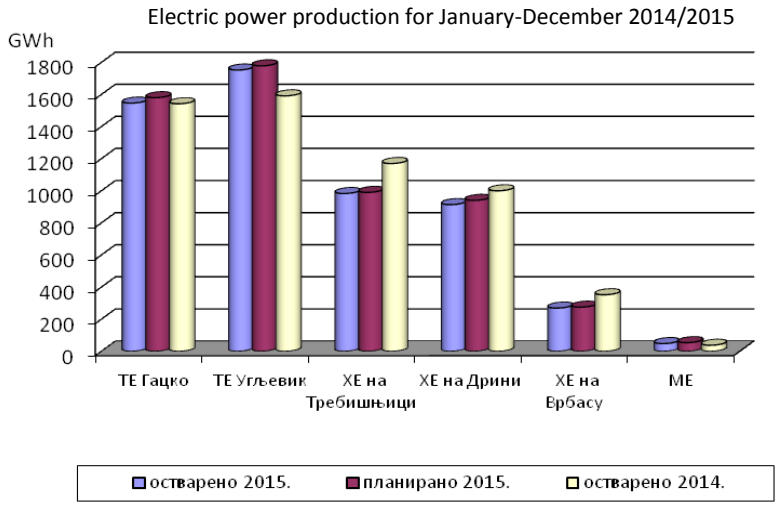


Figure 1: Realised and planned energy production in 2014 and 2015.

The following chart shows the percentage production values of subsidiary companies during 2015.

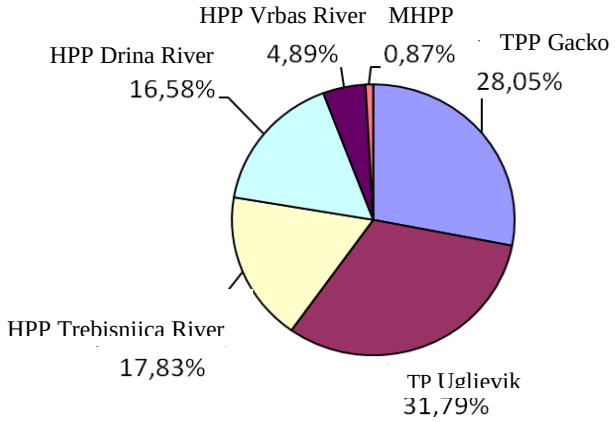


Figure 2: Production of electric power in subsidiary companies in 2015.

Out of 5.512,90 GWh of total amount of generated energy within MH "ERS", as shown in the charts, during 2015– SC "M&TPP Ugljevik" " JSC, Ugljevik generated 31,79%, SC "M&TPP Gacko" JSC, Gacko generated 28,05%, SC "HPP on the Trebišnjica River" JSC, Trebinje generated 17,83%, SC "HPP on the Drina River" JSC, Višegrad generated 16,58%, SC "HPP on the Vrbas River" JSC, Mrkonjić Grad generated 4,89% and MHPP (Mini Hydro Power Plants) generated 0,8%.

b. Installed capacities

The Power Utility of the Republic of Srpska consists of two thermal power plants, three hydro power plants and four "mini" hydro power plants, with the following installed capacities:

Table 1.

THERMAL POWER PLANTS		
No.	Name	Installed power (MW)
1.	M&TPP "GACKO"	300
2.	M&TPP "UGLJEVIK"	300
Total		**600**
HYDRO POWER PLANTS		
No.	Name	Installed power(MW)
1.	HPP on the Trebišnjica River	287
2.	HPP on the Drina River	315
3.	HPP on the Vrbas River	110
Total		**712**
MINI HYDRO POWER PLANTS		
1.	MHPP Bogatići	8
2.	MHPP Mesići	3,08
3.	MHPP Tišća	0,9
4.	MHPP Vlasenica	2,1
Total		**14,08**
TOTAL(HPP+SHPP)		**726,08**
THERMAL POWER PLANTS		**600 MW**
HYDRO POWER PLANTS		**712 MW**
MINI HYDRO POWER PLANTS		**14,08 MW**
TOTAL		**1.326,08 MW**

Total value of energy power consumption in 2015 was 3.957,55 GWh and it included 3.820,83 GWh of consumption by consumers supplied by Electric Power Distribution Companies (gross distributional consumption of 3.661,53 GWh + direct consumers' consumption at 110 kVof 159,30GWh), 13,96 GWh of own consumption by plants within MH "ERS"and transmission losses in value of 122,75 GWh.

Electric power distribution losses realised in 2015 were 427,60GWh, (11,68%) and were lower than the losses realised in 2014 (12,05%). Net energy consumption in RS (the Republic of Srpska) in 2015 was 3.393,24 GWh.

c. Electric Power Production Projection

The main part of RS, when it comes to electric power production development, is natural resources availability (above all, that of coal and hydropower), defined legal framework and existing infrastructure of the transmission network.

The system in RS is well connected to the neighbouring systems and there is a possibility of selling the electric power on the surrounding markets. The position of electric power will depend on demand and supply ratio, market prices and the abilities of regional transmission network.

Organizing and regulating the sector in a much better way will not change the circumstances under which the RS holds the comparative advantages for developing electric power production activities over neighbouring countries.

The projected operation lifetime of existing thermal power plants will expire sometime in the period 2020-2025. It is possible to refurbish them and extend the operation life time for additional 15 years or to gradually close them while simultaneously building new/replacement units at the same locations. All existing hydro power plants will be working by the end of the given period, with appropriate refurbishment of electro-mechanical facilities and construction buildings. For that purpose, new projects for use of hydropower potential of water streamsin the RS will be promoted and further developed. Locations of the objects considered as primary candidates for construction by 2030 are:

- HPP *BukBijela* and HPP *Foča* – expected to be constructed by 2020.
- *Upper Horizons* Project (orig.: Projekat *Gornji Horizonti*) (HPP *Dabar*, HPP *Nevesinje* and HPP *Bileća*) – completion expected during the period 2016-2020.

The preparation and HPP construction process is rather long (10-15 years). That is why, in addition to the already mentioned projects, the preparation of documentation and preparation of construction for the following hydro power plants will commence: HPP *Sutjeska*, HPP *Paunci*, HPP *Mrsovo*, HPP *Middle Drina* and HPP *Dubrovnik II*.

Some of the hydropower potential of water streams is shared with neighbouring countries/entity, thus the use of the said potential needs to be agreed upon with the interested parties. All raised question regarding past investments in the development of power plants will also be solved by agreement with interested parties, with full protection of the RS interests.

d. Electric Power Transmission

The network with voltage level of 400kV is satisfactorily built and connected in a loop across the territory of BIH, as well as suitably connected to all neighbouring systems (Serbia, Montenegro and Croatia).

The 220 kV network in RS has an important role in electric power transmission. Substations (SS) 220/110 kV Prijedor 2 and Trebinje make a significant contribution to the security of power supply of the wider areas of Prijedor and Trebinje. The 220 kV network is modelled in a satisfactory way in RS.

The 110 kV network is formed across the Republic of Srpska. The problem of radial power supply of certain SS 110/x kV needs to be solved.

For transmission lines with a smaller cross-section of conductors, cabling of some parts in urban areas is needed, as well as construction of double 110 kV overhead lines with conductors of a 220 mm^2 cross-section in rural areas, using the existing corridors, if possible.

Elektroprenos (Transmission company) RS needs to ensure the connection of all power plants and new consumers according to the (n-1) criterion defined by the Grid Code. If intensive construction of power plants happens, in which case the installed power of generation facilities in RS would significantly exceed domestic needs, reinforcement of networks of higher voltage levels will be necessary.

e. Electric power distribution

In the long term, the goal is to have a system with one level of medium voltage (20 kV) and one direct transformation (110/20 kV). For that reason, the development of medium voltage network is based upon two, partly connected, principles:

1. Gradual replacement of 10 kV voltage level by 20 kV
2. Gradual implementation of direct transformation 110/10 (20) kV and closing of the 35 kV network.

One of the main tasks in planning the distribution network is finding optimal transient solutions which will ensure gradual switch to a new concept while using the existing network. It is a long and uneven process the speed of which depends on local condition of 35 kV distribution network and local forecast of load increase.

Analysis of the investments needed in reconstruction and development of the electric power distribution system in the RS shows that the planned total investment until 2030 is BAM 1250 million, with a noticeable gradual decrease during the planning period, from about BAM 60 million per year, which was the amount in 2010 to about BAM 40 million in 2030 year.

Observed by the type of facilities in the distribution network, the greatest part of the investment is allocated to the secondary medium voltage network (54%), then to low voltage network (30%), while the rest goes to primary medium voltage network and systems of management, measurement and communication.

3. TREATY ESTABLISHING THE ENERGY COMMUNITY AND INTERNATIONAL RESPONSIBILITIES

The Treaty establishing the Energy Community is an agreement signed between the European Community on the one hand and Contracting Parties on the other hand. The treaty makes establishing the world's greatest internal market in electricity and natural gas possible, for the European Union on one side and nine Contracting Parties in the Southeast Europe (Albania, Bosnia and Herzegovina, Montenegro, Macedonia, Moldavia, Croatia, Serbia, Ukraine and Turkey) on the other side.

International agreements signed by BIH, that are concluded, approved and published pursuant to the constitutional laws and that are now in force, are also part of the legal system of BIH, i.e.RS, which regulates the energy sector.

3.1 International Responsibilities

The most important international agreements regarding the energy sector and signed by BIH, and therefore by RS too, are CEFTA, The Energy Charter Treaty (ECT), The Treaty establishing the Energy Community, The Stabilisation and Association Agreement(SAA) and international agreements on environmental protection (LRTAP Convention, UNFCCC, convention on biological diversity, environmental impact assessment and so on).

For creating the legal and institutional framework of the energy sector in the Republic of Srpska which is based upon the principles of efficient regulations and liberalisations of the sector, freedoms and improvements of competition, secure energy supply, and environmental protection, the signing of the Treaty establishing the Energy Community in 2006, by which BIH, and thus the RS, have committed themselves to implement the legal legacy of the community in energy sector, environment, competition and renewables, is of great importance.

Under the adopted international responsibilities, the Republic of Srpska needs to implement legal solutions contained in the heritage of the European Union into its own legal system, and to form and to harmonise its own legal framework according to the requirements and recommendations of the European Commission, whil ehaving in mind its own specific characteristics and economic and social development.

The main tasks of the Energy Community are:

- Creating stable regulatory and market framework in Southeast Europe and the EU with the aim of attracting investment in natural gas and energy sectors to ensure stable energy supply necessary for economic development and social stability.
- Creating single legal framework for energy and gas trade in Southeast Europe and the EU.
- Improving the security of supply by providing stable investment climate and enhancing connections with other European, African and Asian regions.
- Improving environment, increasing efficiency and the use of renewable energy sources in the region.
- Developing energy market competition and using the advantages of economies of scale.

By signing the Treaty establishing the Energy Community regional institutions needed for the functioning of pan-European energy market – Ministerial Council, Permanent High Level Group, Regulatory Board of the Energy Community, the Secretariat, Electricity Forum, Oil and Gas Forum and Social Forum.

By signing this Treaty, regional Contracting Parties commit themselves to create single energy and gas market to be integratedinto the European Union market and functioning in accordance with the European Union market standards. This will be achieved by gradual takeover of the legal creation of the EU, that is by implementing corresponding directives and provisions of the EU about the following areas:

- Electric power
- Gas
- Environmental protection
- Competition
- Renewable energy resources
- Energy efficiency
- Oil
- Statistics.

This Treaty establishes partially or entirely implemented directives about:

- Energy
- Environment
- Competition
- Renewable sources.

4. PLANNED INVESTMENT ACTIVITIES

Economic relation to the capital in the energy sector of the RS, on one hand, is determined by the possibility to renewing the existing and building new power plants and facilities, and on the other hand, by promoting interest in investing into energy sector business activities.

Therefore, for a stable functioning of the energy sector is of importance to establish and maintain financial stability and business success of energy activities by own and external funding sources, so that the funding of projects is structured in a way that ensures market justification and sustainability.

A large-scale development and investment cycle in energy sector is in front of the RS. It is estimated that for implementation of the RS Energy Development Strategy the value of total investments in the energy sector of the RS in the period from 2010 to2030 amounts to about BAM 11,2 billion (€ 5,75 billion), as shown in the following table.

Table 2. Total amount of the RS energy sector investment in period from 2010 to 2030.

Energy Sector Developing Programs		*Amount in millions BAM*					
		Investment period					
		2010-2015	2016-2020	2021-2025	2026-2030	*Total investments*	%
1.	Power sector	2.832	1.397	675	378	5.282	47,2%
2.	Coal sector	619	51	-	-	670	6,0%
3.	Gas sector	151	161	118	80	510	4,6%
4.	Oil sector	1.091	180	-	-	1.271	11,4%
5.	District heating sector	282	145	90	-	517	4,6%
6.	Renewable energy sources	233	240	466	766	1.705	15,2%
7.	Energy efficiency	254	267	329	375	1.226	11,0%
	Total investments	**5.462**	**2.441**	**1.678**	**1.599**	**11.180**	**100,0%**

Given the fact that the Republic of Srpska does not have that level of own accumulation of capital, the planned funds need to be provided at the international capital market. The overall effects of the energy sector investment will depend primarily on the way of financing the energy facilities, because the se are long-term, complex and financially demanding programs and projects.

The structure of future investments is hard to assess because the majority of investments will be left to the market. So, even though different financing models of Strategy implementation are possible, it is necessary to assess what level of investments will be taken over by domestic investors and what by foreign investors. That is why it is expected that the foreign investors will show the largest interest in investments in energy sector, while lower foreign funding is expected in other investments.

It is estimated that the domestic investors will partly fund their investment using achieved accumulation, and partly using loans (from domestic and international capital market) but, in this moment, it is hard to estimate what will be the ratio between those sources.

5. CONCLUSION

Development of the energy sector is one of the main initiators and pillars of the whole economic system development in every country. The significance of the electric power industry is seen in aiming its development towards competition in the European market.

The previous global economic crisis with regional socio-political aspect has significantly slowed down development dynamics of the RS, as well as BIH. Ina small open economy of BIH where the energy sector holds an important position, smart investments are needed to significantly drive the economic activity towards a specific target.

At the same time, almost every conducted research shows us that the existing organizational system of the electric power industry in the Republic of Srpska, thus in Bosnia and Herzegovina, is highly inefficient. The electric power industry needs to be market-oriented and profitable because that is the only way in which it can be competitive in the open market, influential in the region and seen asa reliable partner to domestic and international companies. Republic of Srpska has significant resources for energy sector development. Thus, creating necessary prerequisites for using those resources, as well a sincreasing the efficiency level of

the existing facilities is needed. The base of future development of Power utility of the Republic of Srpska is in availability of natural resources, coal and hydropower, as well as in tradition and experience.

In contrast to the said strengths, the fact that the society is financially limited when it comes to the financial support of future development can be seen as a weakness. The main goal of the electric power sector is to ensure a permanent and quality supply to all electric power consumers in the Republic of Srpska by the market-formed and acceptable prices, all in accordance with the principles of environmental protection.

Starting the right investments, and market and regulatory reforms in every segment of energy industry, supported by solid and structured implementation by every key actor, is of critical importance for the RS. That is the only way for the RS to ensure parallel convergence towards the EU policies, and to establish the energy industry as a driving force of development of national economy and other industriesas well.

Also, development of Power Utility of the Republic of Srpska need to be in accordance with the principles of Sustainable Development, that is, with the principle of satisfying the current needs of final products without questioning the same possibility for future generations.

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DOI: 10.7251/BMC170701327C

APPLICATION OF THE JET GROUTING TECHNIQUES FOR THE WATER PRECIPITATOR MAKING AT INNER LANDFILL OF SURFACE MINE DRMNO

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ABSTRACT

Inner landfill of open coal pit "Drmno" is anthropogenic deposit built largely from loess, dust and fine-grained sands where it is not possible to define any kind of stratification because the disposal of tailings was indiscriminate. It is a material with weak filtration characteristics ($k = 10^{-7} - 10^{-8}$ m/s) where there is present of pore water and pore pressures. Rounding inner landfill of open coal pit "Drmno" in late 2014.. was observed on the floors of many local depression and parallel gullies. It was concluded that the inner landfill of surface mine Drmno there are no conditions for the development and maintenance of the water precipitator due to poor geotechnical properties of the deposited tailings. The application of jet grouting technology in areas of landfills designated for making water precipitator can significantly improve the geotechnical characteristics of the deposited tailings so that the excavation and maintenance of water precipitator designed dimensions can be implemented without difficulty in the shortest period of time.

Key words**:** Surface mine, Inner landfill, water precipitator, jet grouting

1. INTRODUCTION

The surface mine Drmno is in the mine basin complex in Kostolac with annual production of coal of 9 million tons providing continual operations of installed thermo capacities PD TE-KO Kostolac. At the surface mine Drmno excavation of overburden is done within the framework of the five of BTO system while the transport of the same is carried out in two wings at eastern and western side of the mine. Coal exploitation is carried out along two systems of floor conveyors with common export conveyors to the location of a distributing bunker.

For the excavation of overburden there are engaged six excavators five of which works in the elevation altitude block and one bucket-excavator works in a deep altitudes. Transportation of the overburden on the eastern side of the mine is done in the BTO, IV BRO AND III BTO system, while the westward transportation of overburden is done with II BTO and I BTO system. There are five belt conveyor spreaders for disposal of excavated masses working in alternating elevation and depth work.

Coal excavations and transport is done by the BTD system including two systems of floor transporters where at the first floor work rotor excavators in tandem with self-propelled transporter in altitude work; while the bucket-excavator is positioned at the second floor and works in depth.

Inner landfill PK Drmno is of surface of ~8.4 km^2 according to the geodetic record made 30/11/2014. Volume of disposed tailings is V=436.000.000 m^3 with average thickness of H_{av}=52m. The floor of the internal landfill follows the fall of the floor III of the coal layer with general direction from the south-east to north-west, and in that direction the depth of its development increases.

The space survey at inner landfill at the end of 2014 provided the facts that landfill floors are almost horizontal with many local depressions, parallel ravines and other forms of striking surface erosion. It was also spotted the lack of possibility for construction at floors of facilities aimed to surface drainage – water precipitators due to loose material deposited there which fills and obstructs it with tailings after first rains.

Condition of mining operations at surface mine Drmno by the end of 2014 is illustrated at Figure 1.

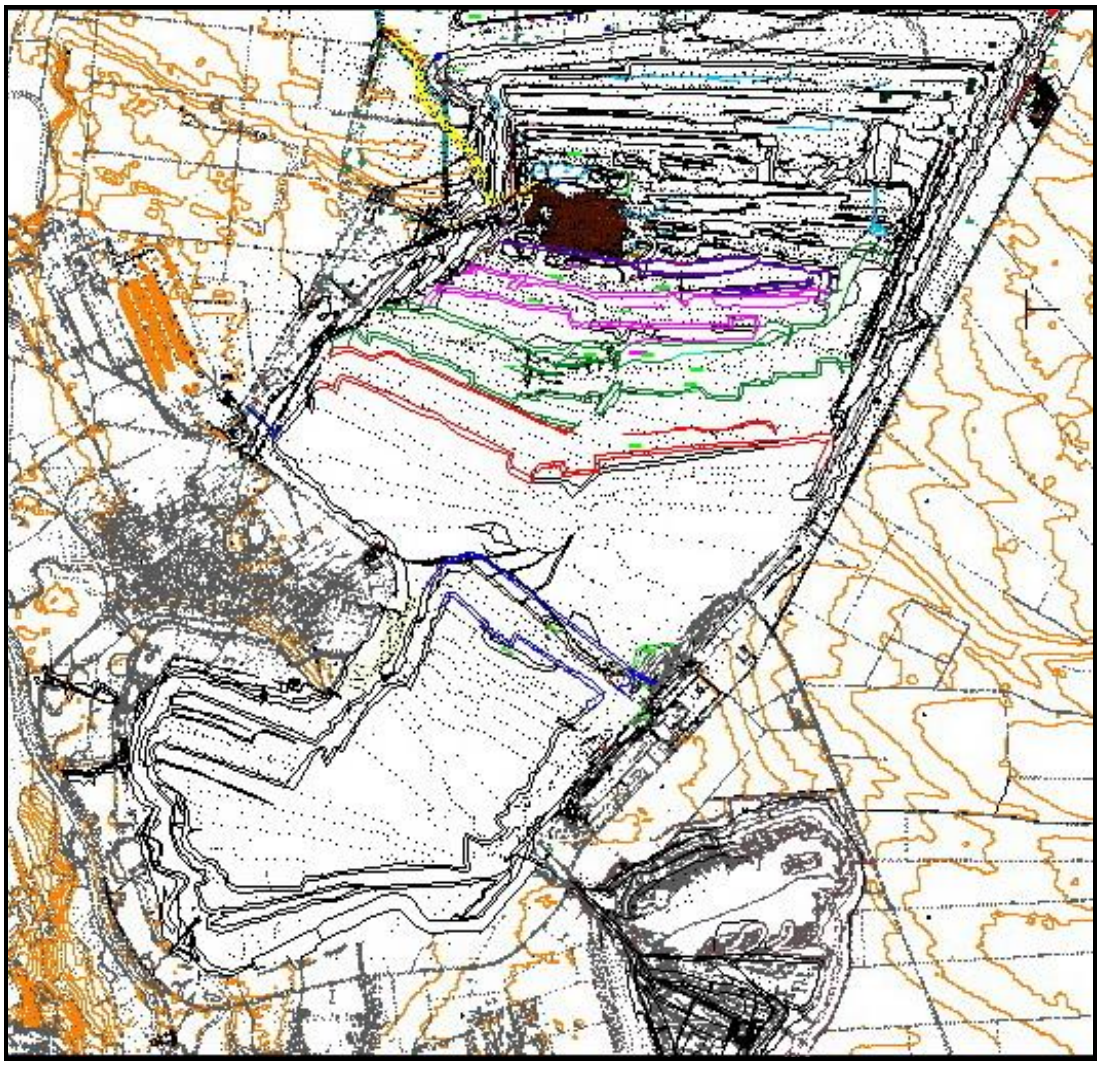


Figure 1: Condition of mining operations at surface mine Drmno by the end of 2014

2. HYDROGEOLOGY CHARACTERISTICS OF INNER LANDFILL

Inner landfill PK Drmno is anthropogenic deposit largely built from marl, dusts and fine granulated sands where is not possible to define any kind of stratification due to not-selective tailings disposal. Based on results of granulometric composition of samples taken from exploration drillings executed in internal landfill by application of empirical pattern (Hazen) there were calculated values of coefficients of filtrations (K) that goes in interval of 10^{-7} do 10^{-8} m/s. Natural humidity (W) of disposed material is 20.3%. Registered groundwater level at measured piezometers in inner landfill PK Drmno is characterised with the lack of bigger fluctuations. Almost all records of groundwater level are horizontal in longer time interval of observations which points to the absence of influence of drainage wells located around inner landfill and rainfalls to fluctuation of the ground water levels in piezometers; and to very slow process of water exchange and law permeability as consequence of small values of the filtration coefficient of disposed material.

Unlike the natural, untreated aquatic environment surrounding the inner landfill PK Drmno where free gravitational movement of ground water persists, and where the work of drainage wells resulted in the level of ground water measured in the interval from 16 to almost 70 mnm in piezometers of inner landfill with registered artesian and sub-artesian levels which point to the movement of ground water in vertical direction upwards as result of the consolidation process. Since disposed material is characterised by large presence of micropores, capillary forces are dominant in the process of binding water to the tailings particles. It is all about hygroscopic material that easily absorbshumidity but releases it hardly. Rainfalls are dominant source of humidity in the landfill. Alternating replacement of the process of disposal of new quantities of tailings, atmospheric precipitations, water infiltration and absorption by disposed masses increase the quantity of water in the landfill. There are also present pores water and pores pressures that substantially affect stability of disposed masses. Measured value of the ground water level in piezometers points to the size of pores pressures in the part of the landfill where the water collector of the piezometer is placed. With altitudes growth of disposed masses in the landfill, the pores pressures also grow which is manifested through measured NVP values in piezometers.

3. CONSTRUCTION OF PRECIPITATOR BY JET GROUTING TECHNOLOGY

Jet grouting is one of the methods of geo-technical soil stabilization. This technology is based on use of the jet of cement plaster which is under pressure and breaks the soil *in situ* mixing with it and the result is a conglomerate of the ground and cement (soilcrete) which is of high hardness and low permeability (mono fluid method). This technology was applied first time in Japan during seventies last century. Jet grouting is used for making colons and anchors and it can be applied in almost all types of soil including dusty soil, some types of clay and various types of sands and gravels.

Basic characteristic two-fluid jet grouting is that soil destruction is executed by the jet of water of huge energy and concreting by the cement milk injecting into the soil under the pressure of 500-600 bar which mix with unbound soil and particles of soil making concrete of 5-20 MB label. The work phases are the following:

1. Drilling to needed depth. In the most of the cases through drilling tool is released pressurised water to degrade in certain level the soil making easier drilling and improving condition for the final injecting.
2. When achieving given drilling depth, through drilling tool is injected cement milk and simultaneously rotates and pick up.

Complete equipment necessary for concrete colons execution by jet grouting technology is illustrated in the table 1.

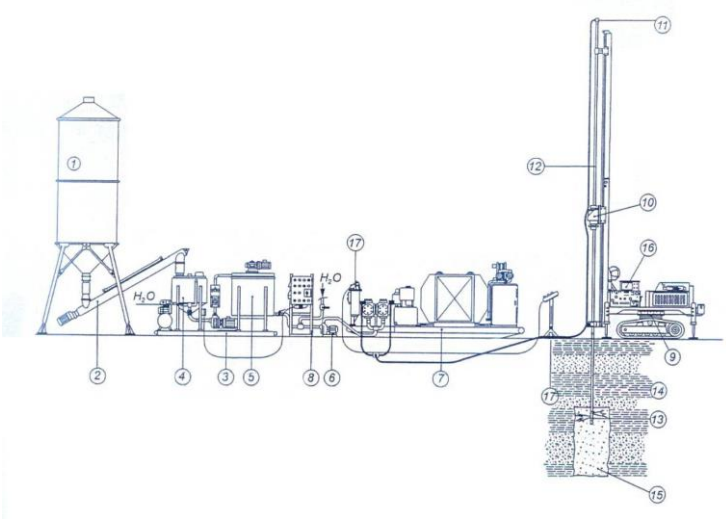


Figure 2: Complete equipment necessary for concrete colons execution by jet grouting technology

Table 1: Equipment for soil stabilization by jet grouting technology

No.	Description	No.	Description
1.	Cement storehouse	10.	Rotor
2.	Transporter	11.	Sleeve
3.	Mixer	12.	Drill hose
4.	Turbine mixer	13.	Jet monitor
5.	Exciter for solution	14.	Natural soil
6.	Pump for pre-pumping	15.	Colon of stabilised soil
7.	Main pump	16.	Control drilling device
8.	Display for manual and automatic	17.	Acoustic and visual device for
9.	Drill		

With application of mono-fluid method of jet grouting diameter of the concrete colon is maximum of 0.6 m and with two-fluid method the colon diameter is of 1.0 – 1.2 m. consumption of cement with sandy materials is of 150 – 250 kg/m^3. Time needed to make 1m' of concrete colon in sandy material of inner landfill is 100 sec.

At the space of the inner landfill PK Drmno, the technology of jet grouting would be applied for the sol stabilization in zones planned for construction of objects for surface drainage – precipitator. This method also may be applied to form walls and barriers (for example colons

30 m of depth in the first floor of inner landfill). Ash may be used for jet grouting colons of similar characteristics, as well as dust and dusty sand.

To make water precipitator of working volume of 1800 m^3 in unbound material of the landfill first step would be construction of the frame of 20 m wide and 30 m long of concrete colons with diameter of Ø 1000 mm and 4 m deep with overlapping of 10%. The project would predict construction of 456 m' of concrete colons and 13 working hours; then the material in constructed frame would be removed by a trencher up to depth of 3 m and waterproof foil would be placed as it is illustrated in the figure 3.

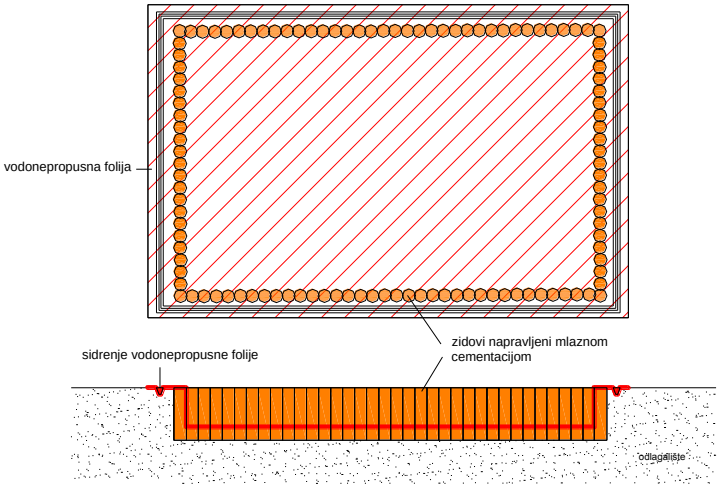


Figure 3: View of water precipitator constructed by jet grouting technology in inner landfill PK Drmno

4. CONCLUSION

The basic advantage of the jet grouting technology is that it can be applied for reinforcing and stabilization of all types of grain soil (grave, sand, dust, clay), or its application it can find in inner landfill PK Drmno which is per its lithological composition anthropogenic deposit built of loess, dust and fine grain sand where is not possible to recognise any stratification due to not-selective tailings disposal. Inner landfill PK Drmno in morphological sense is characterised with huge surface of 8.4 km^2 and average thickness of 52 m with floors having depressions, ravines and other forms of striking surface erosion. Disposed material is hygroscopic with natural humidity W=20.3% and weak filtration characteristics $K=10^{-7}$-10^{-8} m/s. Until present, in inner landfill Drmno was not possible to make and maintain facilities for surface drainage (water collectors and channels) by classical excavation and mining mechanization because of loose material which fills and obstructs it with tailings after first rains. As a result, a large amount of surface water from the inner landfill, after rain, plunges into the lowest point in the surface mine where is placed water GVS collector and pump station leading to the occasional flooding of this area and endangering of the mining mechanization and people. To pump out accumulated water from the bottom of the surface mine the main pump station had to work in maximum capacity longer time with engagement of additional pumps which provided often failures and problems with power. Construction of facilities for surface drainage in inner landfill PK Drmno would result in reduction of intake of surface water at the bottom of the surface mine, prevention of flood and overtime work of

the main pump station, and it would be created conditions or more safe work of people and mining mechanization. Due to all above mentioned the need was imposed for construction of water precipitator in the inner landfill in a way that in the zone of its construction is executed soil stabilization by jet grouting technology aiming to improve geotechnical characteristics of disposed tailings providing that excavation and maintenance of the water precipitator may be realised quickly with money and energy saving. Water precipitator with volume of 1800 m^3 needs 456 m' of concrete colons for 13 working hours, the precipitator would be transformed in the object of underground drainage prior to backfilling with tailings and it would have the function of geotechnical stabilization of disposed masses and reduction of pore pressures in the body of landfill.

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DOI: 10.7251/BMC170701333R

MODEL OF THE MINE OPTIMIZATION SUPPORTED BY THE MINING SOFTWARE SOLUTIONS ON EXAMPLE OF OMARSKA MINE

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ABSTRACT

Ore mining is a complex, and, most frequently, long-term process. Investments are very high, and the risks are significant. Hence, use of all available information about the deposit and general terms of business is highly recommended, as well as implementing the adequate planning and management techniques, starting from the initial modeling of ore reserves, over mine optimization, to operational production scheduling.

In mining industry, there are numerous computer programs and software tools, that enable more accurate and reliable planning and more efficient operation management.

In this paper, an integral mine optimization model is presented, on a real case study, iron ore Omarska Mine, ArcelorMittal Prijedor. Practical implementation and linking of several specialized software solutions for individual phases of operation is discussed: Surpac (Geovia, Dessault Systemes) for geological modelling of ore resources, Whittle (Geovia, DessaultSystemes) for optimization of the pit shell, OPMS XpacSolution (RPMGlobal) for long-term and medium-term production scheduling and XERAS (RPM Global) for financial planning and production control.

Key words**:** ore resource modelling, 3D model, mine planning, mine optimization, mining software solutions

1. INTRODUCTION

Traditional mine planning and modeling methods are based on manual calculations of the mine parameters and manual graphic interpretation of ore body, surface pit and waste dumps contours, bench maps and other mining parameters. Use of information technology in mining, during several previous decades, resulted in intensive development of specialized program packages for both geology and mining, and development of new engineering methodologies, which are significantly different than traditional methods. Using professional computer programs in all phases of mining operations, starting with ore modeling, optimization of the mine operation, to detailed production planning, has become a common practice, especially in developed countries.

The primary objective of a mining enterprise is to maximize the net present value (in further text NPV) of its cash flows [1, 2], which requires optimization of the mine operation. This objective may be achieved by optimization and integration of the entire sequence of the individual mine planning phases. An integrated model of mine engineering stages may be created, in a controlled environment, by using the blend of engineering knowledge and specialized software tools.

Practically applied integrated model for optimization of mine operations is presented in this paper, by phases, on example of iron ore mine Omarska, ArcelorMittal Prijedor. Specialized softwares were used to create geological model, block model and economical model of ore body, as well as life of mine operative model with optimized cut-off grade of metal. Practical use of integrated software packages system is presented, with 3D tools for each phase of mine operation. ArcelorMittal is using a set of modern software solutions for production planning and control, such as: Surpac (Geovia, DessaultSystemes), Whittle (Geovia, DessaultSystemes), OPMS XpacSolution (RPMGlobal) and XERAS (RPMGlobal).

This approach, which is presented in this paper, has contributed to optimization of business parameters of ArcelorMittal Prijedor mine.

2. A MODERN APPROACH TO MINE PLANNING

The economic viability of the modern day mine is highly dependent on careful planning and management. The operation and management of a large open pit mine having a life of several decades is a complex task. A number of significant issues may arise in the planning and management of a mine. Most current optimization techniques can be successfully applied to resolve these problems. There are several software applications which address ore-body modelling and ore-reserve estimation, design of optimum pits, determination of optimal production schedules, optimal blends, the best choice of equipment, its maintenance and replacement policies, design and efficient transportation and logistics network, as the most important aspects of the mine operation. This paper discusses some of these applications, with a focus on the major phases of mine planning: 3D ore modelling, pit optimization and the life of mine scheduling task.

Modern approach to the mine planning refers to merged system of engineering and economic modeling, planning and monitoring, accompanied by practical usage of advanced IT solutions, specialized for optimization of phase in mining operation. Traditional methods of engineering considered manual data processing, which meant long time for processing of data, manual contouring of ore body, geometric analysis of surface pit, stripping ratio analysis [3]. Besides that, data analysis meant relatively simple statistical interpretation. Modern methods, on the other hand, consist of computer aided data processing, with significant shortening of time for processing, and development of 3D block model of ore body, which further enables more reliable 3D planning of excavation sequence and life of mine optimization based on complex mathematical algorithms, using a wide range of input parameters.

The basic phases of the mine planning process are:

- modeling of geological block model
- modeling of economic block model
- optimization of the pit shell
- life of mine scheduling, with detailed economic evaluation.

The results of the individual phases are models, which are, actually, subsequent to each other and represent the main fundament for the following phase in the process (Figure 1).

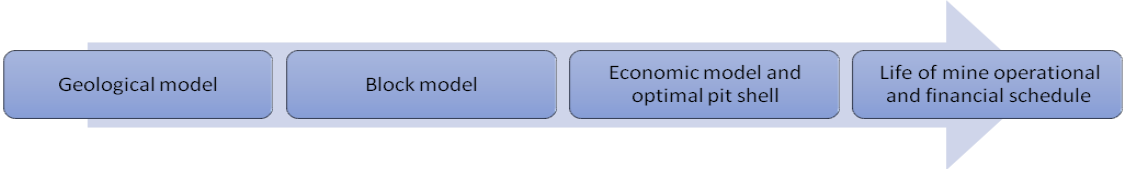


Figure 1. The mine planning process results

Geological modeling and geological block model. Veracity of the mine planning process is imposed by the precise and correct interpretation of geological characteristics of ore body. A geological model is created based on exploration drill-holes data, earlier in 2D, and with development of specialized computer programs for spatial modeling, now in 3D environment. Spatial modeling of ore body consists from several phases, which are: creating drill holes data base, positioning of sections, modelling of ore body domains, detailed statistical analysis, creating a block model and evaluation of mineral resources and the block model validation. The drill holes data base is a set of relational tables, in which the data about ore samples, collected during exploration, are stored. Geological cross-sections of ore body are then created based on the display the drill holes in 3D graphics environment. Defining the sections direction and distance between them depends on the spreading and dipping of the ore body, as well as on distribution of the drill holes in the grid. By connecting each cross-section, a closed DTM[1] areas are created, so-called solids, which represent closed geological structures of the ore body. Prior to 3D evaluation of grades for each parameter of interest, a detailed statistical analysis of the samples and its composites data is necessary.

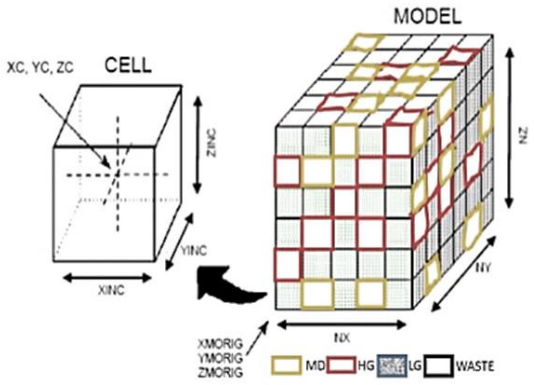


Figure 2. Schematic layout of the ore block model

Block model (Figure 2) representsa spatial model of ore body in blocks, whereby each block has certain "attributes", like grade percentage of beneficial or detrimental components of ore, volume and specific volume of ore, information on different types of ore and similar. Values of attributes per blocks are assigned or evaluated by geostatistical estimation methods. The size of the model is defined by X, Y and Z coordinates, set in such manner that the model envelopes the whole ore body. Rotation of the block model follows the grid of exploration drill holes and cross-sections through the ore body, while optimal block size is influenced by distribution of exploration drill holes in the grid and composites size.

[1] DTM - Digital Terrain Model

The most common used methods of geostatistical estimation of ore bodies are Inverse Distance, Nearest Neighbor and, the most reliable one, Ordinary Kriging. Discussion on the geostatistical estimation methods are beyond this paper, and are recommended be found in

*Practical Geostatistic*s (Isobel Clark, W. V. Harper 2000) or*The Art and Science of Resource Estimation, A practical Guide for Geologists and Engineers*, (Jacqui Coombes, 2008) or other.

Block model of the ore body, with attributes related to grades of beneficial or detrimental components, volume of each block and ore density, enables calculation of ore resources, and is a fundament for all other phases of mine planning. Hence, it is crucial that geological model is credible and reliable simulation of real geological characteristics of ore body, since all mistakes that are made in geological model are transferred to the following phases of the project.

Well known software packages for geological and block modeling in mining are Surpac (GeoviaDessaultSystemes), Vulcan (Maptek), Leapfrog (ARANZ), Datamine, Minesight, Gems and other. Surpac (GeoviaDessault Systems)was used for creation of geological and block model in this paper.

Optimization of the pit and economic block model: Nowadays, problem of optimization of the pit shell is managed by the complex algorithms incorporated into the specialized software solutions. During the optimization procedure, the geological block model is upgraded into an economic block model for the deposit, with assigned economic values to each block (in further text EVB). The value is equal to the value of the ore that can be mined, decreased by the cost of the ore mining and waste removal in order to mine the ore, as well as the cost of ore processing and marketing. Accurate estimate of the EVB plays the key role in the process of the mine optimization. Hence, the feed parameters, as well as the business strategy of the investor have to be wisely and carefully defined, since both have a great influence on the final results. The results have to represent and artefact of the engineering intelligence, and as such, must be rational and feasible.

Still, the main downside of the optimization is that the best algorithms and software tools are providing the possibility to optimize the pit using only a certain set of input parameters, implying the fact that the optimal pit shell is very conditional to the selection of the right feed. There are several methodologies used for defining of the optimal pit shell. Their mutual feature is complexity of the mathematical algorithms, needed in order to deal with the complex nature and the wide range of the feed parameters that interact with each other. Lately, the most used method is Lerchs-Grossman 3D Method, which gives the most reliable results of optimization. Other methods are Floating Cone Method, Lerchs – Grossman 2D, Linear Programming, Zhao – Kim 3D, and other.

All currently available methodologies are using ore resource block model as a matrix to determine optimal contour of the pit. In all of them, an iterative process is conducted in order to determine the list of blocks which have highest possible total value, while obeying set of spatial and technological constraints of the future pit, like maximal slope grades, metal recovery and similar.

Analytical problem of optimization can be shown as follows [4]:

Functions V, C and M are defined in each point of 3D space as follows:

$$M_{(x,y,z)} = V_{(x,y,z)} - C_{(x,y,z)} ,$$

where:
$M_{(x,y,z)}$ - *profit by unit of volume,*
$V_{(x,y,z)}$ – *value of ore by unit of volume,*
$C_{(x,y,z)}$ – *excavation cost by unit of volume.*

The problem is to find the function V, which returns the maximum of integral:

$$V_{max} = \int M(x,y,z)dxdydz$$

Deciding what material in a deposit is worth mining and processing, versus what material should be considered waste is one of the most important dilemmas to solve in mining. The optimal cutoff grade is one that strikes the correct balance between the material that will be wasted and the one that will be processed and sold. If the cutoff grade is too high, much of the mined material will go to the waste dump area. If the cutoff is too low, then the input capacity of the entire mining and mineral processing operations will be fully stretched, while revenues do not necessarily increase. Besides the grade - tonnage distribution, the optimization of cutoff grade requires the knowledge of detailed operational costs, mining sequence, mining operation, processing, and product constraints, as well as price of the product [5].

Complex mathematical algorithms behind the process of pit shell optimization impose using specialized program packages. There are several software solutions for pit optimization, such as: Whittle (Geovia, DessaultSystemes), Micromine Pit Optimisation Module (Micromine), SimSched (MiningMath), and other.

Operational and financial scheduling of the life of mine: The next step in mine planning is detailed scheduling of ore and waste excavation per years, for the whole life of mine. Scheduling the life of mine is essentially the process which incorporates economic strategy with engineering optimization, while considering geological, spatial and technological limitations [6]. This task is already partially done. while optimizing the pit shell, since annual production is set as one of feed parameters in software Whittle. However, for detailed life of mine scheduling, besides pit's geometry and main mining targets, other important mining factorsneed to be considered. In this phase, a comprehensive operational and economical evaluation takes place, in order to define necessary main and ancillary equipment, its maintenance and replacement strategy, haulage routes, waste dumps locations, ore processing parameters, administration, infrastructural facilities, reclamation strategy and other.

Complex task of the mine scheduling, and a great number of factors in conjunction with each other, demand taking advantage of modern developed IT solutions. Given the resource model, data on optimal equipment fleets, availability and utilization, main constraints and targets, life of mine model is developed, with a detailed schedule for each year. The selection of mining blocks and the most appropriate waste dumping locations is automatically managed, while ensuring all rules and constraints are honored.In the process, software determines the destination for every block that is mined, ensuring haulage is at the core of the schedule.

Quantity of ore which is planned to be excavated by years is usually a constant value, while quantity of overburden which is planned to be excavated by years can vary, depending on pit's geometry and strategy for overburden removal. Details of mining pit configurations, face

advance, minimum work-area or other calculations can be found in *Open Pit Mine Planning and Design* (Hustrulid, Kuchta, 2006).

Current software solutions for mine scheduling can be tailored for complex mining environments to deliver robust and practical schedules. The advantage of the IT supported mine scheduling is a possibility to develop a model with all the influential factors in mining considered, as well as spatial distribution of blocks of ore and waste, on their locations and time, as a fourth dimension.

Results of the operational scheduling, indicating the required number of engine hours, required maintenance, fuel and other consumables consumption may be used for the further economic evaluation of the life of mine schedule, and determination of the maximal NPV of the mine as a project.

There are compatible software tools that, in integrated system, cover both operational and financial life of mine scheduling. Good example is a set of RPMGlobal software solutions, OPMS-Xpac, for operational scheduling of the mine, and XERAS, for the financial evaluation of the operational schedule results. These two software solutions are used in the case study in this paper, for the Omarska Mine life of mine scheduling. There are other mine scheduling tools, also, such as: MineSched™ (GEOVIA, DessaultSystemes), Minemax Scheduler (Minemax), NPV Scheduler (Datamine) and other.

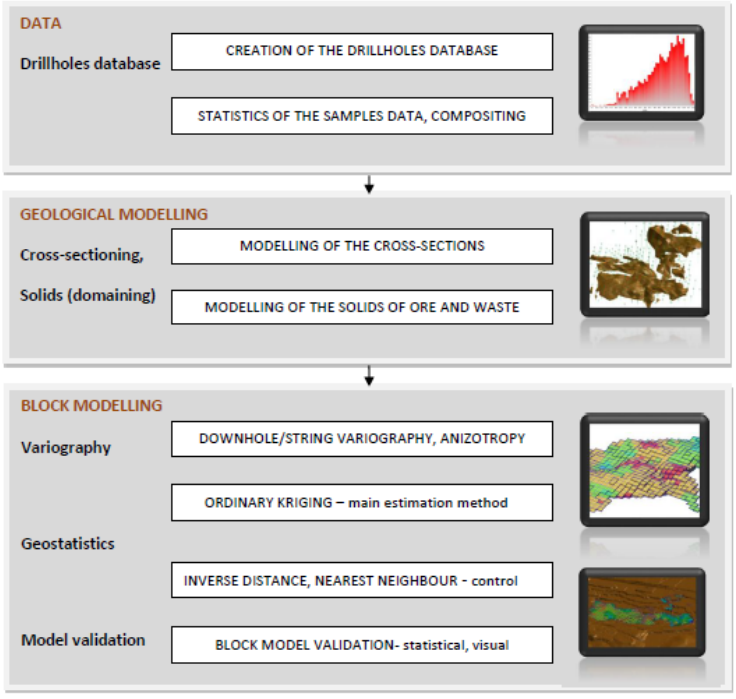


Figure 3. Phases of geological and block modelling

3. CASE STUDY – MINE PLANNING PROCESS OF OMARSKA MINE

An iron ore mine considered as a case study in this paper is an iron ore mine, located in Bosnia and Herzegovina, called Omarska Mine[2]. The mine operates under administration of ArcelorMittal Prijedor, one of the mining units of a world's largest steel producers, ArcelorMittal. The deposit where ore is currently mined is Buvac deposit.The mining of ore from Buvac started in 2008, and the deposit is partially mined out.

Buvac deposit resources and reserves block model (Figure 4a.) has been modeled using Surpac software. Geological modeling of ore bodies and its delineation from surrounding waste rocks has been done based on the exploration drillholes data. The procedure (Figure 3) is presented on Figure 3b.The main parameters of iron ore, important for production, were estimated using Ordinary Kriging, as a primary estimation method, and Inverse Distance and

Nearest Neighbor, as a control estimation methods. The block model contains estimated values for Fe, SiO2, and Mn grade, as well as grain size composition of ore. Total resources of iron ore amount to approximately 50 million tons, with almost 30 million tons of minable ore reserves inside the optimized pit shell, at the operational cutoff of 35% Fe dry [7]. Grade-tonnage curves for the remaining part of the reserves after 2016 are presented on Figure 4b.

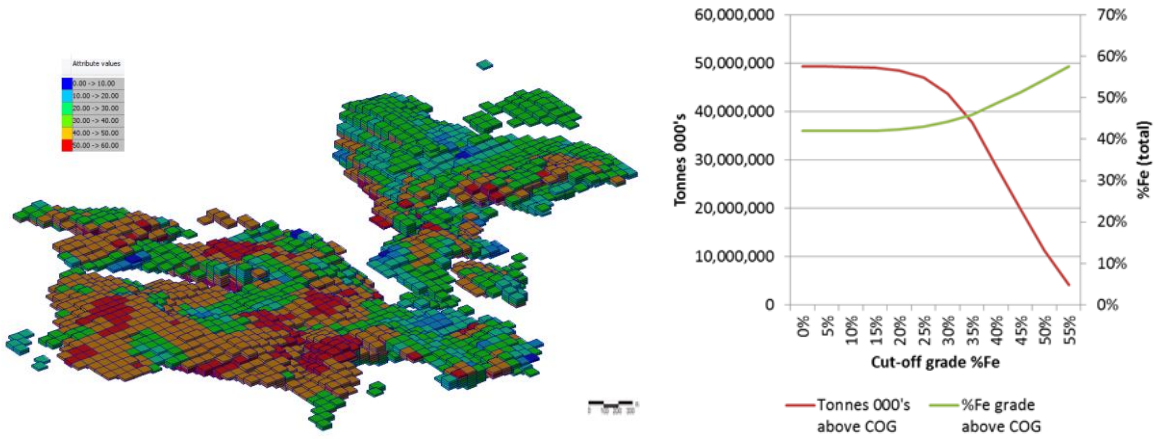


Figure 4.a) Block model of Buvac Iron Ore b) Grade-tonnage curves for the deposit

The ore resources 3D block model, created in Surpac, has been used as a base for creation of the spatial economic model of the deposit and optimization of the pit shell in Whittle software, implementing the Lerchs-Grossman algorithm. Main operation and economical parameters were estimated based on experience in the previously mined similar deposits in the Omarska Mine (Jezero deposit, Mamuze deposit).

This represents the first phase of the software integration. The true advantage of this approach, versus the traditional, manual approach, in this phase of mine engineering, is the fact that, having the right inputs, the software is enabling us to very quickly generate several pits, by variating the main parameters, such as price, cut-off grade, operational cost, or similar. This outturns aninvaluable knowledge base for any engineer dealing with the complex process of the mine planning and optimization.

[2] Real Omarska Mine data is changed, due to the confidentiality policy. However, the data used is kept representative enough to present the integrated model in this paper.

The optimal pit shell obtained in Whittle is an artifact of the block model, shaped in characteristic block structure (Figure 5a). Such shell is imported back to Surpac, which is then used for spatial engineering of the correct pit outlines, considering main mining and geotechnical constraints, such as pit slope, bench height and slope, transportation routes and other elements of the pit geometry (Figure 5b)

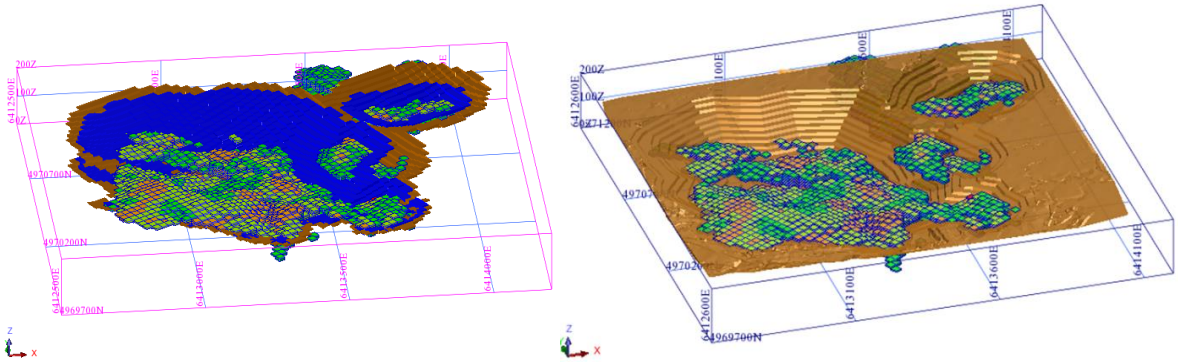


Figure 5. a)Example of pit outlines in Whittle b) Optimal pit reconstructed in Surpac

Although the general ore and waste extraction sequence is determined during the pit shell optimization in Whittle, in order to schedule the life of mine operation in detail, it is necessary to redefine the sequence in which blocks should be extracted, in the presence of several physical and technical constraints. Projected market requirements, equipment availability and possible utilization, its replacement policy, optimal transportation routes and other cost saving strategies need to be evaluated in this phase.

Again, detailed mine sequencing may be determined by the traditional methods, by geologists and mining engineers, familiar with the deposit, using primarily drillholes data, sections and level stage plans, and engineering knowledge and logic. However, several very powerful tools are developed in last decades, since the information technology found its ratification in mining. There are softwares which are user-friendly, comprehensive, and moreover, their scheduling results may be directly used in the next step – financial evaluation of the mine plan, in another, consecutive tool. Good example of such integration is the set of RPMGlobal software solutions for operational and financial mine scheduling, used in Omarska Mine.

Life of mine operational schedule is developed using OPMS-Xpac software. The block model of the ore reserves and resources remaining after 2016 and obtained optimal pit shell, as well as the mine-specific data related to time constraints, equipment data, spatial constraints or any other condition that may be specific for the mine, are used as the base in this process.

An advantage of the software supported scheduling mainly reflects in controlled integration of the entire set of the mine-related parameters, time saving, and again, the possibility to rather quickly develop a set of feasible life of mine scenarios, amongst which the optimal one may be selected. Also, powerful suite of reporting and analytical tools, such as dashboards, reports, 3D status plots which represent the deposit at any point during the schedule, as well as 3D animations to accurately visualize how the mine develops over time, allow to quickly and easily analyze, compare and communicate scheduling outcomes. Stage plans can be exported as triangulated surfaces to inform other applications of how far the mine will have progressed at any point in time. Figure 6 shows a set of the reporting tools of the OPMS_Xpac, which are very useful in the process of decision-making.

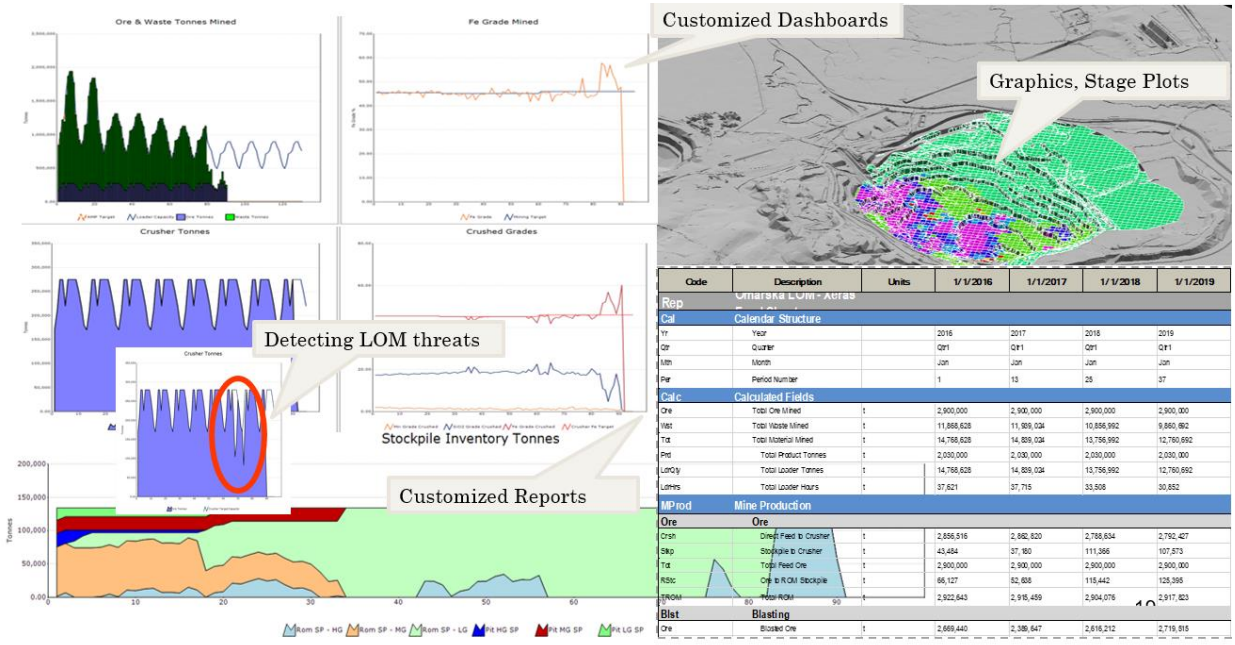


Figure 6. Reporting and analytical tools of the OPMS_Xpac

The results of the operational schedule of ore mining on Buvac pit were imported into XERAS, where financial aspects were evaluated, considering operational costs, manpower costs, consumables cost, CAPEX, depreciation, inflation rates and other financial parameters throughout the life of mine. The software enabled fast and accurate financial planning, easy importing of data and creating customizable reports, which allows more time for refinement of the model and analyzing different scenarios, to determine the optimal solution, with the highest NPV.

Over time, when the mining starts, the selected mine schedule with the highest NPV needs to be occasionally questioned, considering important changes in operational and economic aspects. There, the software tools usage and integration of the results and models, come to be very convenient.

4. CONCLUSION

Throughout the years, in mine planning, new processes and software technologies have appeared and replaced the traditional ones, because of their benefits when applying them in a real life. They enabled usage of more complex mathematical algorithms, with better and more reliable outputs.

This paper shows, on a real mine case study, how a set of integrated mining specialized software solutions may be used to advance the mine planning process. The main benefit of this approach is a possibility to integrate and directly use models from one phase in another phase and another tool, which brings additional control over the process.

Another advantage of the IT supported scheduling is the possibility to simulate a great number of scenarios in a short time, evaluate them to ensure that the maximal value of the mine is gained.

The data presented in the paper are kept general, without too much detailing on the mine capacities, financial outcomes, implemented technologies, used equipment and other mine-specific data, since the aim of the paper was to show the benefits and advantages of the overall integration of the mine planning phases in the computer aided environment, where a result of one phase and one software output, represents a direct feed to another phase, and to another tool. That way, overall process, which is driven by the engineering logic and upgraded knowledge base on the deposit itself, emerges optimal result of the mine operation.

However, if the reader is interested in a more detailed presentation of the real mine modelling and planning, authors are available for further discussion.

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DOI: 10.7251/BMC170701343A

DYNAMIC MODELING OF ORGANIZATIONAL STRUCTURE IN THE CONTEXT OF HUMAN RESOURCES OPTIMIZATION

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ABSTRACT

Dynamic modeling of organization that supports qualitative management of a company's resources requires development of a client-oriented organization structure that should respondto complex valuative requirements of interested parties (stakeholders). Stakeholders' satisfaction factors are numerous, but in the center of the scene there is always a man as the creator and beneficiary of this value. In a complex business environment, requirements of stakeholders demand for business response that is market acceptable, fast, correct, and in compliance with standards. Market acceptance of products or service and stakeholders' requirements are the subject of a continuing adjustment of the organizational structure and employees as the carrier of knowledge and skills needed to execute job tasks that result from the purpose of the very existence of the company. This work presents a practical framework for organization of a company that came out as an answer to insecure and inconsistent legal frame for effective management of a company, as well as a desire to offer to companies modern tools, know-how and solutions to dynamic optimization of organizational structure in accordance with international standards.

Key words: Dynamic model, processes, resources, organization, objectives, optimization, plan, and Key Performance Indicators (KPI).

1. INTRODUCTION

Law is binding up an employer, which employs more than 15 employees, to normatively regulate employment and thus establish organization of work posts as a base that enables him to meet the legal frame for regulation of relations between an employee and employer. Companies tend to turn to normative regulation of relations with an employee by adopting the following documents: Employment Rule Book, and Organization and Work Posts' Systematization Rule Book. Organization and Work Posts' Systematization Rule Book usually establishes a functional organizational structure that through functional organizational parts treats 'work posts' as the lowest functional organizational units (OU). Job tasks that an employee should execute i.e. do are usually distributed to the work post. To recruit a vacant work post, certain performances are required from an employee in order to do the job tasks distributed to this work post. Recruitment of the work post is done by an employment contract, which explicitly states that the employee is assigned to a certain work post defined

by Organization and Work Posts' Systematization Rule Book. Work posts are an integral part of an organization, and the human resources, as the carriers of knowledge and skills, are treated as executors of job tasks of work posts under conditions and time limit as agreed in the employment contract. At first sight this model contains important elements of organization, but the problem arises when the business environment requests quick adaptation of organizational structure of a company. This means that any slightly more significant addition or deduction of job tasks that an employee should perform within his/her competencies would imply adaptation of provisions of Organization and Work Posts' Rule Book. If we add to this a request to transfer an employee from one OU to another, the situation becomes additionally complicated. To support such requested adaptations in job description of the employee and the transfer from one OU to another, it is necessary to amend internal regulatory documents that regulate this area. Procedure to alter and amend internal documents is complicated and time consuming. We may freely say that such 'usual' model of organization is inflexible and static.

Addition or deduction of job tasks from an employee requires change of job description in the organization rule book. un adapted rule book creates a legal insecurity both for employer and employee. it becomes a source of frustrations that often escalate and end at court. the epilogue of the court procedure is usually for the benefit of the employee. this is because the company has no internal legislation that would support a demanding organizational flexibility. when generally observing the usual model of organization at any company, the following questions arise: how to integrally and optimally manage resources of a company? how to create a flexible organization of a company's business resources? how to meet requirements of stakeholders [1]? how to measure success of a company's organization?

The aim of this work is to publish solutions that originated in practice as a compromise between requirements of the lawmaker that prefers a social liberal political system on one side, and a company that has to continuingly optimize its resources in the selfless fight to survive on the other.

2. MODELING COMPANY'S ORGANIZATIONAL STRUCTURE

It is clear that every company is made with some purpose, or rather 'mission', and that led by certain values, it recognizes certain strategic objectives through its vision. Map of strategic objectives is articulated by a strategic plan, out of which again tactical and operational plans with their specific objectives come out. It is evident that a company, in accordance with its business model,has to plan, engage, then organize needed resources, and through process activities achieve results that will more or less meet the expectations of stakeholders in order to accomplish corporate objectives.

2.1. Modeling business processes

Map of organizational structure is based on business model and strategy, and it begins with 'General process model', i.e. with business processes modeling*(BPM)* [2].There is the most common classification of processes into three main groups i.e. macro-processes: Management processes, Realization/Operating processes, and Support processes[3].Every macro-process has its own processeswhich include certain **tasks** as sub processes of processes, simply named *individual processes* [3]. Further division of tasks to activities and operations is not the subject of this work. Key postulate is that certain processes, i.e. tasks demand for certain resources in order to be carried out. The question is who is primarily responsible for mapping

these processes and by that itself for defining resources needed to reach the key performance indicators *(KPI)* [4,5]? Who can be that person? This person is someone with an adequate knowledge of the process. This answer leads us to conclusion that we need two key resources: **person** and **knowledge**. It is evident that the knowledge as the resource is limitless, but the person has limited knowledge and **time**, as the third resource. Therefore, we need more persons with certain specialist knowledgewho would assume authority and responsibility to design the process and at the same time assume the role of 'the process owner' [6]. 'Process owner' creates process documents, organizes the method of validation and measuring of process objectives through validation and measuring of process tasks. General process model represents the general map of business activities that arise from the business model and the management of macro-processes falls into exclusive authority of the company's general manager. The general manager delegates rights and responsibilities to design individual processes by designating 'process owners'. The process owner does not have to be a company's employee. It can be any 'outsource' person.

Process owner breaks down a process to individual tasks, then forms a catalogue of tasks, assesses the relative value of each task in accordance with criteria for task valuation, estimates duration of tasks, and determines competencies that are required to perform the task.

What is left at the end is to redistribute the tasks through functional organizational parts i.e. Organization Units (OU) to work posts (WP) as the simplest OU. Therefore, the functional organizational structure needs to be modeled.

2.2. Modeling functional organizational structure

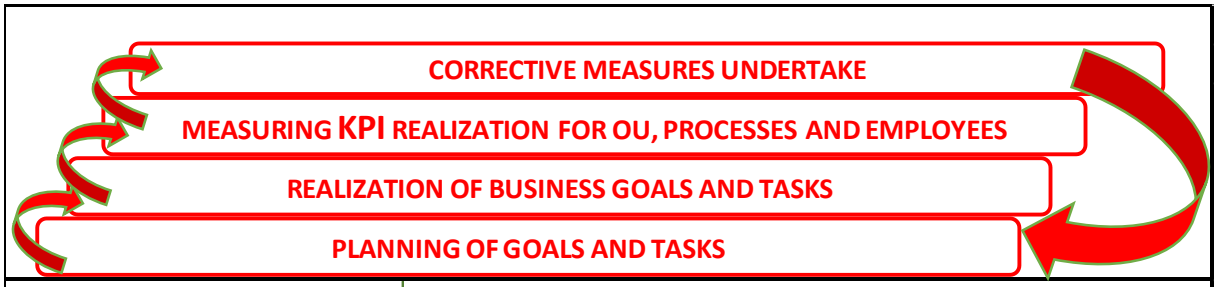
Usual model of organizational map is a functional organization that gets distributed depthwise and withwise into functional OU. Such 'standard' model of organizational map may be establishedin divisional and territorial principle with adjustment of depth and width to corporate requirements. The final instance of a functional organization is a work post (WP) as the lowest functional organizational form.

By intercrossing the OU, including WP, with processes i.e. tasks we get Process-functional organizational structure i.e. *Model of matrix organizational structure* [7].

2.3. Matrix organizational structure

Matrix organizational structure (example Figure 1) offers a compromising solution for organization and redistribution of tasks to OU that on one side provides process knowledge and process authority (process owner), and on the other functional authorization (manager of OU) to organize execution of tasks distributed to OU. The manager of OU has the authority to redistribute the tasks to work posts in accordance with corporate requirements and market demands.

Prevalence of tasks distributed to a work post determines the functional title of the work post and required competencies to perform the mentioned tasks. The method of determining the title of a work post and required competencies to perform the tasks is not the subject of this work. Therefore, we shall move on to the area that deals with planning an organizational structure.



X marks that in the specified functional organizational part, certain tasks are carried out, which the owner of the process should recognize and delegate to the manager of OU, and then the manager of OU should redistribute these to the WP	Functional organizational part															
Prcess title	DIRECTORATE	Power generation sector	Deaprtment 1...	Department 2....	Coal production sector	Exploitation department	Transport department	Support sector	Warehouse	Maintenance	Procurement	Administration	Project 'Sector'	Project 1	Project 2	Sector' of outsource tasks
SUPPORT PROCESSES																
Legal support	X	X		X							X	X		X	X	X
General support												X				
Accounting	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X
Etc.																
MANAGEMENT PROCESSES																
Human resources management	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X
Risk management	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X
Quality management	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X
Etc.																
REALIZATION PROCESSES																
Coal production					X	X										
Etc.																

Figure 1 Model Matrix organizational structure (shortened example)

2.4. Planning an organizational structure

It is a fact that a strategic document may project the general organizational structure that supports strategic objectives, but the practice imposes tactical and operational planning of organizational structure.

2.5. Tactical planning of organizational structure

By modeling the organization, we have recognized three resources, and those are: knowledge, people, and time. People, the performers of a work, have limited time. Execution of a job task requires competency of the performer (knowledge and skills) and time. Two kinds of time appear. Time needed to complete some work (***Planned time to execute job tasks expressed in hours***) and time during which the performer (employee) is available. We will call the time needed to perform some work as 'time normative for execution of job task', and the time during which an employee is available as 'planned effective working hours of a performer'.

We have envisaged in the model **a relative value of a task** that presents, in a sense of plan, a base for determining a planned financial value of a job. Figure 2 presents a model of planning job tasks through processes.

Process title and description of belonging tasks in that process	Relative value of job/hour	Planned time for job execution in hours	Relative value of job	Planned value of job in BAM
1	2	3	4=2*3	5=4*Z¹
SUPPORT PROCESSES				
Legal support				
Develops drafts of business agreements				
Develops drafts of general documents of the company				
Provides legal interpretation of laws and internal documents				
etc.				
General support				
Develops plan of activities of general support				
etc.				
A-Sum of planned hours for execution of all planned tasks:		**Σ3**		
B-Relative value of all planned tasks:			**Σ4**	
C-Value of planned tasks in BAM:				**Σ5**
(Z¹-Presents value of an hour in BAM for the relative coefficient 1)				

Figure 2 Annual plan of job tasks presented through processes

Annual, i.e. tactical plan of job tasks by processes provide us an answer to the question – How many effective hours is needed annually to perform all planned tasks? The answer is the sum of all hours for all planned tasks. The question – How much is that all going to cost us? The answer is the sum of all multiplications of relative values of job tasks and time duration of job tasks multiplied by the worth of the relative coefficient 1 (one).

We have received answer to the question of how long every job tasks individually lasts and costs on annual base. We can calculate how many employees we have to engage to perform all the jobs tasks based on the total number of hours for all job tasks divided by the effective hours of one employee on annual base. Such calculation is used as a landmark, not as the rule, to estimate the number of employees. Question arises – How to plan employees? The answer is a logical distribution and grouping of job tasks through OU. Afterwards, the managers of OU redistribute the tasks to WP and they do it in several iterations until an optimal solution for the organization is reached. In the end we reach the result that places undistributed tasks in to 'OUTSOURCE' OU where they are organized in processes. Job tasks in 'OUTSOURCE' OU, beside the title of the job task, contain also the estimated value and planned duration, which present a subject to contract with outsource contractors.

2.6. Operational planning of organizational structure

Operational planning of organizational structure belongs to direct managers and it is carried out daily, weekly, decadic, and monthly. Such planning may be called implementation oforganization or realization of tactical plan.

2.7. Method of implementation of organization

We start from an assumption that the strategic goals (Financial perspective, Client perspective, and Process perspective) [8] are assigned to the general manager of a company and that these goals are a source for tactical planning (which implies planning of organization and systematization of work posts) and the general manager is in charge to redistribute tactical goals to lower management level (Figure 3). Real business life asks for being operative, meaning that the tactical goals flow into operational tasks that have their time goals and norms. Therefore, operational managers with their direct executors get into reaction with the cruel market reality, where they are themselves present as the work force. Efficient business operations require, besides others, efficient management of the work force as the most complex resource. The goal of this work is not to treat the complete human resource management, but to present models for designing a tool that supports a dynamic model of efficient recognition and engagement of the best subjects for performing the required job tasks.

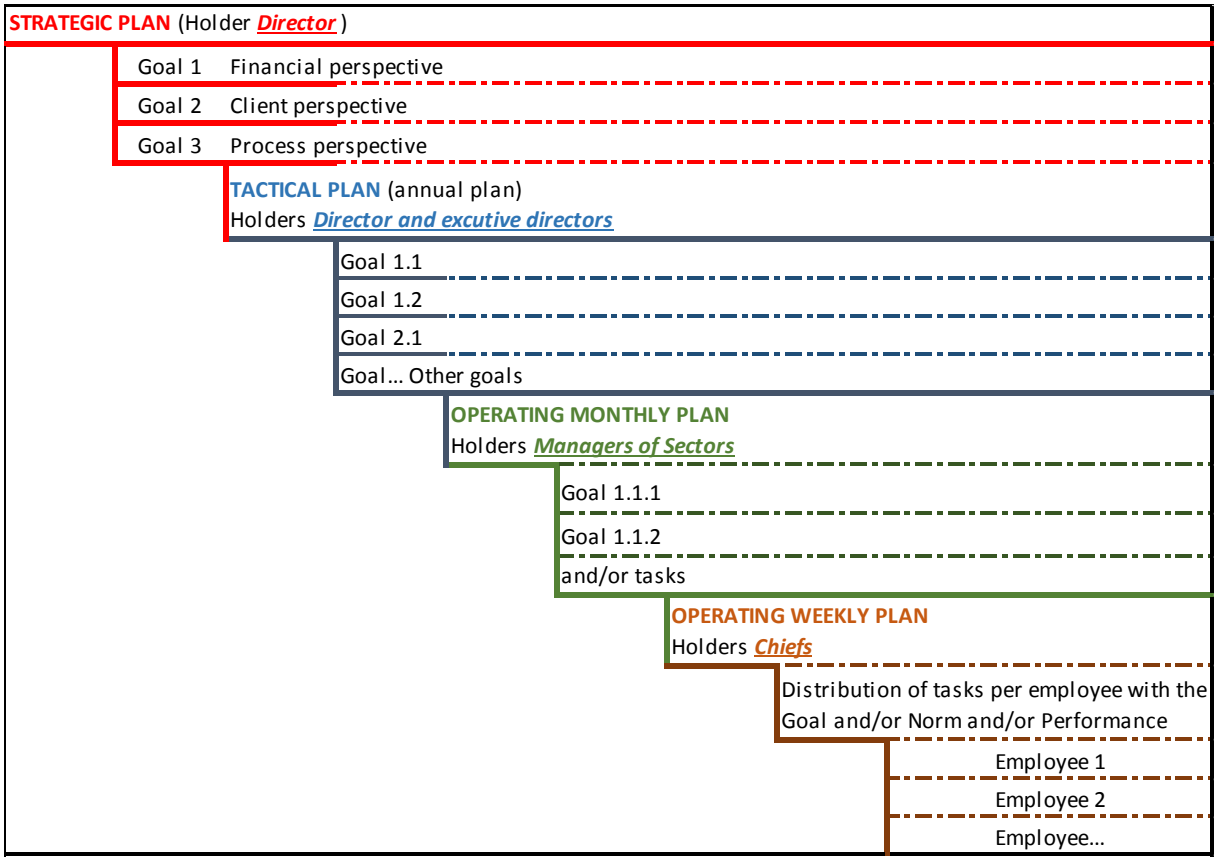


Figure 3 Map of distribution of goals and job tasks

2.8. Operational implementation of organization and measuring the key indicators

Market demands are becoming more stringent and if a company wants to survive, it has to meet those demands. Adjustment means flexibility of an organization and flexibility of a work power. There is no long-term organizational structure any more. Now organization strives more and more to the project structure. This means that OU is formed when needed and not forever. It has become a demand, almost on a daily basis, to adjust organizational structure depthwise and widthwise. Better to say, conditions must be met in order to form and/or close an OU when needed, i.e. to decrease a number of employees in accordance with the scope of assigned job tasks. Therefore, the demand is to have a changeable number of engaged employees and therefore the number of work posts too. Employment Rule Book and Organization Rule Book are supported by 'Deming' application that enables negotiation of price of work with an employee and the scope of job tasks that an employee can perform competently. Assignment of job tasks to employee may be done on daily basis by issuing a writtenIndividual Work Order on assignment of job tasks. Decision contains job tasks arranged by processes that the employee should perform, as well as their relatedness to OU, also goals and norms, supervision of a manager and so on. The collection of all Individual Work Order on assignment of job tasks on a specific day represents the actual Organization and Systematization of WP. Besides, an employee receives a work log that is adjusted to the Individual Work Order and at the same time it is a tool to measure employee's performances and a tally sheet for calculation of salary.

3. CONCLUSION

Process and functional approach to managing an organization of a client oriented company demands for tactical and operational redistribution of strategic goals by processes and lower organizational levels. Actualization of business objectives require engagement and use of resources in an effective and efficient way by managing performances of organization through managing performances of OU and employees.

'Deming' application, with its accompanying deeds and documents, offers a 'clean' situation and modern tools to measure the Key Performance Indicators (KPI) and establishes fair and correct relations between employees and employer. Transfer of employees from one OU to another is simple. Employee may work in several OU at the same time. He/she always knows how much he/she has earned based on effective work, also how high was his/her performance rated, and what monetary award should he/she expect for the work. Managers of OU are given a tool for the effective management of pertaining processes and employees by dynamic monitoring of KPI, based on which they can operatively undertake certain corrective actions in order to optimize business performances.

Information on tactical actualization of the plan by continuously monitoring the KPI provides an opportunity for the company to 'shift' and adjust organization on daily basis, i.e. to efficiently and quickly perform organization's restructuring in accordance with market conditions, business standards, and the law.

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DOI: 10.7251/BMC170701351R

PROJECT OF SYSTEM FOR COAL QUALITY MANAGEMENT IN EPS WITH THE EXPECTED POSITIVE EFFECTS

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ABSTRACT

Primary goal of coal exploitation at open cast mines is to provide sufficient amounts of fuel with necessary quality for the combustion in power plants and with minimal costs, along with the respect of the highest environmental standards.

Considering a long period of coal exploitation in Kolubara coal basin, some of its parts with good and consistent quality are already finished. Availability of deposits with an equable quality has been reduced and variations are more and more expressed.

In Kolubara mines new system has been developed and is in a progress of realization through project of management and coal quality control – homogenization, which will enable excavation of parts of deposit with changeable and lower coal quality. Benefits of this system will be significant for both mine and power plant.

Key words: coal exploitation, quality, homogenization, control, benefit.

1. INTRODUCTION

Significance of coal, as one of the main raw material, for electric production is more and more expressed. Participation of coal in total reserves of Serbia, in percentages, is 84%. Because of that it is important to invest resources for the purposes of mining sector development, procurement of new equipment, to secure stability in supply for PP and respect toward legislation and regulative in area of environment protection.

In lignite mines long-time period of use and deposit's exploitation led to the appearance that the best parts were already finished, and the work on coal digging, in time, goes toward parts with more expressed complex conditions. This is related, primarily, on changes in terms of depth of coal seams and quality in them.

In order to valorise all useful parts of deposit, it is necessary to implement system for coal quality control and management – homogenisation, which will enable equalization of coal according to required value of the observed parameter from the excavation site, transportation and before train's loading. Thereby, an economical combustion of coal in power plant's boilers will be achieved without expressed oscillations.

2. EXPERIENCES IN FOREIGN MINING COMPANIES

Lignite is one of the world's basic resource, and it's main use is for production of electric energy. Global increase of demand for electric energy, without possibility of compensation with some other fuel, along with opportunity for mass production of coal at open cast mines, is one of the reasons for introduction of system which enables controlled excavation of all parts within coal deposit for the purposes of getting demanded quality. Newly built power plants have express demands, not only for the quality, but also for the protection of environmental from nus-products during coal combustion.

The biggest mining companies in Europe, over the time, developed their own concepts for coal quality management according to their specifics and requirements of consumers.

2.1. German experiences

Germany is, more than one century, a leader in lignite production, along with production of mechanization for the work at open cast mines. Currently, within four lignite basins (Rheine, Mittelhoch, Hallmstedter and Lusatian), production is more than 200 mill.tons of coal, and 26% of total electric energy is produced from it. For this large production Germany was the first country which started development of system and implementation of processes which enables procedure for excavation of coal with different quality, regarding observed parameters, and mixing in order to achieve required demand. [2]

Considering specifics of each coal basin and defined open cast mines within them, independent systems were developed and they enables fulfilment of all demands for the quality according to final consumers. Joint thing to all of these systems is approach to the problem, development of activities according to the requirements and making of integrated model for coal quality management.

According to the previous experiences and customer's statements this system for coal quality management guaranties compliance of contractual obligations and ecological parameters along with significant decrease of costs for electric production. Additional meaning in implementation of system lies in increase of technological discipline of all employees.

2.2. Greek experiences

In Greece coal is basic energy resource, and production of energy with coal combustion meets 70-75% of its needs. Within two coal basins (Ptolomais and Megalopolis) with production of 60 mill.tons, different and uneven characteristic of coal led to development of system for coal quality management. [2]

Concept of coal management in both basins has been achieved with homogenisation at stockyard. To fulfil this, it was necessary to make good exploration of deposit, production planning and tracking of excavated coal. For that, planning works are previously done according to geological model, plans of customers' consumption, preparation of operational working plan for the mechanization, and coal transportation to the stockyard. With corresponding method of coal disposal (mixing/deposition) at stockyard the harmonisation of coal's required quality is achieved.

From the stockyard, coal is taken and transported toward customers, with previous control via online analysers before and after mixing.

2.3. Bulgarian experiences

In Bulgaria there are large reserves of lignite in basin Marica East, at open cast mines: Trojanovo 1,Trojanovo north and Trojanovo 3,with annual production of 25 mill.t. [2]

Changeability in coal's characteristics led to introduction of system for coal quality management which is reflected in implementation of certain activities to achieve harmonisation of quality parameters before combustion in power plants. Essence of concept, for all open cast mines, is reflected in:

- geological exploration works and development of geological model of quality,
- development of technological model for mechanisation's work according to predicted quality and capacity,
- control, comparison and processing of realized and planned data,
- use of online measurement instruments and scales for tracking information at the excavators,
- use of devices for measurement of ash and moisture content in coal, and devices for weight measurement at the belt conveyers for control of quality and capacity.

Equalization of quality is achieved during excavation on benches by digging the sub-benches with different content of observed parameter, and limitation of capacity for the excavator during their work in coal with different characteristics. Further on, mass flow with corresponding quality and capacities is controlled on conveyers, and during loading into wagons, too.

3. CONCEPT OF SYSTEM FOR COAL QUALITY MANAGEMENT

3.1. Problems in production regarding coal quality

Coal is the most important energetic potential in Serbia, and in structure of total energy reserves participate with over 84%. Main goal in exploitation of coal at open cast mines is to procure sufficient amount of fuel for combustion in power plants, with appropriate quality and minimal costs. [1]

On the area of Kolubara coal basin availability of deposits with equable quality decreased, and variability increases. In order to satisfy demands for realisation of planned balance for coal's excavation and demands for minimum requested quality for power plants, it is necessary to implement system which will allow rational use of mining resources, increases reliability of thermo capacities and optimal use of power plants, improves environment's demands and reduce negative impacts on it, provides production of cheaper electricity.

3.2. Concept of adopted system of coal quality management

The main objective of system's implementation is excavation and distribution of coal for power plants with sufficient amounts, required quality with minimum excavation expenses and maximum usage of deposit.

Conceptual solution for achieving this goal is possible through activities that involves:

1. making of geological model.
2. making of technological model,
3. basic planning, simulation and excavator's work analyses,
4. control of material flow with use of online analysers and belt scales,
5. use of the stockyard,
6. delivery of coal (needed quality and quantity for power plant) with reporting - automatic sending of certification for train immediately after loading [3]

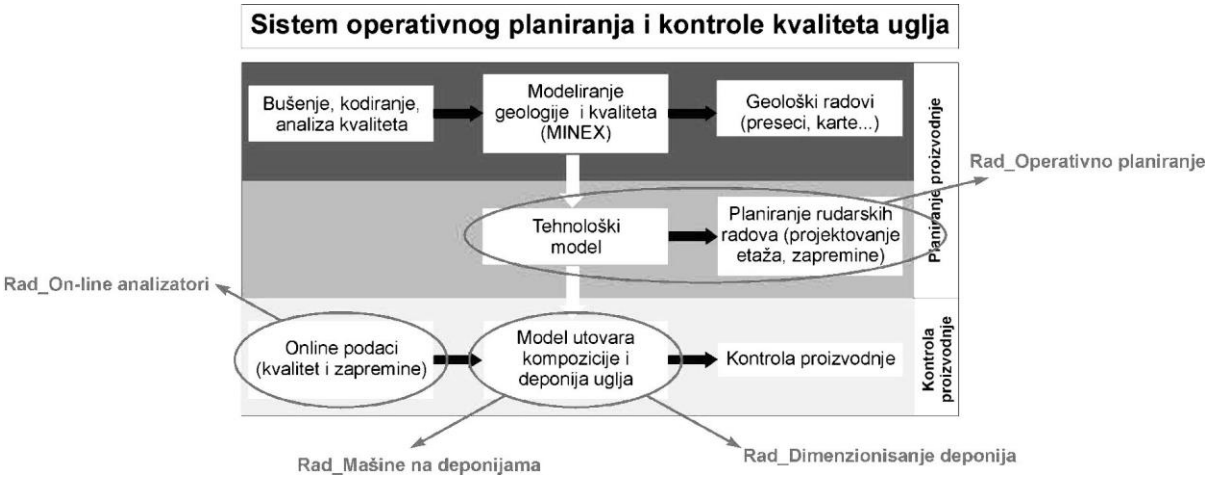


Figure 1. Concept of CQMS

1. **Geological model** - is done in software package MINEX, which is support to mining planning for stratified deposits. From data base certain structural information are at disposal: location and geometry of exploration boreholes, lithology of members, sample analysis and data of stratigraphy eg. defined layers, etc. These data are in further use for technological modeling. [3]

2. **Technological modeling** - based on the planned balance for coal excavation, technical characteristics of equipment and geological conditions inside the deposit, technological blocks anticipated for excavation are created through the vertical division of benches. For the blocks, defined in MINEX, informations for further modeling will be assigned, and technological blocks are created. The way of selective work, number of sub-benches and quality for each of them along with the capacity of excavators is determined during formation of technological blocks. [3]

3. For the purposes of coal's production optimization in function of demanded quality and capacity for equipment in use, it is necessary to develop and apply **operational planning**, based to the previously elaborated plans at different levels.

Operational planning is defined through development of technological model for excavator's work, and determination of:

- production capacity for each excavator on daily basis for every technological block,
- production plan with coal's needed quality for each excavator in all technological block on daily basis.

Next activity in operational planning is **simulation of excavator's work** and analysis for created technological blocks on all benches, along with transportation of material to destination (loading into wagons or stockyard).

If simulation of chosen work for excavators satisfy for the quality and capacity criteria, plan is put on realization. If not, new technological blocks will be created with certain alterations, and new simulation starts until fulfillment of all criteria, or coal with some other known quality will be send to stockyard and disposed with use of Strata method. Then, such plan is forwarded to the corresponding data base, from where is used for mechanization 's work. [3]

4. Control of excavator's work with use of **belt scales and online analyzers** – in order to achieve better control of excavator's work and to provide faster reaction for eventual corrections in system work, it is important to install online analyzers and belt scales as integral part of system for coal quality management. These devices are positioned at specific places and provide real time data about quality and capacity of excavated coal.

Use of scales and analyzers provides tracking of material's flow from source to destination, and possibility to compare parameter's values in real time and planned values from the corresponding models in software. Like this it's possible to compare and process real time data and planned data and eventually to use them for further planning purposes.

5. **Stockyard** - In case that it's not possible to achieve set goals regarding quality and capacity of excavated coal during some work period, it is important to develop model for coal's

disposal on stockyard (old and new) which will enable homogenisation of coal with different quality until fulfilment of required quality.

Through this model it is possible to know exact position, quantity and quality of coal at each part of stockyard. This state of piles on stockyard is consequence of detailed planning for coal's disposal with known average quality which comes from different places inside open cast mine and has different values in quality. According to Strata method of disposal it is certain that average requested quality in piles will be achieved, and after that it can be taken with reclamer and transporter to the loading place in wagons.

6. The ultimate goal of the system for management and quality control of coal is delivery of product with particular characteristics for the consumer. Control of product and issuance of certificate is final activity in process of coal management.

Before loading in wagons, coal has to have requested values of observed parameter, and accordingly certification eg.declaration of train is automatically sent after finished loading.

4. EFFECTS OF INTRODUCTION OF SYSTEM FOR COAL QUALITY MANAGEMENT

Success of system for coal quality management is best seen in integrated system of open cast mine-power plant.

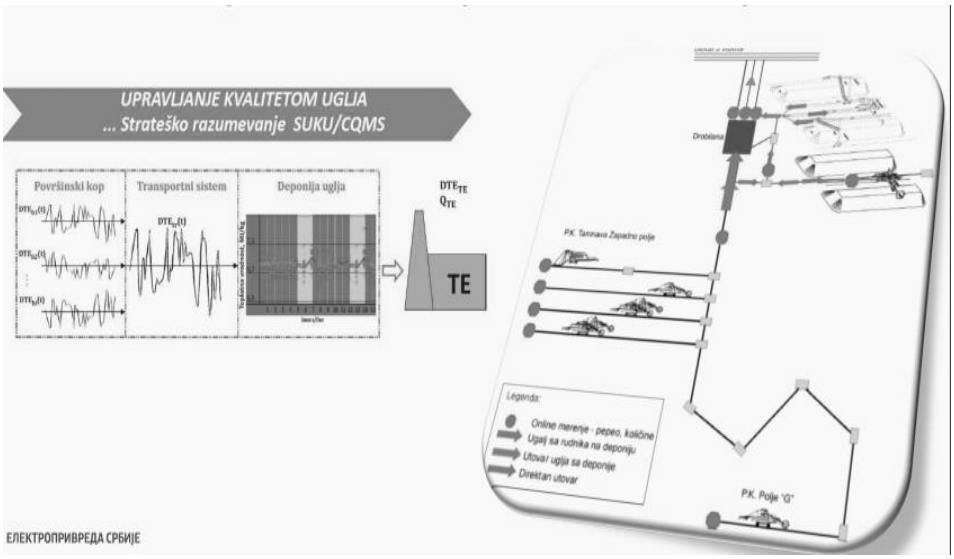


Figure 4: Integrated system OPC-PP with implementation of CQMS

Work on coal exploatation without posibility of quality control can have negative impact on financial effects for payment of raw material, less utilisation of deposit, decrease in capacity of basic mechanisation, bigger production expences, more intensive negative impacts on enviroment, shorter working life for open cast mines etc.

Negative impacts in power plant, without introduction of this system, are reflected in increase of liquid fuel consumption, overheating and damage of burner, increase of ash concentration, bigger transport expenses, larger degree of abrasion of some sets in process of coal preparation, increased slogging in boilers, decrease of combustion efficiency and utilisation of boilers, greater emission of pollutant and negative impact on environment etc.

With implementation of CQMS in coal deposits with changeable characteristics (larger stratification and spatial change in quality, content of ash, moisture, sulphure..), an optimal working plan will be defined in order to provide continual delivery of requested quantity wit needed quality for the power plants.

From the aspect of producer-open cast mine, positive effects of CQMS implementations reflects through optimisation of deposit, larger total amount for excavation, longer working period for mine, increase of economic effects through decrease of excavating expenses, less transport and disposal of bad quality coal, market valorisation of coal with low quality , decrease of negative impacts on environment etc.

From the position of power plant, as an consumer, positive effects of CQMS introduction are improving of combustion conditions, increase of utility in boilers, reduce of maintaining costs, decrease of consumption and costs for electricity and additional fuel for own needs, optimisation the usage of coal, increase of productivity on generators, work optimisation, achieving of favourable environment conditions (reduction of emissions, stabilization of the ash amount, reducing the amount of sulphur from residual fuel oil, etc.).

For the EPS the most important thing is total and rational use of natural recourses and production of cheaper electric energy so the company can be concurrent on market.

5. CONCLUSION

One of the lignite characteristics in Kolubara coal basin is express stratification inside the deposit, particularly in western part of basin, and in some of the layers variations in quality occurred. In order to provide continuity for power plan supply it is important to implement a system which will enable detailed planning and coal exploitation with control of quality, and at the same time a techno-economy effect of exploitation would rise during utilisation of low quality parts of deposit.

With CQMS introduction detailed planning, analysis and simulation as well as control and tracking of coal quality during excavation, transportation, deposition and train loading will be provided, which will ensure sending of goods with known quality.

System for coal quality management has multiple benefits. Open cast mines would work rationally without uncontrolled spreading of raw material from natural recourse and with possibility of expansion of coal production as raw material. In power plants this system provides and creates regular conditions for coal combustion with appropriate quality thereby producing cheaper electric energy, and impact on environment will be significantly reduced.

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DOI: 10.7251/BMC170701359V

APPLICATION OF THE ISHIKAWA AND RCA METHODS FOR DETERMINING THE CAUSE OF ACCIDENTS AT WORK IN STEERING MINING MACHINES

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ABSTRACT

Contemporary business conditions in the mining industry require high levels of security or risk assessment and avoidance of unexpected occurrences. The consequences of an unexpected occurrence often manifest as injury of workers, termination of the technological processes in production and/or damage to material and natural resources. Steering mining machines in open pit mines is an activity in which a large number of risks and hazards at work occur and accidents are very common. In order to have a preventive effect on raising the level of safety, reducing the risk at work and protecting workers from injury and health damage as well as reducing damage to the means of labor (mining equipment) it is necessary to know the causes of accidents. This paper presents the implementation of the Ishikawa and RCA methods for the identification of all the causes of accidents at work that may lead to negative consequences and extrication of the underlying cause, towards which corrective activities are directed.

Key words: accident at work,risk, cause,ISHIKAWA method, RCA method

1. INTRODUCTION

A large number of potential hazards and harmful effects to the health and safety of workers, as well as the likelihood of occurrence of incidents in radar, are very high. That is why mining in which a number of accidents occur, often with serious and fatal consequences, pose a challenge for occupational safety, mining and other engineering profiles in order to do as much as possible in terms of preventing accidents at work.Mines are very complex systems that include a whole range of activities (such as extraction of ore, crushing, grinding, flotation, melting and refining, numerous engineering processes, maintenance of heavy machinery and equipment, etc.) and various issues related to general safety of employees (potential for natural disasters, transport risks, medical evacuation, availability of local health facilities, etc.). Because of this, the number of potential hazards and harmful effects to the health and safety of employees, as well as the likelihood of occurrence of incidents are very high [5]. In addition to the basic sources of danger, in the exploitation of mineral resources,

there are also special hazards that can lead to large-scale mining disasters (water breaks, fires, explosions of gases and dust, large scale erosion caused by mountain bumps, etc.).

According to the analysis of the number of injuries in the Republic of Srpska mines conducted by the Government of the Republic of Srpska and the Republic Administration for Inspection Activities in the period from 2005 to 2009, it was found that the greatest number of injuries occurred to the workers directly engaged, that is, managers of mining machinery, cave diggers and maintenance workers (locksmiths, electricians, mechanics, welders) [1]. The most common causes of injuries are complex conditions of exploitation, insufficient training and training of workers - miners for safe work, and disregard of basic and special measures of protection at work. The greatest number of recorded injuries is mainly due to human factors, disregard of laws, regulations, as well as instructions detailing the performance of mining operations, handling and maintenance of mining machines, plants and devices, as well as measures of protection at work for the performance of all mines operations. The use of heavy mining machinery in unfavorable and difficult working conditions and insufficient training of operators are the most common causes of mechanical and thermal injuries, and unfortunately sometimes fatal cases.

The most common dangers when working with mining machinery include:

- disturbance of transverse or longitudinal stability (static-inclination of the terrain, dynamic-inclination of the terrain, characteristics of micro-relief, presence of obstacles),
- wrong handling,
- poor visibility,
- other harmful effects, unfavorable atmospheric influences: rain, sun, snow, cold, noise and vibration, harmful vapors from oil and gasoline, dust ...

It is of major importance to determine the dominant causes of accidents in order to undertake preventive and corrective actions.The occupational safety and health system should design preventive actions that will prevent the occurrence of accidents at work. Primarily, accidents with possible employee injuries should be prevented. By applying the occupational safety and health system, corrective measures are taken to mitigate the consequences of an accident. In order to successfully prevent accidents, it is not enough to study only the events that result in injury, but also the events that pose a potential threat to the work process.

Accidents at work occur as a result of the three main etiological factors: the source, the cause and the man. In some accidents man can play the role in the effect of one of the two preceding factors.In practice, the cause is often identified with the source of the accident. The origin of the accident is in fact the state, whereas the cause is a phenomenon which in the present state triggers an action that results in an accident.Classification of the causes of accidents used in literature [3] is as follows:

- causes dominated by human (subjective) factor, which refers to the personality of workers,
- objective reasons relating to the degree and duration of objective danger which is connected to the work environment, workplace or social environment.

2. METHODS FOR DETERMINING THE CAUSE OF ACCIDENTS AT WORK WHENHANDLING MINING MECHANIZATION

Engineering methods are used to improve the quality, reliability and safety of the process by analyzing the causes and consequences of errors in the working environment system.Many engineering methods, tools, and techniques are used for the identification of risk factors and the causes of disorders in security systems, that is, the cause of accidents at work, Figure 1.

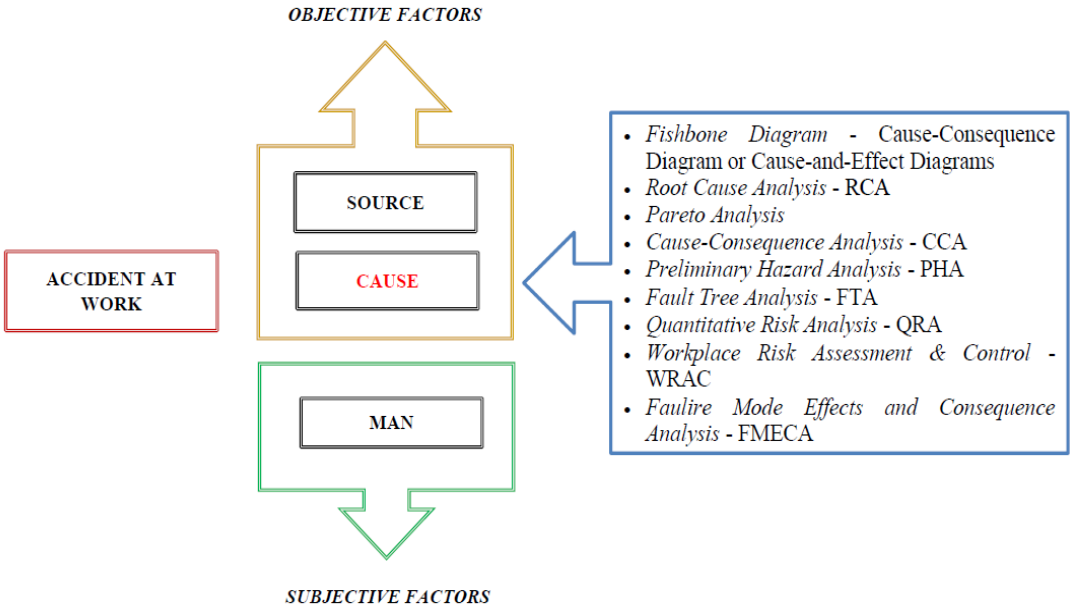


Figure 1 Methods for determining the cause of accidents at work[8]

2.1. The application of the ISHIKAWA method for determining the cause of accidents at work

The Ishikawa method belongs to the cause-effect group, and is named after the author Kaoru Ishikawa and is often referred to as a "fishbone diagram" because of its appearance. It is a tool that helps in the identification, sorting and displaying possible causes of a specific problem or quality characteristics of the system. A graphic diagram shows the relation between a given output, and all the factors that influence this output (result)[4]. By applying this method all possible causes are investigated that may lead to certain consequences and thereby the main one is singled out. The Ishikawa diagram is a tool that directs corrective action and represents a good basis for improving the quality of processes and preventive action.Sequence of activities for the implementation of the Ishikawa method is as follows[8]:

- defining problem/consequences/errors,
- selecting structure of maximum seven causes (e.g. people, equipment, materials, environment, methods, measurability, data system, etc.),
- developing diagrams and analysis of all causes,
- evaluating the level of impact including the proposed sequence of measures fortheir elimination.

Advantages of this method are reflected in the following [9]:

- Determining the cause of the problem using a structured approach.
- Active participation of team members and use of their knowledge related tothe considered process.
- Presentation of cause-effect relationships in a clear and easily readable way.
- Integration of the knowledge and skills of the team members about the observed processes.
- Identification of areas in which data should be collected for further study.

Once designed, the diagram can become a "living tool" which is further improved by entering the details of newly acquired knowledge and experience.The Ishikawa method is applied to accident situation which resulted in a minor worker's injury that occurred while controlling the cooling system of the Kamatsu PC 450 LC hydraulic excavator. Due to slipping on the working platform and a number of circumstances that can be characterized as a general lack of caution at work, the worker fell down and got minor back injury after slightly being caught by the tracks, Figures 2 and 3.



Figure 2The place where the injury occurred - Kamatsu PC 450 LC hydraulic excavator [8]

In the Ishikawa diagram that presents this accident situation the main categories of possible causes include: work organization, work-machine, worker-hischaracter traits and work environment. The main categories are broken down into lower categories. After the analysis of possible causes, the main cause-lack of general caution in work - is defined.

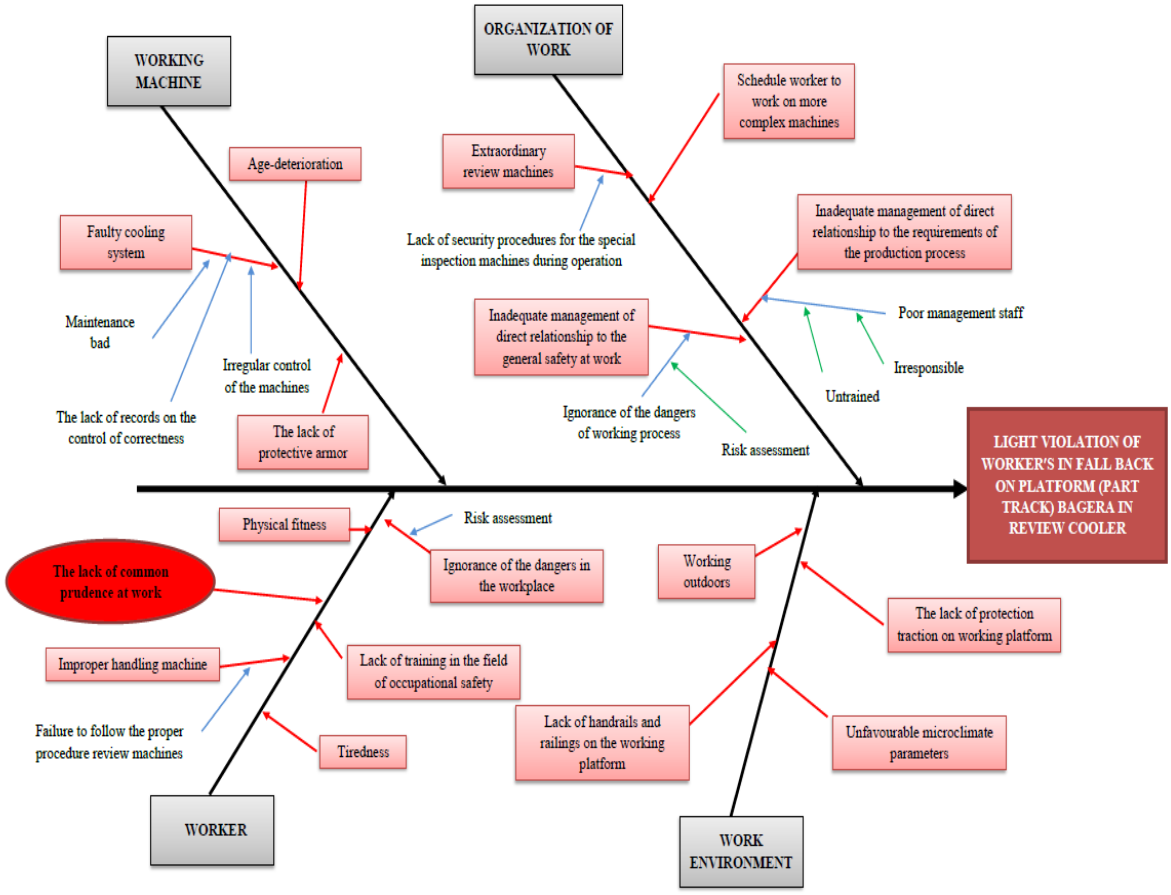


Figure 3 The Ishikawa diagram - minor worker's injury [8]

2.2. The application of the RCA method for determining the cause of incident-accident at work

The Root Cause Analysis - RCA was primarily developed and designed to be an aid to engineers at improving the quality, reliability and safety of technical systems and their components, within a proactive approach. RCA is a methodology sequence of logical steps or stages of the process that will lead to the determinationand elimination of the underlying causes of the accident. Advantages of this method include [7]:

- Improvement of reliability, availability, quality and safety,
- Permanent elimination of potential causes of accidents,
- Placing emphasis on a proactive approach,
- Minimized costs,
- The creation of teams and the exchange of ideas.

This method is specific in that it is applied when an incident-accident has already occurred. The main advantage of the method, in terms of safety at work, is that it examines the risks that have occurred and shows the relationship between the cause and effect.

The sample analysis (RCA) monitors the cause-effect relationship from the end result of the accident-incident back to the source of the accident-incident, i.e. the cause of events. The goal is to determine what happened, why it happened and the most important is to determine what should be done in order to reduce the possibility of an accident occurring again, Table 1.

Table 1 Questions that help define the problem (incident, accident) and gather data [2]

Category	Questions
What	• What happened? • What are the symptoms? • What is the complaint? • What went wrong? • What is the undesirable event or behavior?
When	• When did it occur: what date and what time? • During what phase of the production process?
Where	• What plant? • Where did it happen? • What process? • What production stream? • What equipment?
How	• Whatwas the situation like before the incident? • What happened during the incident? • How is the situation after the incident? • What is the normal operating condition? • Is there any injury, shutdown, error, or damage? • How frequent is the problem? • How many other processes, pieces of equipment or items are affected by this incident?

The RCA method was applied to the incident situation caused by the reverse driving of the CAT 777 dumper. The CAT 777/6 dumpercabin and the CAT 777/8 dumperbucketcame into contact at the end of the effective working hours when the lineof 5 dumpers was temporarily stopped at the landfill, and the operators wrote a daily report. For unspecified reasons, the operator of the CAT 777/6dumper moved backwards, after visiting two dumpers, on the approach road of the landfill, did not notice the third dumper (CAT 777/8) in the right mirror. There were two dumper contacts, the rear right end of the bucketin the left railing on the platform next to the left door of the cabin, and in the rear left-hand door of the CAT 777/8 dumpercab causing damage to them. The driver was not wearing a safety belt, although he was instructed to use it. Also, when reversing, he did not use the acoustic signaling as one of the main rules of safe backward movement, although he passed training for the demo driver and was authorized to drive at the landfillby the manager, Figure 4.

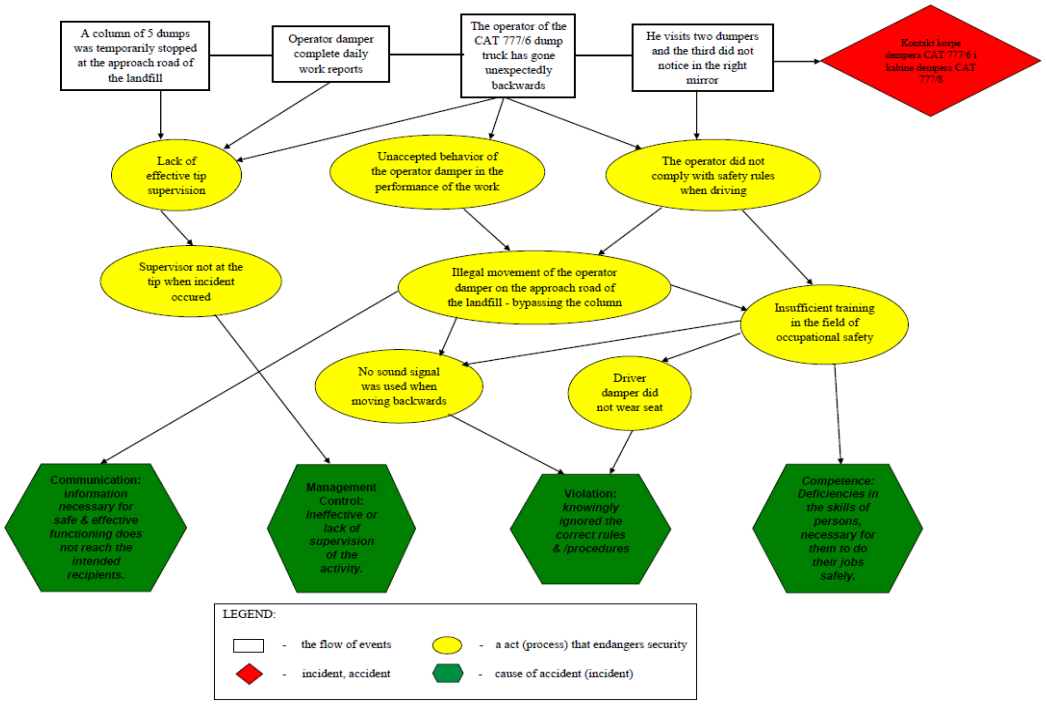


Figure 4 The RCA analysis of the incident which resulted in damage to mining machinery

The RCA analysis of the accident situation defined four categories of causes: communication, management control, violation of safety procedures and competence.This incident situation ended only with the damage of equipment (mining machinery), and without any consequences affecting handlers, Figure 5.



Figure 5 Data related to the damage on the CAT dumper

3. CONCLUSION

Quality management tools have become indispensable in dealing with problems related to all areas of human activity, even in mining. Working conditions are safer if the accidents at work are reduced to a minimum.Risk assessment and discovering the causes of accidents enables the control of effectiveness of the security, safety and health at work and management of the system with the least number of failures, i.e. accidents with workers' injuries.

The Ishikawa and RCA methods are applied for the detection and analysis of the root causes of these problems and unwanted accidents in the organization in order to define preventive and corrective measures for the improvement of processes with increased risk.By analyzing the underlying causes through three stages, data are collected, their analysis performed and the identified causes are finally validated.The specificity of the RCA method that is applied when an incident-accident has already occurred should be exploited in the handling of mining machinery in a way to design handling procedures which will reduce the numberof accidents inflicting injuries to workers.

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DOI: 10.7251/BMC170701367V

APPLICATION OF THE SYSTEM FOR DUST CONTROL ON THE HAULAGE ROADS ON EXAMPLE OF OPEN PIT "BUVAČ"

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ABSTRACT

Dust control in mines with surface exploitation mostly refers to the periodically sprinkling of haulage routes, landfills and stockpiles with water. The truck transport systems at the open pits are followed by generation of large amount of dust due to dampers and large trucks movement on haulage roads. The negative impact of dust is reflected in environmental pollution, human health impacts, traffic safety due to reduced visibility, as well as the length of equipment's service life. Due to the extreme dust generation from haulage roads in dry weather, dust control in such systems is most often carried out using water trucks. The use of water dumpers and water trucks for haulage roads wetting in addition to uneven sprinkling and decelerating of other transport units in the system, causes high operating costs. Therefore, the idea was to install and run a test system on a section of the permanent haulage road at the open pit "Buvač" with sprinklers installed on one side of the road and to examine the functionality and cost effectiveness of this kind of dust control. This paper deals with issues related to the hydraulic calculation of the sprinkling system, a review of equipment selection criteria, and issue analysis related to the functionality of the sprinklers and the water quality for this application. Technical solutions, guidelines and recommendations for the further development of this system at the open pit are proposed.

Key words: open pit, haulage routes, dust control, sprinkling system, sprinklers

1. INTRODUCTION

Dust control can be realized in different ways depending on the size and the type of the dust particles. For haul road dust the best method of dust control depends on the type of road aggregate. Haul road aggregate on the surface mines are usually gravels. On unpaved haul roads, large trucks go in and out of mines multiple times a day creating large amounts of airborne dust. The most common method of haul road dust control is surface wetting with plain water, but others include adding hygroscopic salts, surfactants, soil cements, bitumen, and films (polymers) to the road surface. Dust control by the water application to the road surface ensures an optimal approach to dust problems [1]. Due to the extreme dust from haul roads in dry weather, dust control in such case is most often carried out using water trucks which causes uneven sprinkling and decelerating of other transport units.

For both small and large areas of haul roads the sprinkler irrigation has been widely introduced in individual schemes. The technique includes a complete irrigation system with a pump, distribution pipes and laterals on which the sprinklers are placed. The system has high water efficiency, is easy to install, and the equipment is readily available on the market. However, high investment costs, as well as high (electricity, power consumption) costs for the operation of the pressure pumps, have been a major constraint and are often a reason why the implementation of the technology failed or has been abandoned. The most important is that the stationary sprinkler systems can reduce costs compared to mobile equipment [2]. Figure 2 illustrates an example of wetting with the sprinkler sprayers set along the track path, on the side of road.

The paper presents the analysis of possible wetting of haul roads by the sprinkler system at the open pit Buvač, Omarska mine. The water from the wells and collectors would be used for sprinkler supply.

2. CHARACTERISTICS OF DEPOSIT BUVAČ

Metal-genetic iron ore area "Ljubija" is situated in the North East of the Republic of Srpska, Bosnia and Herzegovina, in the valleys of rivers Sana, Una and Gomjenica, between mountains Kozara and Grmeč.

The mining iron ore "Ljubija" with established reserves of about 450,000,000 tons represents the largest deposit of iron ore in the territory of the former Yugoslavia and the region. The production of iron ore was begun 1916.

Deposit Buvač, along with the deposit "Jezero", belongs to the Omarska basin, within the north-eastern part of the Sana Paleozoic, or the Ljubia's Metalogen area. Because of their specificity, both of the deposits can practically be treated as separate deposits. Omarska mine is located between the Prijedor and Banja Luka, and far 20 km from Prijedor and mine "Ljubija". An explotation field of Omarska with a layout of the open pits is presented on Figure 1.

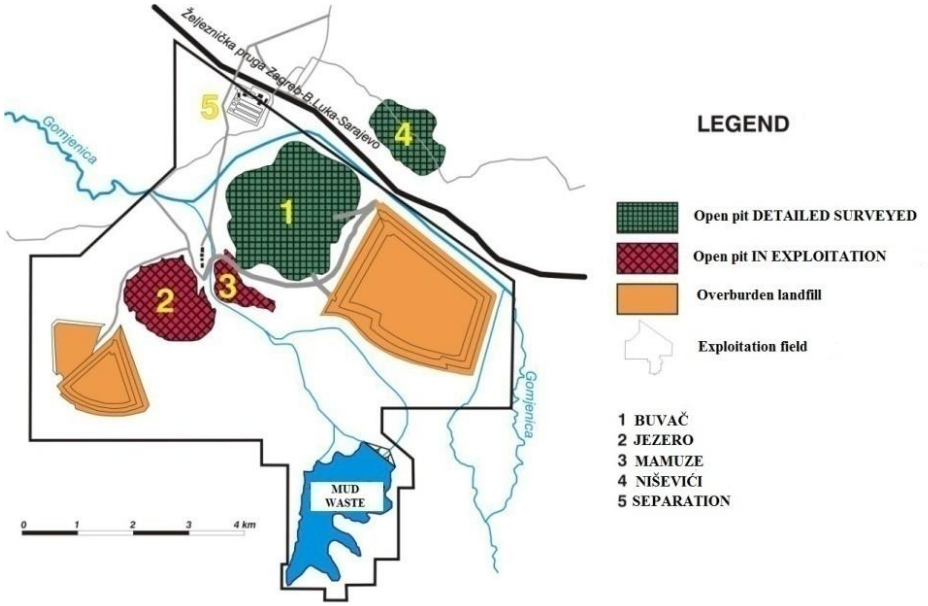


Figure 1 Explotation field of the Omarska mine [3]

The area of the deposit belongs to a moderate continental climate characterized by sudden rise of temperatures in spring, winter low precipitation, moderate cold winters, hot air and frequent cold air congestion which can largely affect the quality of haul roads.

At the open pit "Buvač" about 5000 m of transport roads are currently used, which are located either on the final slopes or out from the contour of the pit (permanent roads). In addition to the permanent roads, temporary roads within the pit and landfills are also watering. The length of the temporary roads is about 5000 m, which means that the length of the haulage road that has to be wetted is about 10000 m [5]. This requires the daily work of two water trucks. In order to install a sprinkling system on a permanent haul, the need for using the two tanks would cease.

3. EQUIPMENT FOR SPRINKLER SYSTEM

Basic elements of sprinkler system are sprinklers, pipes and the pumps.

Two general types of sprinklers used are impact or gear-drive sprinklers which produce moving streams of water and spray nozzles with water discharge at all points in the wetted pattern at all times. In general, water application rates are higher for spray nozzles than for impact or gear drive. Because the water application rate from spray heads is high, they should be operated for a relatively short period of time during each irrigation event to supply the desired depth of water [6]. HDPE pipe of standard dimension rate are connected with fittings or welding. For uniform application of water is essential proper irrigation system pressure. A Pump could be selected if the pump characteristics satisfy the desired flow and pressure.



Figure 2 Sprinkler system on haul road [4]

3.1. Dimensioning of the running system

Distance between the sprinklers

The nominal range of sprinklers cannot be considered as useful as it is obtained in ideal conditions, while for application in the real conditions the influence of wind must (need to) be considered (taken into account) [7].

Based on this rule, the distance between sprinklers expressed in feet is equal to the nominal (working) pressure of sprinklers expressed in PSI.

$$1 \text{ PSI} = 0.068948 \text{ bar}; \; 1\text{feet} = 0.3048 \text{ m}$$

Based on previous for pressure of 1 bar the optimal distance between sprinklers is about 4.42 m.

Optimal range of sprinklers is determined on the basis of designed road width. The rotation angle of sprinkler rotor is about 180°, and only exceptionally more. The optimal sprinkler selects from manufacturer's catalog.

As example in Table 1, the characteristics of sector impact sprinkler *GROUNER* TYPE 342 is listed, which has the option of adjusting the rotation angle of sprinkler rotor.

Table 1 Characteristics of sprinkler GROUNER TYPE 342

Diameter of injector, mm	Working pressure, bar	Wetting radius, m	Water consumption m^3/h
7,5	2 - 5	14,5 - 19	1,9 – 3,0

The distance between the sprinklers for given pressure P (bar) is:

$$Lstr = P \times 4.42, \text{ m}$$

The distance between sprinklers based on the general rule is 50 to 60% of wetting diameter, is

$$Lgp = (0.5 \div 0.6) \times D, \text{ m}$$

Where:

P - expected sprinkler pressure, bar,
D - wetting diameter, m.

As the optimal distance between the sprinklers, the average of two results can be taken.

Since the dumpers disperse the water by the wheels so that the higher value of the sprinklers distance does not leave the dry spots.

3.2. Hydraulic calculation

The hydraulic calculation is a standard calculation and it includes the summing of local and line losses for the accepted diameter of pipeline, and the notes to be taken into account are given below.

The pressure drop (loss) on sprinklers is equal to the nominal (catalog, operating) pressure of sprinklers. Regardless of the number of sprinklers in the line, the pressure drop is taken only for one sprinkler and this one with the upward working pressure [8]. If all the sprinklers are the same, the drop in sprinkler pressure is identical.

The total head (pressure) that the pump has to overcome is determined for the branch that has the greatest losses in order that the farthest sprinklers have optimum working pressure. If the system has only one pipeline, the pressure drop for one branch (lateral) is calculated, i.e. the pressure drop on the last sprinkler in the line, will be relevant to the pump selection.

3.3. Sprinkler water consumption

When calculating water consumption it is necessary to determine which number of sprinklers will be run simultaneously. There is a possibility to mount the electric valves on each sprinkler or on a branch with a number of sprinklers, so determine the number of simultaneously active sprayers.

Water consumption if all sprinklers are run simultaneously is calculated:

$$Qn = n \times Q\,(p),\ m^3 / h$$

Where are:
- *n* - number of sprinklers which are activated simultaneously,
- *Q* (p) - water consumption of one sprinkler (m^3/h) depending on the water pressure on it (bar).

3.4. Pump capacity

The pump must provide the following capacity:

$$Qp = Qn,\ m^3/h$$

Where is:
- *Qn* - water consumption if all sprinklers are run simultaneously, m^3/h.

Schematic view of sprinkler system for haul road dust suppression at the open pit Buvač is shown on the Figure 3.

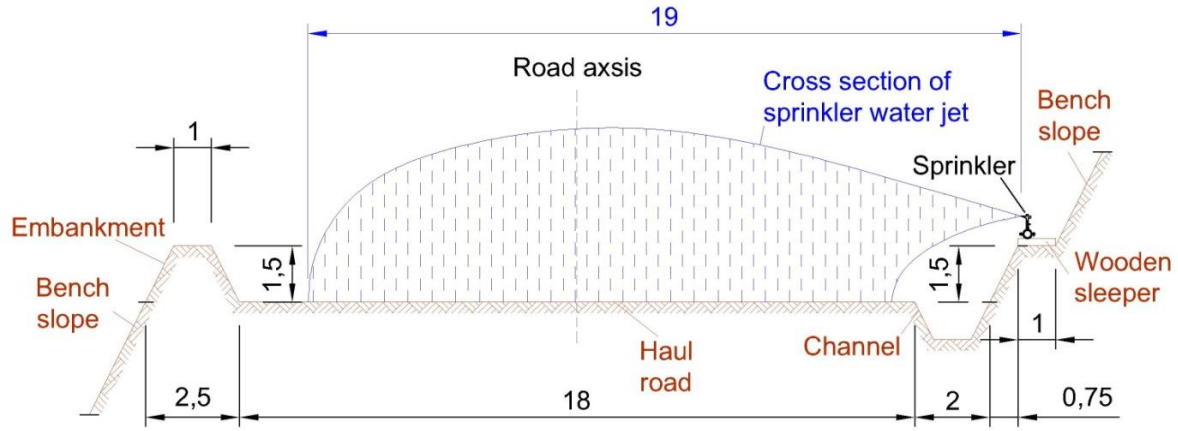


Figure 3. Schematic view of road wetting by sprinklers [9]

3.5. Parameters adopted for application at open pit Buvač

For application at open pit Buvač sprinklers' number was 21. For adopted sprinklers' working (running) pressure of 5 bar overall water consumption was 63 m^3/h. Overall head that a pump needed to overcome was 12 bar (see data in Table 1).

Based on previous data the well pump *Lowara Z660* was selected with electric motor rated power of 22 kW.

This pump does not have an inner cooling system, therefore it must be installed into a pipe which diameter is 2" larger than the pump diameter.

4. CONCLUSION/RECOMMENDATION

It is necessary to choose sprinklers with a larger opening so that the droplets of the jet are as big as possible, thus reducing the influence of wind on the wetting pattern. The water purity for this application is extremely (very) important. The fine clay particles from the water lead to blocking the rotor, which requires premature services, i.e. disassembly and cleaning of the rotating mechanism.

By installing electric valves and the system automation it is possible to significantly reduce the pump size and pipeline diameter since less water consumption is needed simultaneously for running of the system - fewer number of sprinklers run simultaneously.

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DOI: 10.7251/BMC170701375Z

SYSTEMATIZATION OF INJURIES AT WORK IN "EFT RUDNIK I TERMOELEKTRANA STANARI"

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ABSTRACT

Successful exploitation of mineral raw materials in addition to modern technological processes depends on quality technical protection. Injuries at work and professional diseases bring with them big expenses, both for the family injured and for the employer and society as a whole. The aim of this job was to get for a period of work "EFT-Rudnik i Termoelektrana Stanari" from 2010. to 2016. collect data on injuries, perform their categorization, statistical processing and analysis by all categories, on the basis of which measures will be proposed for more efficient reduction of the number of these injuries by applying appropriate measures of protection in technology, technique and organization of work.

Key words: injuries at work, types of injuries, injury analysis, protection measures.

1. INTRODUCTION

Work in mining is always related to certain dangers, which can be eliminated or reduced to the minimum possible measures by implementing technical protection measures. The aim of technical protection is to ensure safe working conditions for the technological process as a whole, with adequate utilization of facilities and equipment.

In practice, the causes of injuries are often identified with sources of injuries. The source of the injury is a condition, and its cause is a appearance that in the present situation causes an injury action. For our mining practice, where injuries appear under the influence of natural and technological factors, as well as the behavior of the worker himself, the causes of their appear can be divided into:

- natural, which depends on montangeological conditions,
- technical - technological, which depends on the applied technical and technology,
- subjective, which depends on the man [1].

Our and worldwide research confirms that over 80% of the causes of injuries, makes a human factor. In our mining practice, as the subjective cause of injury is lack of attention (carelessness) [2].

Work discipline must be rigorously set up, not only for productivity, but also for work safety, which requires a higher educated and professional level of employees, and therefore also understanding and attitude towards security and technical protection.

On the "Stanari" mine, is open pit excavation of coal and it is carried out with two systems of exploitation:

- continuous excavating system (CES) with the application of the excavator bucket wheel C-700, traveling belt conveyor, the appropriate number of conveyor belts, and stacker for waste dump to inside and outside dump,
- discontinuous system with the use of hydraulic shovel excavator of different bucket volumes and dumpers trucks of various load capacity for the transport of spoil and coal [3].

Technological processes of extraction, overburden conveyance and waste damp, excavation coal and transport to primary crusher, crushing and transport of coal by a conveyor belts to depot of a thermal power plant are carried out under unfavorable working conditions such as:

- work in the open, where the adverse impact of atmospheric factors: precipitation, wind, temperature changes and the etc,
- diverse and robust mechanization,
- dusts,
- the presence of noise, vibration and the etc.

All this points to the fact shows that work on the open pit are carried in potentially hazardous environments. All of the above conditions and specificity at the same time can be regarded as a source of injury, which at one time can result in injury to the lighter, heavier or death consequences [4].

2. ANALYSIS OF INJURIES AT WORK

The narrow field of research in this paper is the Mine of lignite and Thermal Power Plant Stanari. Data from monthly and annual reports on injuries "EFT Rudnik i Termoelektrana Stanari" were used as data source, which were statistically processed and analyzed [5].

An overview of work injuries for the period from 2010 to 2016 is shown in the tables and diagrams 1-10. For easier processing all injuries are grouped in the following categories: work place (table and diagram 1), type of source (table and diagram 2), type of causes (table and diagram 3), parts of the body (table and diagram 4), qualifications (table and diagram 5), shifts (table and diagram 6), time intervals (table and diagram 7), working days (table and diagram 8), work experience (table and diagram 9), relation to easy, hard and deadly injuries (table and diagram 10). In order to better understand the injuries in certain categories, it was used in addition to tabular and graphical display of the number of injuries.

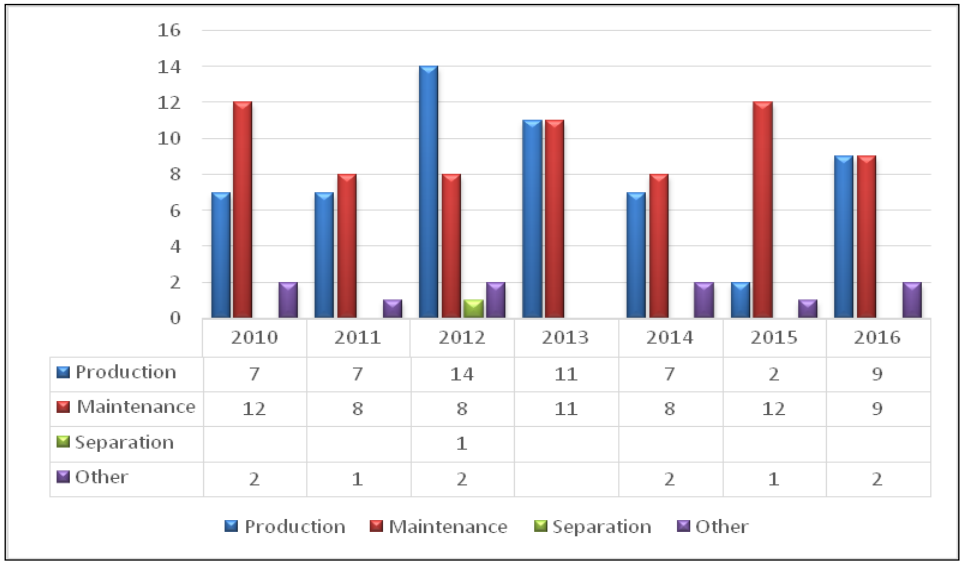


Diagram 1. Overview of injuries to the workplace of the worker

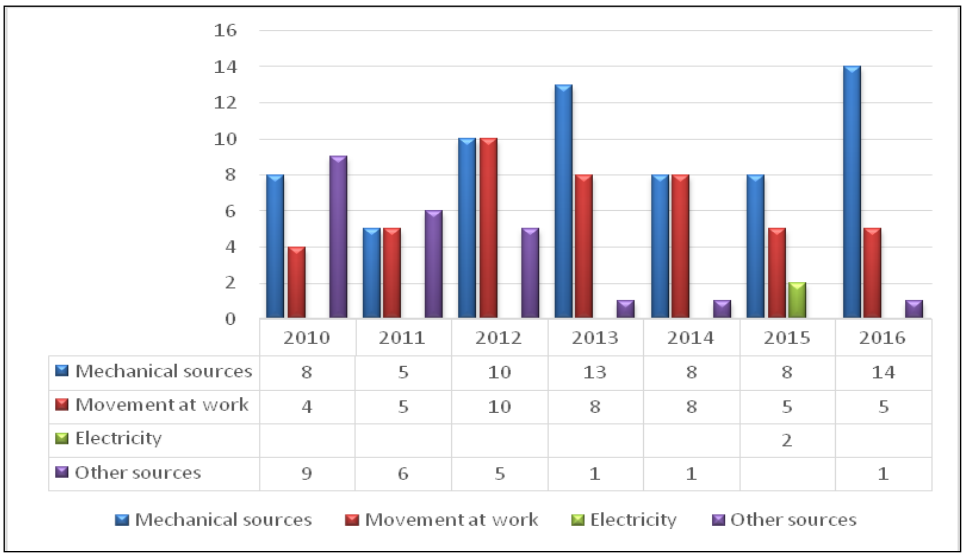


Diagram 2. Overview of injuries by type of source

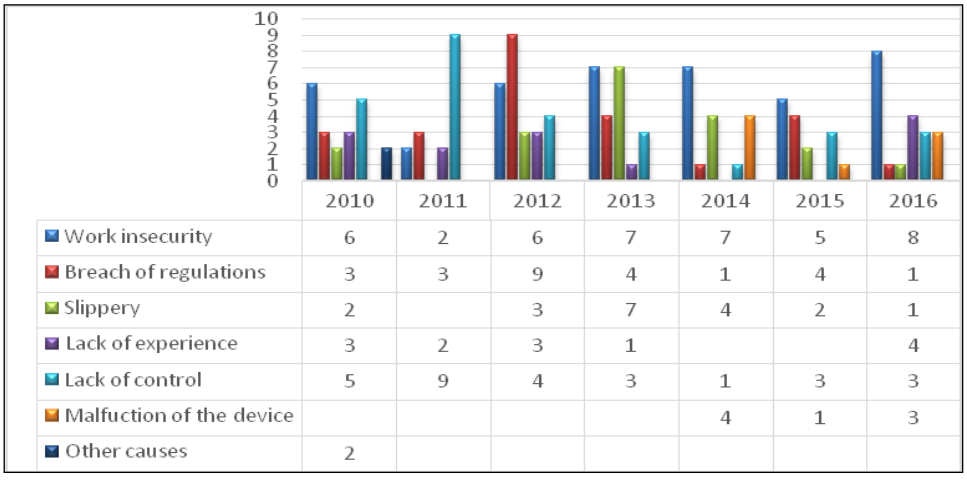
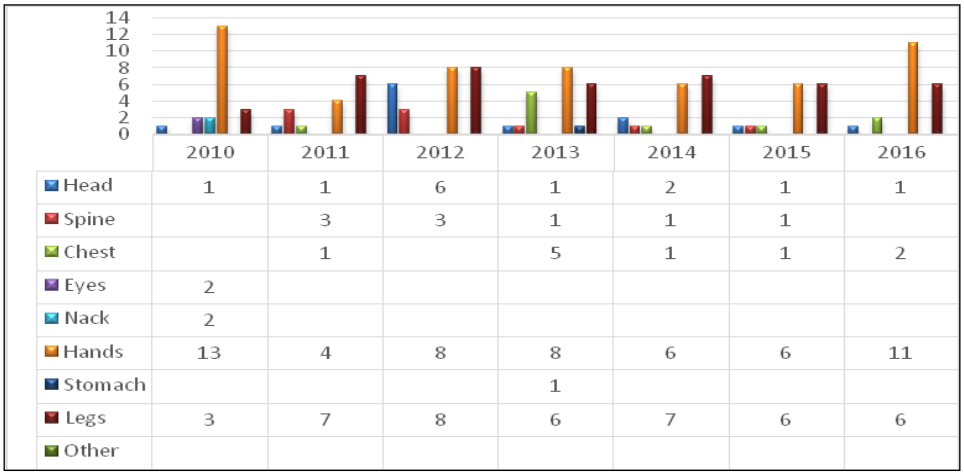
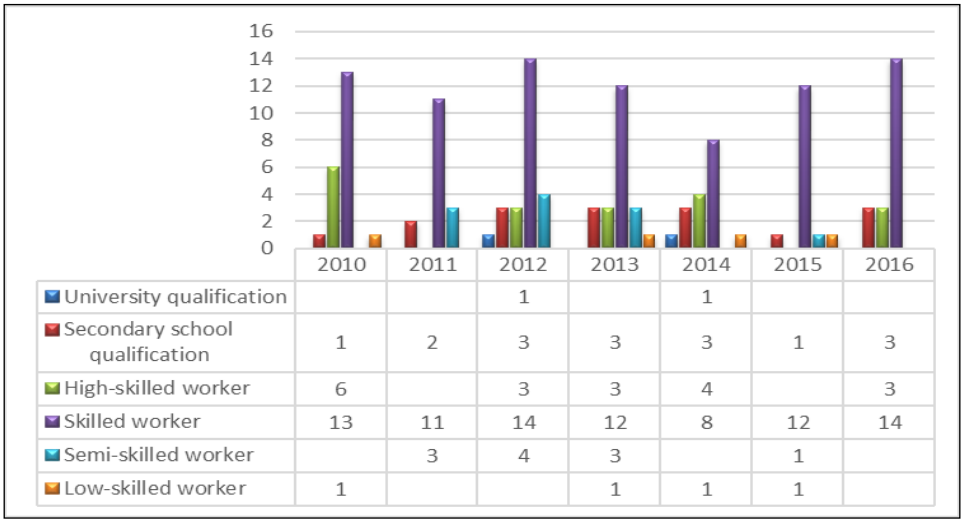


Diagram 3. Overview of injuries by type of cause



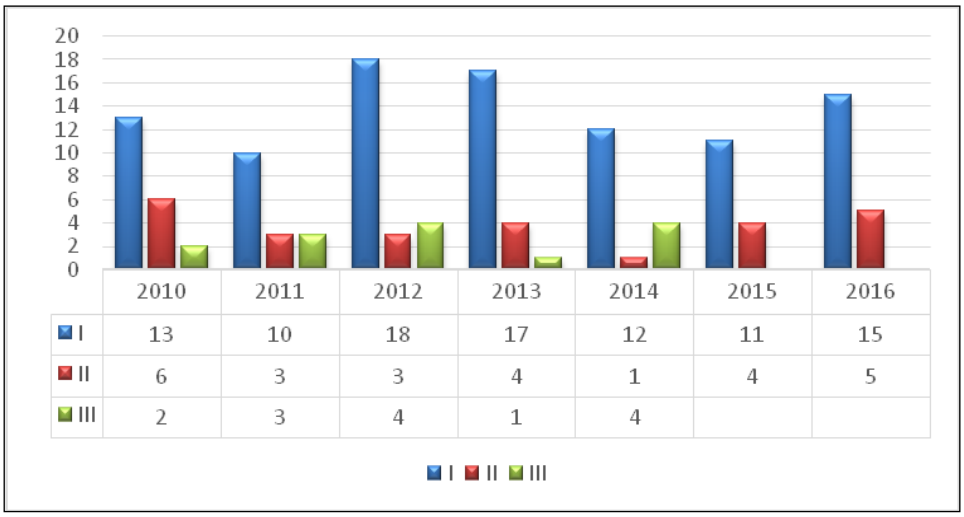
	2010	2011	2012	2013	2014	2015	2016
Head	1	1	6	1	2	1	1
Spine		3	3	1	1	1	
Chest		1		5	1	1	2
Eyes	2						
Nack	2						
Hands	13	4	8	8	6	6	11
Stomach				1			
Legs	3	7	8	6	7	6	6
Other							

Diagram 4. Overview of injuries by parts of the body



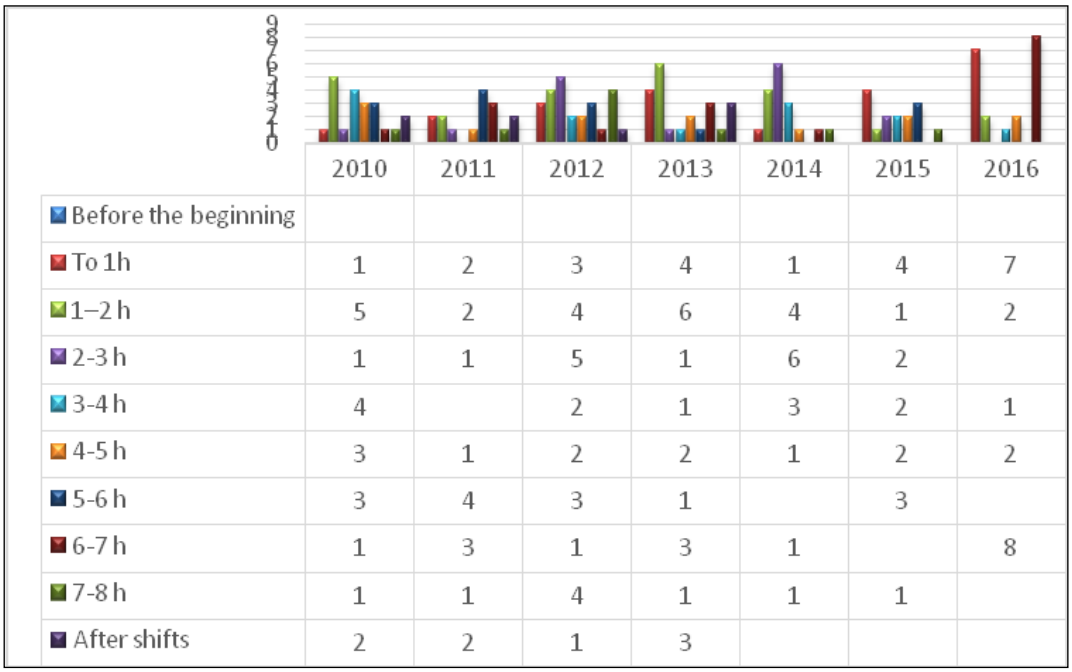
	2010	2011	2012	2013	2014	2015	2016
University qualification			1		1		
Secondary school qualification	1	2	3	3	3	1	3
High-skilled worker	6		3	3	4		3
Skilled worker	13	11	14	12	8	12	14
Semi-skilled worker		3	4	3		1	
Low-skilled worker	1			1	1	1	

Diagram 5. Overview of injuries according to the qualification structure of the employess



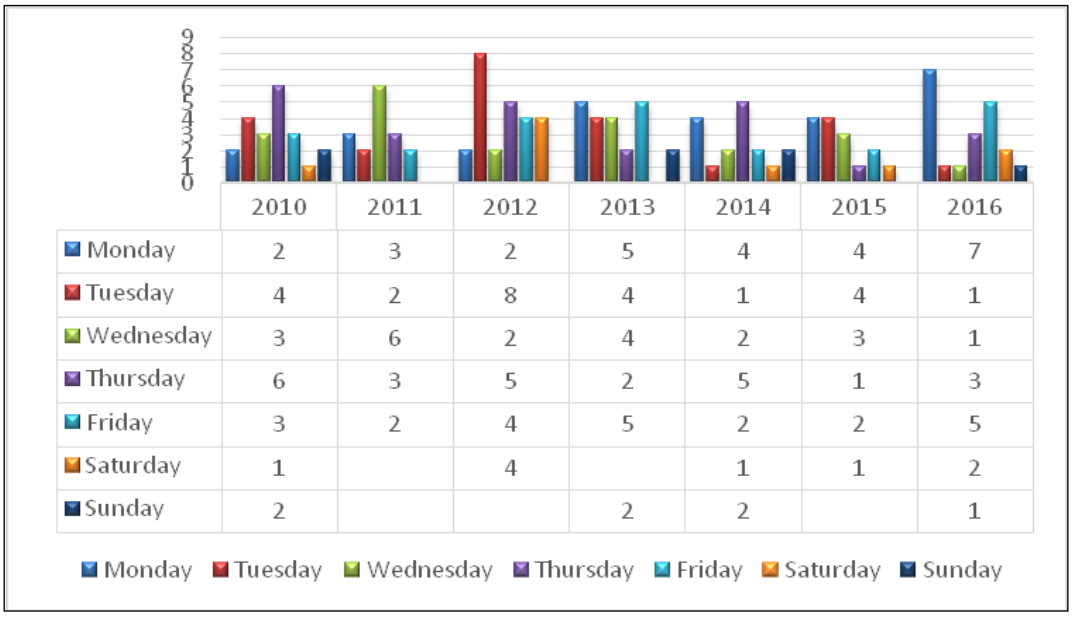
	2010	2011	2012	2013	2014	2015	2016
I	13	10	18	17	12	11	15
II	6	3	3	4	1	4	5
III	2	3	4	1	4		

Diagram 6. Overview of employees' shifts



	2010	2011	2012	2013	2014	2015	2016
Before the beginning							
To 1h	1	2	3	4	1	4	7
1–2 h	5	2	4	6	4	1	2
2-3 h	1	1	5	1	6	2	
3-4 h	4		2	1	3	2	1
4-5 h	3	1	2	2	1	2	2
5-6 h	3	4	3	1		3	
6-7 h	1	3	1	3	1		8
7-8 h	1	1	4	1	1	1	
After shifts	2	2	1	3			

Diagram 7. Injury overview according to the time interval of the employess



	2010	2011	2012	2013	2014	2015	2016
Monday	2	3	2	5	4	4	7
Tuesday	4	2	8	4	1	4	1
Wednesday	3	6	2	4	2	3	1
Thursday	6	3	5	2	5	1	3
Friday	3	2	4	5	2	2	5
Saturday	1		4		1	1	2
Sunday	2			2	2		1

Diagram 8. Injury overview on weekdays

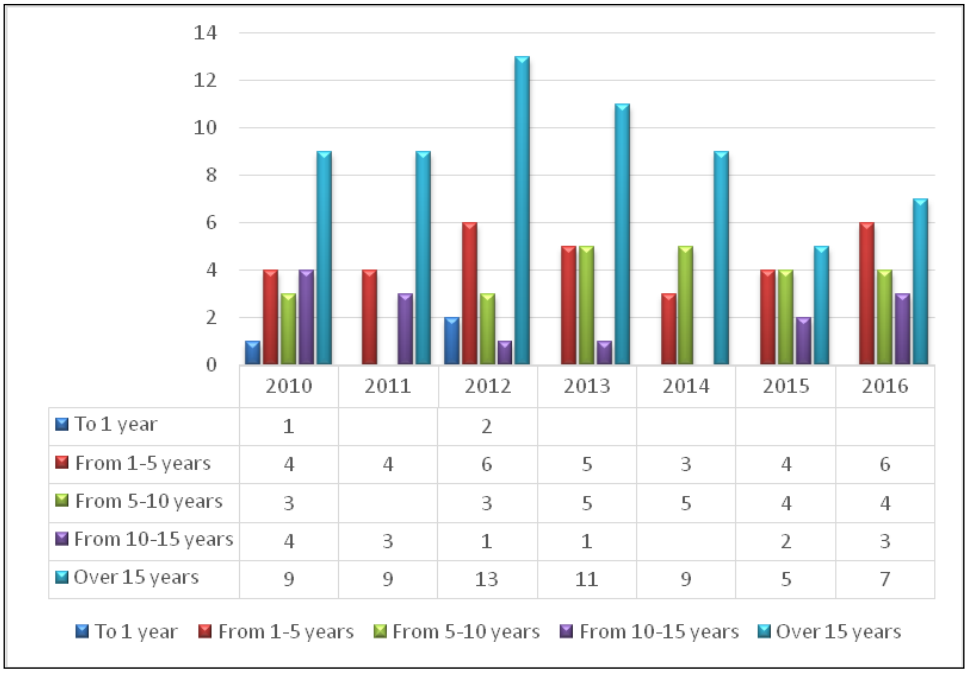


Diagram 9. Overview of injuries according to the work experience

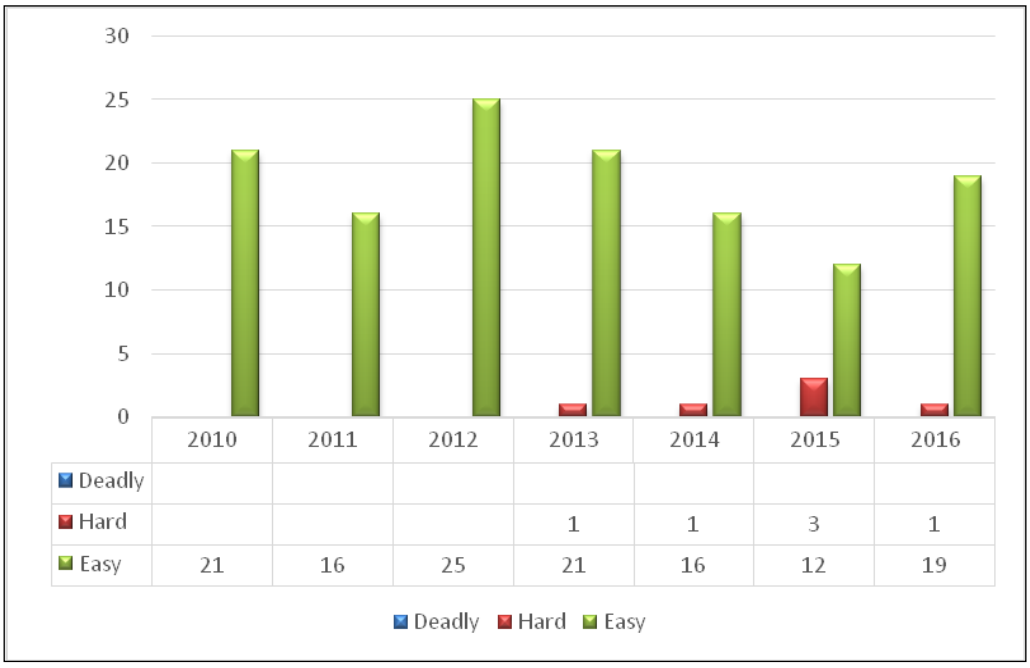


Diagram 10. Overview of injuries by weight

Analyzing the injuries for the period from 2010 to 2016, it is obvious that the greatest number of injuries is results from the careless and insecure way of maintenance work (50% of injuries), under which we consider maintenance of complete machinery and all devices necessary for exploitation, maintenance of infrastructure, and more. The reason for this large number of injuries is most often careless, lack of adequate work space and other.

All repairs to the CES and discontinuous equipment are carried out in their own direction, as well as designing, fabricating and assembling new three transporters with a rubber band over 2.5 km in length. This scope of work certainly affected the stated number of injuries.

Observed by type of source, also the highest number of injuries is the mechanical sources of injuries (48.5% of injuries).

Parts of the bodies that are most often exposed to injuries are extremities, in this case most hands were injured (41.2% of injuries). We can associate this fact with a large number of injuries to maintenance and carelessness of workers, because these jobs are mostly handled. Legs are most often hurt because of various skating.

Most often, they are hurting skilled workers (61.76% of injuries), because this category of workers performs more difficult jobs with a higher risk of injury, they are direct executors of the work and the most numerous are the qualification structure of the workers.

The highest number of injuries happened in the first shift (70.6% of injuries), as most employees are mostly employed and most important, and most of the activities take place in that shift.

Comparing the time intervals, it is evident that the highest number of injuries happened between 1 pm and 2 pm (33.8% of the injuries) of the work, that is, at the beginning of the shift.

Viewed on weekdays, the highest number of injuries happened on Monday (19.8% of injuries), or at the beginning of the week.

The highest number of injuries is among workers with over 15 years of work experience (46.3% of injuries). Employees to 5 years of work experience had the most injuries (25.7% of injuries), which is logical because these workers do not have enough work experience.

In 2016, the production in the "Stanari" mine was 2408000 t, and compared to 2015, when it was 1213000 t, it is on the rise. In the same period, the number of employees increased from 595 to 699 workers, and the number of injuries increased from 12 to 19 injuries, but 3 serious injuries were reduced to 1 injury.

Analyzing previous relations, it is evident that the number of injuries is decreasing, and consequently the general safety of employees is increased, that is, the level of safety and health of employees is improved, which is the result of systematic and consistent implementation of prescribed and designed protective measures, and consists of application preventive and current measures.

3. CONCLUSION

Exploitation open pit of lignite by its nature is responsible for the potential risks of accidents and individual injuries of employees.

This mine has shown that the conditions for reducing the number of injuries are reflected through regular information of employees on all occurrences and dangerous at work and their consequences, encouraging and stimulating employees, improving working conditions and organization of work as well as raising the technological and working discipline of employees.

Based on the obtained results of the survey, a comprehensive examination of the samples is enabled, sources, manner and consequences of injuries, in order to timely take adequate measures of protection, that is, safety and health of employees.

Analyzing the overall situation for a given period, it is evident that the number of injuries is decreasing and thus the overall safety of employees is increased, that is, the level of safety and health of employees is improved, which is the result of systematic and consistent implementation of the prescribed and planned protection measures, which consist of application of preventive and current measures.

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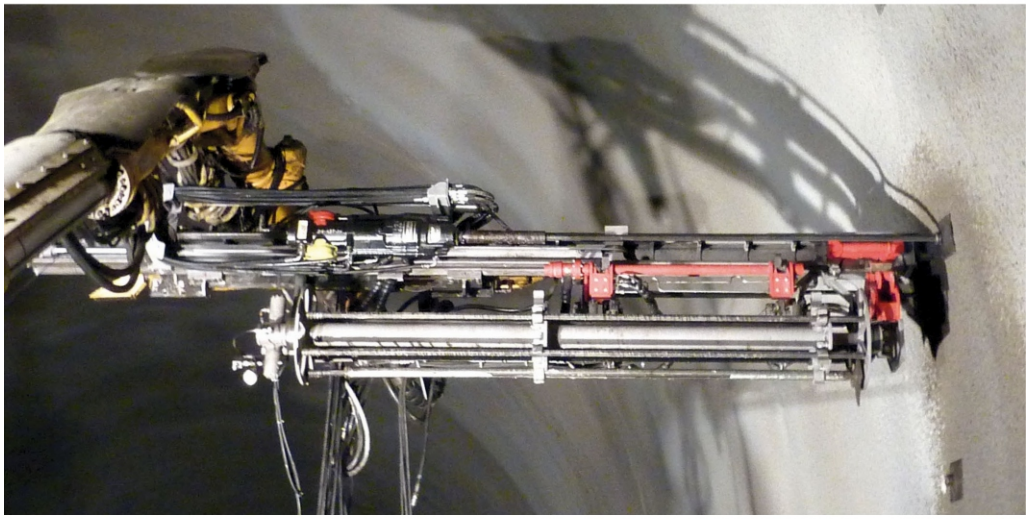
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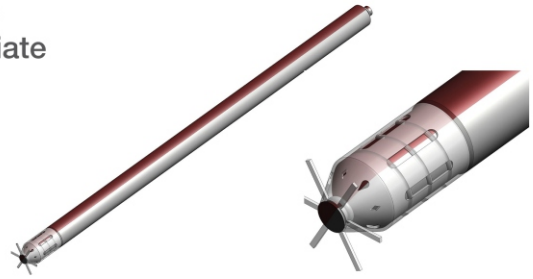
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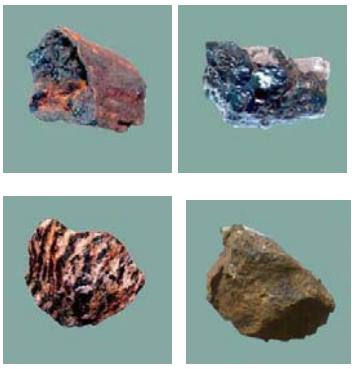


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"Omarska"
Eksploatacija je u toku

Tri izgrađena rudnika: "Ljubija", "Tomašica" i "Omarska" nalaze se jugozapadno, jugoistočno i istočno od grada Prijedora, na udaljenosti od 14 do 25 km. Rudonosno područje, odnosno Sansko-unski paleozoik, površine oko 1200 km2 zauzima prostor između Novog Grada- Prijedora-Bronzanog Majdana, Sanskog Mosta i Budimlić Japre.



Do 1992. godine Rudnici "Ljubija" su bili glavni snabdjevč željeznom rudom kapaciteta za proizvodnju čelika u Jugoslaviji.

U toku dosadašnjeg rada Rudnici željezne rude "Ljubija" Prijedor, su vlastitim znanjima razvili I proizvodnje: - prirodnih željeznih oksida - obrade metala eksplozijom - mašinskih dijelova i sklopova za strojeve i postrojenja - protektiranje pneumatika za teška-teretna i teretna vozila - sada "PROTECT" i- mjernih instrumenata (termometri, manometri u štednji likvidirano, ampermetri) i dr. .

Sa kompanijom ArcelorMittal osnovano je zajedničko preduzeće 2004. godine na paritetu 51:49 u korist ArcelorMittala i pokrenuta proizvodnja željezne rude koja je i danas aktivna. Dostignuti kapacitet proizvodnje iznosi do 2,2 mil. t/god. koncentrata željezne rude.

ArcelorMittal

ArcelorMittal Prijedor









ArcelorMittal Prijedor is a Joint Venture Company founded in August 2004 by ArcelorMittal Holdings and RŽR “Ljubija” a.d. Prijedor, ArcelorMittal Holdings being major shareholder with 51% share. Main activity is production of iron ores, developed in Buvač Surface Pit, in Omarska Mine. Employing 810 people we produce annually 1,7 mt to 2,1 mt. Till date we produced and dispatched 21,5 mt of concentrate. In production improvements, purchase of new equipment, development projects and local communities was invested over 100 million BAM.

Beside iron ores we also produce limestone fractions of various grain sizes, being used internally and externally for civil construction works. We also provide mechanical and electrical maintenance services and transportation of goods and people.

Main focus is on Health and Safety of our employees, long-term sustainable business and care for our communities. ArcelorMittal Prijedor is certified with ISO 9001,ISO 14001 and OHSAS 18001. We have received several awards for business excellence and corporate responsibility from RS Chamber of Commerce, City of Prijedor and International Community.

With our values: Quality, Sustainability and Leadership we are trying to transform tomorrow for our employees, their families, our communities and overall economy of RS and B&H.

The University of Banja Luka



The University of Banja Luka, founded on November 7, 1975 and comprising of 17 faculties today, is the leading higher education institution in the Republic of Srpska and the second largest one in the whole of Bosnia and Herzegovina.

At the moment, there are 811 fully employed professors and teaching assistants, 342 professors working part-time, and 175 visiting professors, the administration of the University totaling 559 employees.

So far, the University has produced 31,500 graduates, 700 hundred specialists, 1,150 MA/MSc and 600 PhD degree holders respectively, with 20,000 students currently enrolled at its study programmes.

University of Banja Luka is a member of the European University Association – EUA and is a signatory of the Magna Charta Universitatum. The University is a member of the International University Network for Academic and Research Cooperation, the General Assembly of Interuniversity Centre for Research and Cooperation with Eastern and South-Eastern Europe (CIRCEOS), UniAdrion universities network with headquarters in Ancona, Italy, EMUNI foundation with headquarters in Slovenia and the Agency for Francophone Universities (AUF).

Faculty of Mining



The Faculty of Mining was established in 2009. It grew out of the Mining Department at the Faculty of Technology, in which the first generation of students was enrolled in the academic year 1997/1998, into the Faculty of Mining Engineering as part of the University of Banjaluka with the headquarters in the city of Prijedor.

The first cycle studies are held at the study program of Mining Engineering while the second cycle studies are held at the study program of Mining and Geology Engineering.

The Mining Engineering study program is based on the three-stage model (4+1+3).



RUDARSKI INSTITUT d.o.o. BANJA LUKA

za projektno - konsultantsko - istraživačku djelatnost

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DJELATNOSTI:

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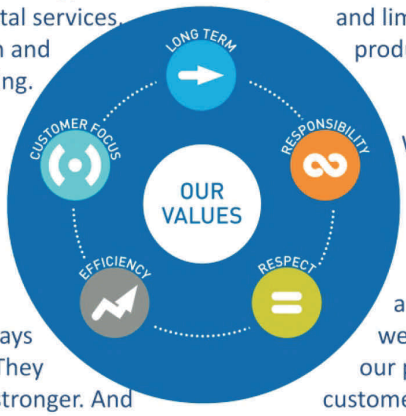
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B) DEPARTMENT OF ECOLOGY
C) LABORATORY: mineralogical Laboratory, PMS, laboratory for geomechanics, geophysics and chemical laboratories
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- RESEARCH IN GEOMENAGEMENT and evaluation of mineral resources,
- RESEARCH IN THE FIELD OF CHEMISTRY,
- RESEARCH IN THE FIELD OF RECULTIVATION OF DEGRADED AREAS
- RESEARCH IN THE FIELD OF APPLICATION OF COMPUTER TECHNIQUES IN MINING AND GEOLOGY.
- RESEARCH IN ECONOMIC GEOLOGY, GEOMENAGEMENT

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