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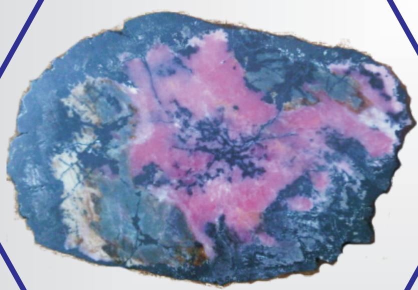

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Academic Ph.D. Milenko Ljubojev, Principal Reasearch Fellow Mining and Metallurgy Institute Bor
Full member of ESC
E-mail: milenko.ljubojev@irmbor.co.rs
Phone: +38130/454-109, 435-164

Editor

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Preprinting

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19210 Bor, Zeleni bulevar 35
E-mail: milenko.ljubojev@irmbor.co.rs
Phone: +38130/454-110

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Instituta za rudarstvo i metalurgiju Bor
Redovni član IAS
E-mail: milenko.ljubojev@irmbor.co.rs
Tel. 030/454-109, 435-164

Urednik

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Suzana Cvetković, teh.

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E-mail: milenko.ljubojev@irmbor.co.rs
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CONTENS

SADRŽAJ

MINING AND METALLURGY INSTITUTE BOR
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Slobodan Vujić, Nenad Radosavljević*, Aleksandar Milutinović**, Mihajlo Gigov**

APPLICATION OF THE UNMANNED AEROPHOTOGRAMMETRY IN MONITORING CONDITIONS AND CHANGES AT THE OPEN PIT MINES

Abstract

In the area of monitoring the spatial conditions and changes at the open pit mines, introduction of aerophotogrammetry with unmanned aerial vehicles significantly improves the effectivity and efficiency of the mining system monitoring in real and extended time. Since this is a relatively new technology, the paper presents our first experiences in the application of unmanned aerophotogrammetry, interesting also from the point of view of implementation into GIS systems.

Keywords: *open pit mine, aerophotogrammetry, unmanned aerial vehicle*

1 INTRODUCTION

In order to make the technological advance of aerophotogrammetry with the introduction of unmanned aerial crafts more noticeable, a reminder: photogrammetry as a method of visual measurement through which the shape and position of photographed object reconstructed is almost as old as the photography technology (mid XIX century). Photogrammetric recording can be terrestrial (from the ground), aero photogrammetric (from a plane or helicopter) and satellite. Aerophotogrammetry has developed in parallel with the aviation development, and today this is an almost unavoidable recording technique of the earth's surface. Recording is done from a plane with special measuring cameras, the result of which are the aero photogrammetric recordings, as the sum of visual information about a land surface that is the focus of recording. It is used for making the national maps, cadastre, cadastre-topographic and topography plans, theme maps, photomosaics, photoplans, orthophoto, etc. In order that the aero photogrammetric recordings can be used in stereo photogrammetric restitution of details, the recording is done with a suitable overlap of details (most commonly, 60% in length and 25% cross). Beside the plane of suitable flight characteristics and special cameras, for the classic recording of this type, a pilot, navigator and a cameraman are also hired.

The development of unmanned aerial crafts for civilian purposes is a novelty and is today on the rise, and presents a significant technological advancement in aerophotogrammetry. According to this technical-technological concept, the miniature robotic aircraft acts as a plane, pilot, navigator and a cameraman. The miniature highly sophisticated aircrafts, equipped with a propulsion and navigation systems, auto-pilot and a suitable camera, is an equivalent to the classic aero photogrammetric systems.

* *Mining Institute, Belgrade, Serbia, slobodan.vujic@ribeograd.ac.rs*
** *University of Belgrade, Faculty of Mining and Geology, Serbia, aleksandar.milutinovic@rgf.bg.ac.rs*

In the domain of measurement, monitorring of conditions and surface changes, aerophotogrammetry is of inexhaustible possibilities and satisfies a basic condition - measurement accuracy and significantly increases the surveillance and management efficiency in long-term and in real time. Considering the current events, development and application trends in this field, it is realistic to expect an increase of professional usage the unmanned aero photogrammetric technology in various areas. The rise of development and market competition for this technology is very noticeable at present. According to prognosis, this technology will have a great influence on employment, new job openings and profits, i.e. in the US alone during the first three years of development, this technology had created 70,000 new jobs and influenced the economy by 13.6 billion dollars.

2 HALLMARKS OF THE UNMANNED AEROPHOTOGRAMMETRIC TECHNOLOGY

The idea of unmanned aircrafts is not new. The idea originated from the time of the first air balloons, and was intended primarily for military and espionage purposes. This case repeats the historic rule that military activities and wars never bring joy and welfare, but they always influence innovation and birth of new technologies and discoveries, i.e. during the II World War, the radar, electronic and mechanical computer, rockets, and nuclear reactor were constructed and implemented, or development the global positioning satellite system (GPS) during the seventies. Without going too much into classification and categorization of modern unmanned aircrafts, two basic development directions will be mentioned in terms of control, one relates to the remote control and the second to the autonomous control.

Table 1 *Comparative display the basic performances of unmanned aircrafts of two prominent manufacturers*

Order No.	Performance	Aircraft model	
		TrimbleUX5	**senseFlyeBee**
1.	Wings	Fixed	Modular
2.	Weight	2.5 kg	0.63 kg
3.	Wing span	100 cm	96 cm
4.	Camera	16.1 MP	16 MP
5.	Launching method	Catapult	From hand
6.	Flight duration	50 min	45 min
7.	Landing method	Autonomously on the underbelly with reverse engine operation	
8.	Landing area	50 x 30 m	20 x 20 m
9.	Extreme meteorological conditions	Winds up to 65 km/h Drizzle Temperatures from -25 to +55^{0}C	Winds up to 45 km/h
10.	Ground recording resolution	2.4 – 23.9 cm	3 – 30 cm
11.	Maximum flight altitude	5,000 m	800 m [?]
12.	Recording overlap	variable, 80% standard	50% - 85%
13.	Planning and surveillance software	Yes	Yes
14.	Data processing software	Yes	Yes

A functional categorization of unmanned aircrafts in the area of aero photogrammetry has not been created, but the crafts that are pertinent to the contents of this paper are classified as miniature unmanned models weighing up to several kilograms. This choice of craft is not coincidental, to the contrary, the low cost of acquisition and craft exploitation, satisfaction of functionality requirements, mobility and operation functionality, are the main determining performance criteria for the aerophotogrammetry crafts. According to the flight method, there are aircrafts of plane and helicopter type. In recording of larger surfaces, the aircrafts of the plane type are preferred. In order to illustrate and compare them, Table 1 presents the performance of aerophotogrammetry systems of two prominent and popular manufacturers.

The aircrafts can be launched from hand, or via catapult, Figure 2. Catapult launch does not depend on the skill of the operator, it is safer (the operator is at a distance from the propeller) and is more reliable in terms of ascent angle and ascension speed. The aircraft that are launched from hand are smaller and weigh less, they are more suitable for transport, handling and transport by hand.

Figure 1 *Unmanned craft launch, from hand and by catapult*

Aero photogrammetric crafts are controlled using a computer that is integrated in the aircraft or via remote control if necessary. The structure of the unmanned aerophotogrammetry system, besides the aircraft with the camera, propulsion system (battery and electric motor), sensors and GPS, is comprised of: the control system, a system of radio connection with the craft in order to transmit instruction, i.e. flight planning, surveillance and management, software for planning and flight programming and processing and recording interpretation software.

3 APPLICATION OF THE UNMANNED AEROPHOTOGRAMMETRY TECHNOLOGY

In comparison to the classic, the advantages of unmanned aerophotogrammetry are: availability, price, safety, efficiency and a wide range of applications. This technology does not require any special training for aero photo recording, which with classic technologies is an exclusive privilege of specialized geodetic and engineering institutions. The technology is available to a wide range of users and is familiar in terms of exploitation. It enables measurement of difficult, inaccessible or dangerous terrains and objects without risks. Data gathering and processing is efficient and does not depend on numerous factors as with classical aerial recording, and is faster than any terrestrial measuring technology.

This technology has great application potential, and even though it is not omnipotent, it does not have strict boundaries of practical uses in measuring, control and monitoring the state of field surfaces, regions and objects. There is a wide range of applications for the unmanned aero photo technology in mining, geology, geo-technology, geo-morphology, geodesics, geophysics, GIS, geography, biology, agriculture, forestry, water industry, construction, traffic, electric industry, urbanism, architecture, archeology, ecology, monitoring and preservation of the environment, spatial planning, land cadastres, safety surveillance, search and rescue operations, etc.

In mining, especially with the open pit exploitation of mineral ores, the unmanned aero photo technology provides an efficient visual surveillance in real time which was not possible until recent times:

- Instabilities in soil shifts (settling, creeping, slipping, sliding, landslides, erosion processes);
- Monitoring of geological research operations;
- Positioning and surveillance of the condition and changes in the exploitation fields;
- Monitoring of the operation dynamic at the open pits and depots;
- Actualization of the condition of the open pits and depots;
- Monitoring of the excavated masses of ore and waste, supplies at the depots etc.;
- Monitoring of the impact on the environment, reclamation activities and landscaping;
- Indication and monitoring of breakdowns;
- Security surveillance;
- Expropriation support etc.

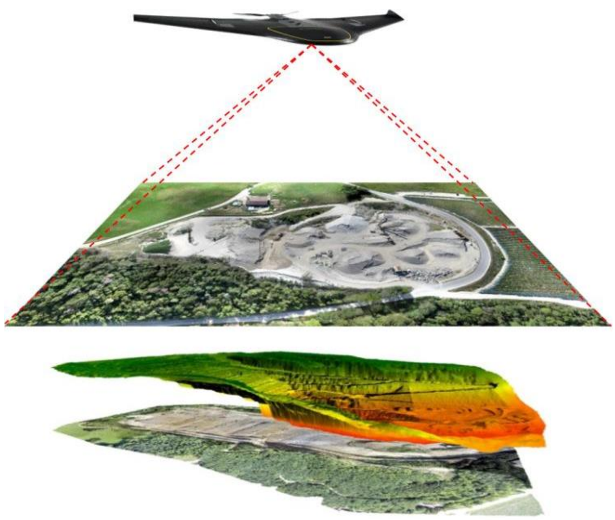

Figure 2 *Principal display of the aero photo recording of the open pit and the outcome results (orthophoto and digital model).*

The innovation of this technology is obvious, and its importance for surveillance upgrade and management of mining systems, with safety and production results, will be quantified during the time to come.

Unmanned aero photo technology can be applied in three ways in mining; the first is service by necessity, the second is periodical service, and the third is concession (selling/buying) of technology. The first two models, in investment and exert view, are suitable for mines of small and medium production capabilities, i.e. of modest financial potentials. The third model requires a direct investment of 25 to 60 thousand dollars and expert training, and is acceptable for large mining systems, however it is still recommended as step two due to the personnel adaptation and application the new technology, it is recommended to use, at least for a while, one of the first two models.

A team of experts from the Mining Institute and Livona of Belgrade had on July 24th, 2013 conducted the first aero photo recording in Serbia using an unmanned aircraft at the open coal pit Drmno in Kostolac. After a month, the recording was conducted again in order to compare the conditions of operations and excavated masses in the area of the excavated front. A comparative analysis has shown that the results of the aero photo recording via the unmanned aircraft are in correlation and are equivalent with the results of conventional geodetic measurements which are conducted according to regular procedure.

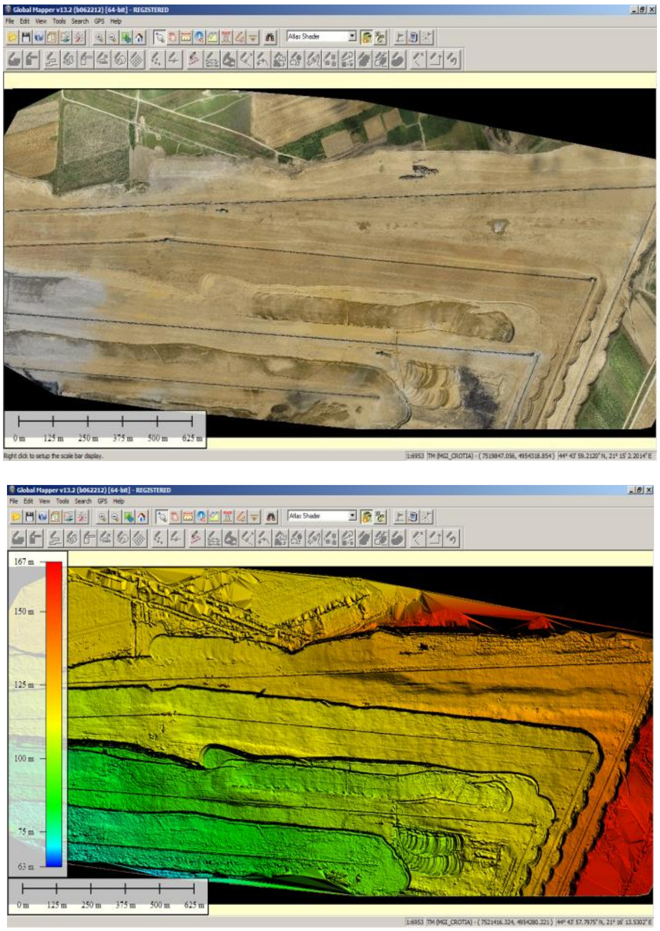

Figure 3 *The recorded area of the excavated front of the open pit of Drmno, orthophoto and digital model, July 24th, 2013*

The results obtained are not only a confirmation of the validity of the unmanned aerophotogrammetry technology, but also of its practical uses in mines of great spatial operations such as the coal open pit of Drmno.

The team has conducted experimental test recordings of the open pit of Stražilovo in Sremski Karlovci and of the terrain between the future open pit of Srednja strana and the Special nature reserve of Slano Kopovo in Novi Bečej in 2013. The results in these conditions were identical to the results of the recordings at the open pit of Drmno.

CONCLUSION

The advantages of the unmanned aero photo technology in comparison to the conventional photogrammetry technologies lie in lower cost, constant availability and accessibility. The significant comparative advantage of the unmanned aero photo technology also lies in the fact that it does not require specialized personnel for photogrammetry, pilots, navigators and other assisting personnel. It can be expected that with further advancement and development of unmanned aero photo technology, the advantages will be even greater.

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Slobodan Vujić[*], Milinko Radosavljević[*], Igor Miljanović[**], Mihajlo Gigov[*]

INVESTMENTS AND BENEFITS IN COMPUTER SUPPORTED SYSTEMS FOR REMOTE MONITORING AND MANAGEMENT IN REAL TIME

Abstract

Contemporary computer supported technologies unify a multitude of functions, from geological explorations, through planning, design, mining, auxiliary and logistic activities, monitoring, analysis, decision making to the real time management. This approach requires the information-management systems with a multi-level hierarchy structure, with high monitoring-management efficiency. Accepting the challenges of time, the Serbian mining took the starting position in this technological game timely, with some good ideas and projects, but without efficient realization. There is an opposite example however, in 2000, a computer system for remote monitoring and management in the real time at the "Majdan III" open pit mine was built. Fourteen years of continual monitoring of this systems operation and the quantity of technical and technological data on its functioning, has provided a reliable and objective depicting of the monitoring-management efficiency, investments, costs and benefits. This paper presents the actual results achieved with this system and comparatively analyze investments and benefits for the similar systems in coal and non-ferrous metal mines.
Keywords: *mining, computing, monitoring, management, investment, benefit*

1 INTRODUCTION

Great investments were made on a global scale into innovation and development of mining technologies in order to increase the productivity and production efficiency. In accordance with these efforts, the mining companies dedicate a special attention to development and application of automation, computer supported systems of remote control and management, and development a new generation of robotic ("smart") machines. Significant progress was made du to the accomplishments in the computer technologies department, but foremost due to the achievements in the communications department, spatial positioning in real time, sensory, regulation technique, systematic and mathematic-model philosophy.

One of the current subjects is the energy efficiency. Automation and process management increase the energy efficiency through decrease in oscillations, an increase in stability and consistency in task completions according to the defined work program of technological system. Lesser fuel consumption results in reduced gas emissions from internal combustion engines that power machines, lower operation expenses and other benefits. Computer supported systems for remote control and real time management enable the ecological monitoring, a reliable and precise change monitoring and change analysis (noise, seismics, dustiness, gas concentration,

[*] *Mining Institute, Belgrade, Serbia, slobodan.vujic@ribeograd.ac.rs*
[**] *University of Belgrade, Faculty of Mining and Geology, Serbia, igor.miljanovic@rgf.bg.ac.rs*

temperature, level and quality of ground and surface water, etc.) in the area impacted by the mining operations, and in case of emergencies, timely correction operations.

Integration of computer supported systems for remote control and management with the business platforms of mines creates a reliable and efficient ambiance for decision making support.

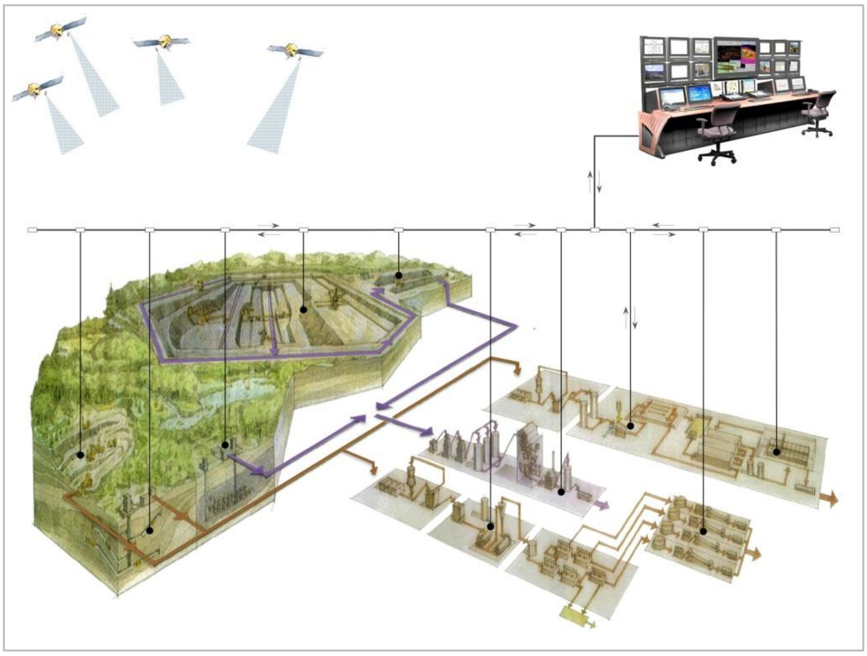

Figure 1 *Illustration the computer supported mine system for remote control and real time management*

Mining production is a sequential series of connected processes starting with excavation (or drilling and blasting), loading, transport to the ore processing and final product transport. Installation of automation and remote management of production line, means usage of sensors providing the data necessary for monitoring and operation management, which is essential for work safety, especially under difficult weather conditions at the open pits (reduced visibility, snow, rain and other weather patterns), for prevention the collisions and truck turnovers during material unloading at depots, for machine failure and breakdown prevention, efficient planning and equipment and machine maintenance.

Automation and process management has a significant role in the mining economy, it is of immediate use when it comes to cutting costs, increasing the work discipline, temporal and capacitive usage, detection and elimination the loop holes in the technological process, decrease in the consumption of energy and standard materials, over utilization prevention, malfunctions on machines and equipment, etc.

The usage and combination of knowledge of mining engineering and science with the achievements in the computer technology field, robotics, mechatronics, telemetry, satellite positioning, sensory and communication technology, applied mathematics, fuzzy logic, artificial intelligence, management theory and systems sciences, has achieved impressive, but not final results in this field in a short amount of time. Numerous issues and tasks are yet to be solved as well as issues in automation and process management in mining, foremost the issues with sensors. There are still no sensors comparable to human senses.

2 EFFECTS OF IMPLEMENTATION THE REMOTE CONTROL AND MANAGEMENT SYSTEMS IN MINING

The base functions of computer supported IMS (Information-management systems), i.e. the control and management systems of mining complexes, and the effects that are made through this systems implementation, related to:

- Control of machines and equipment (prevention of failures, malfunctions, and maintenance regiment. Prevention of borderline situations and incidents and operation modes of equipment and machines, which eliminates or reduces malfunctions, failure and equipment damage which extends its lifetime and reduces the maintenance costs);
- Process control (indications of crisis and incident operation modes);
- Process management (machine and equipment management, and process parameters management);
- Optimization the mining parameters of machines and equipment;
- Technical process efficiency (the technical process at a particular level and operation mode);
- The efficiency of reaching management decisions (timeliness and expert grounds);
- Economical usage of available resources (management response to changes in operation conditions);
- Minimization the subjectivity of human factor, especially in conditions of sudden and critical operation disruptions of equipment and facilities;
- Positive incentive and motivation of workforce (training, development, education);
- Work safety (reduction of the disruptive influence of atmospheric conditions, reduced visibility, snow, rain, etc.);

There is a vast array of useful quality and quantity effects from the control and management systems on technical, technological, economical, job and wider areas.

The resulting key effects are efficiency, productivity and cost effectiveness of the production process, which through direct savings compensates investments in IMS in short-term.

Indirect effects are of special significance (increase in work discipline, responsibility, attitude towards work and work assignments, etc.) since the computer control and management systems have been introduced into the production process. The introduction of control and management systems influenced an increase of motivation of employees for work and development, due to the advantages provided by the system in completion the work assignments, handling of modern technology and enabling independent development of application solution, etc.;

The control and management systems exploitation directly and indirectly influences the creation of objective technical conditions and knowledge and experience accumulation for further development the systems hiring the existing workforce and spreading the system application, and through it the growth of positive effects of investment in IMS.

Investment in computer supported control-management mining systems in mining is not a classical investment endeavor in terms of going into the new production programs, creation of new capacities, cornering the market with the new products, services, etc. Therefore, starting with the relevant instructions of current international methodologies (UNIDO, World Bank, European economic commission and their approach methodologies to the projects with the goals to increase the production, energy, resource efficiency, etc.), there is and can be no classic investment analysis as with the industrial feasibility studies.

With evaluations such as these, the focus is on the technical-technological solution for the control-management system and evaluation the effects of its implementation for the purpose of improvement the exploitation characteristics of existing capacities and to raise the realization quality of production assignments, with the rational energy consumption, material resource and other savings and positive effects.

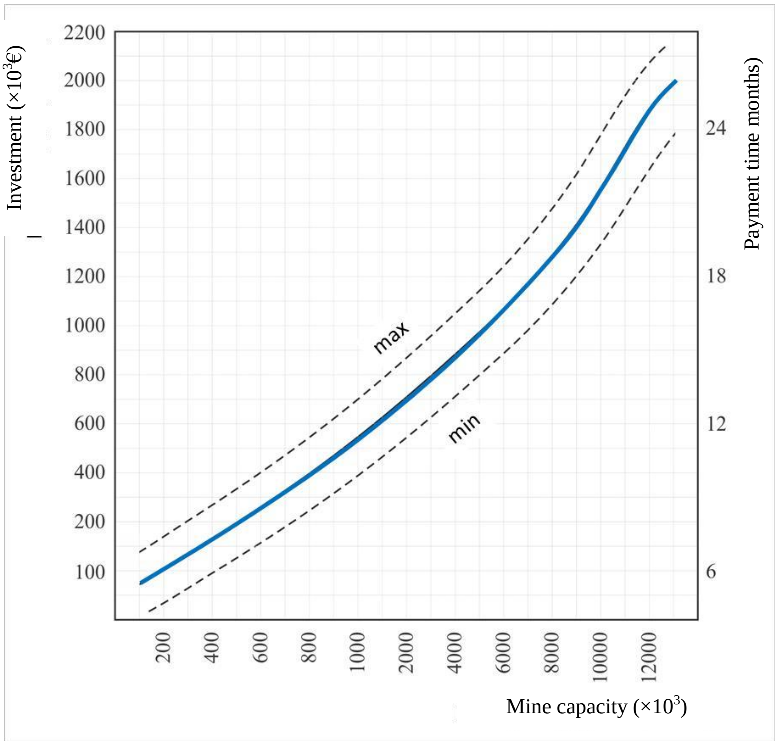

Figure 2 *Codependence of investments in mining control-management systems, time it takes for a payoff and the mine capacity*

Certain elements in these projects cannot be precisely quantified, so an evaluation model is used, based on experiences with similar projects or analogous situations. This also pertains the evaluations of qualitative investment effects.

All of this does not reduce the expert basis, objectivity and the wholeness of the final evaluation about investment justification for the computer supported control-management systems.

According to our analysis and numerous expert reports regarding examples and practices, the period for repayment the investments in such systems can range from several months to two years maximum, a year on average, depending on the size and complexity of the production system, work conditions, manners and conditions of securing the financing, etc. Figure 2 presents a dependence of amount of investments in SMS (Control-Management System) and the time it takes to repay them on the mine capacity. The amount of investments made into the control-management systems and the time it takes to repay them depends on numerous factors in mining, therefore the diagram should be taken as an orientation indicator.

3 OUR EXPERIENCE

The open clay pit Majdan III of Potisja from Kanjiža is the first on which the control-management technology with the satellite positioning system and navigation was applied on, and the first mine in Serbia with and integrated computer support system for remote control and real time management, Figure 3.

Fourteen years of operation monitoring and experience with the ECD (excavator-conveyor belt-disposer) complex with a computer supported IMS, enables an argumented and objective overview the effects, results and gains.

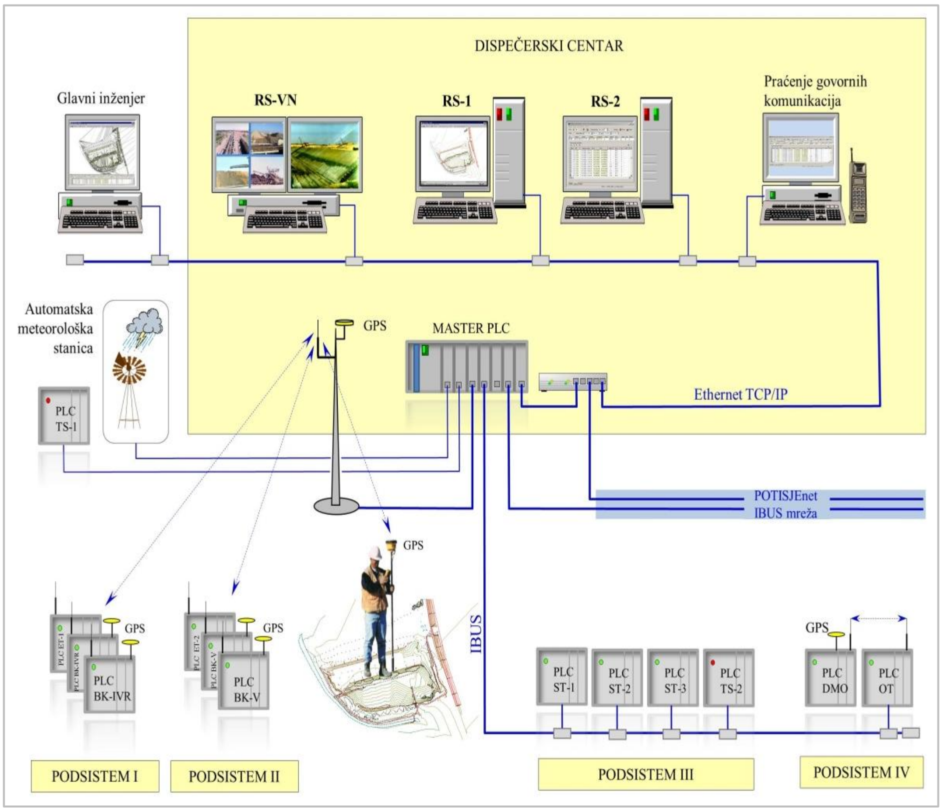

Figure 3 *Physical topology of remote control and management system of the open pit Majdan III Potisje Kanjiža (1998)*

Because the sophisticated control-management and exploitation (ECD) technology are intertwined, many benefits and savings were made, and these are the most significant: transport expenses have been reduced 4 times as much as the costs of cyclic transport at the previous open pit Majdan II; monthly oil consumption has been reduced by 50,000 liters; electricity consumption has been increased but the monthly savings is about 21,600 Euros; the effective operating time at the open pit was increased by 792 hours a year; reliability and system operation safety is high in all weather conditions; maximum efficiency in excavation, homogenization and selective clay deposition has been achieved; negative environmental impacts have been removed, there are no gas emissions or dust, the noise level has been reduced, etc.

Specific investments in the information-management system OP Majdan III amount to 0.375 Euros per ton of clay at annual level, for t depreciation period investments are 0.095 Euros per ton of clay.

In order to keep up with the competition according to the world standards, technical, technological and organization improvements are not sufficient. An efficient control and real process management system is also required. As the first project of integrated IMS in our country, *Project IMS of the Open Pit Tamnava West Field of the Mining Basin (MB) Kolubara*, its task was to affirm the idea and pave the way for introducing and applying the computer supported information - management technologies in our mining. The IMS of the open pit Majdan III Potisje Kanjiža was designed and built on the ideas of the IMS of the open pit Tamnava West Field, Figure 4, and it can be noticed that the IMS project of the open pit Tamnava West Field has a positive reflection.

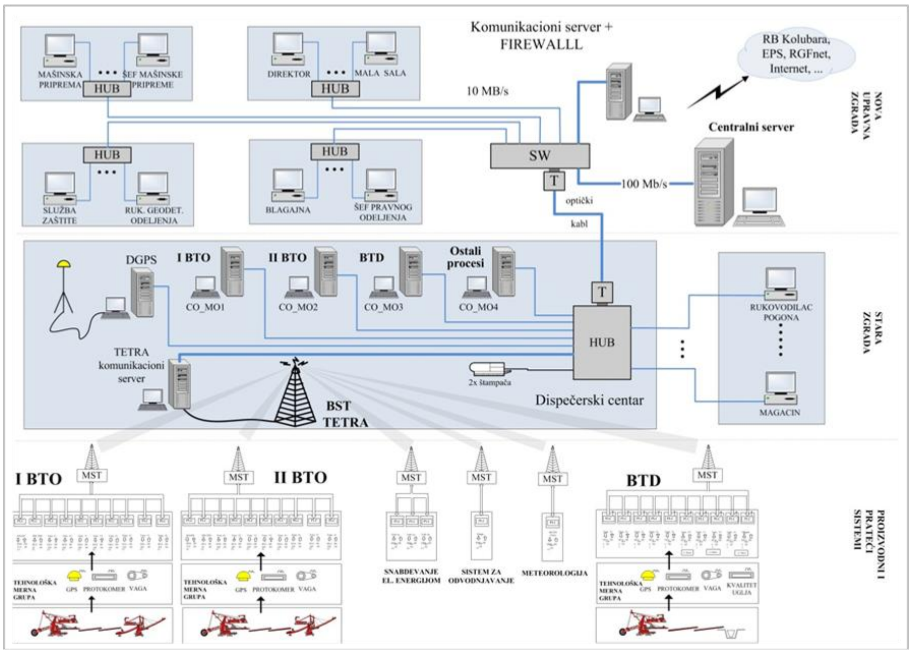

Figure 4 *Physical topology of remote control and management systems of the open coal pit Tamnava West Field MB Kolubara (1996)*

The funding required to construct IMS at the open pit Tamnava West Field are modest when compared to the investments in main and auxiliary equipment and machines. According to the measured production of the open pit Tamnava West Field, the specific investments are 0.078 Euros per ton of coal.

The "cost-benefit" analysis shows that the investment in the computer supported system of control and mine management and flotation of the mining company Rudnik Rudnik, Figure 5, with a production increase by 1% and savings, will be repaid in less than 18 months. The effects of establishing the remote control and management systems through flotation of Veliki Krivelj in the Bor Copper Mine are identical.

According to the *Project of Control-Management System of the Open Coal Pit Bogutovo Selo of the Mine and Thermal Plant Ugljevik*, the annual production investments are 0.175 Euros per ton of coal, and for depreciation period the installed equipment into the control-management system it is 0.044 Euros per ton of coal. In comparison, the same funds are required to obtain a mid-class bulldozer.

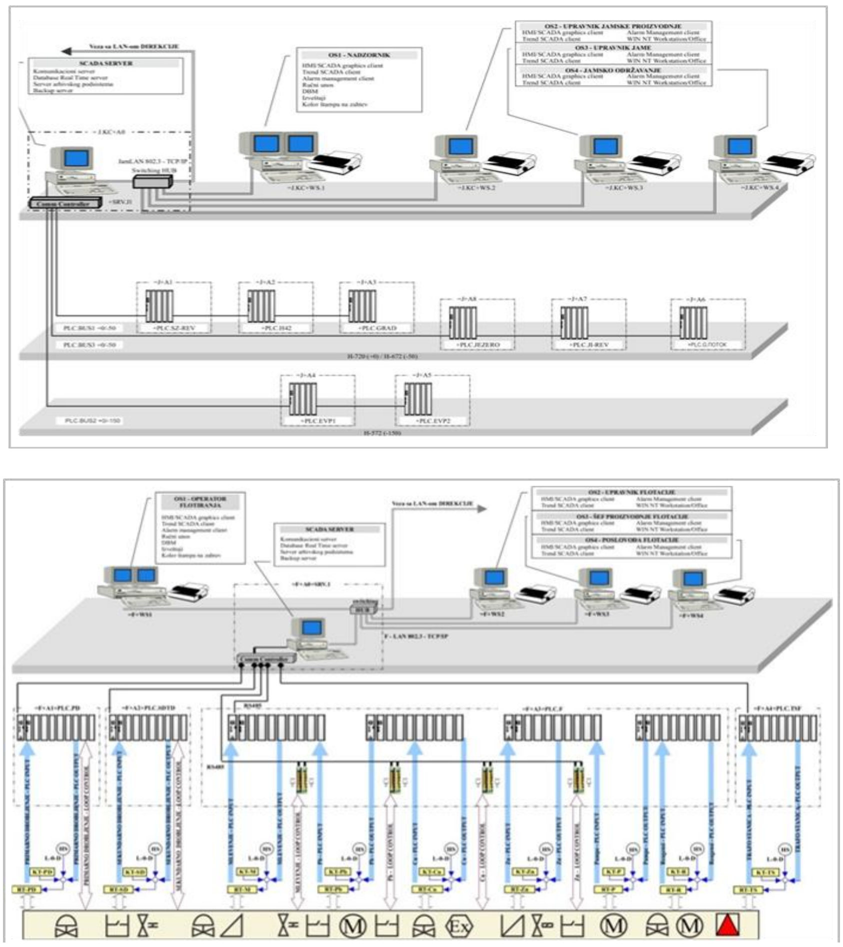

Figure 5 *Physical systems topology of remote control and management systems of the mine (up) and flotation (down) of the mining company Rudnik Rudnik (2000)*

The specific construction costs of the *Computer Supported System for Remote Control the Open Coal Pit Drmno - Thermal Power Plant and Open Pit Mines Kostolac (TPP-OPM)*, Figure 6, are 0.185 Euros per ton of coal per year, and for depreciation period the equipment of the control-management system it was 0.045 Euros per ton of coal.

At present production levels of the open pit of Veliki Krivelj, Copper Mine Bor, the specific investments in the remote control-management system, Figure 7, are 0.039 Euros per ton of ore. Investments in the remote control-management system of the Copper Mine Majdanpek have a similar effect.

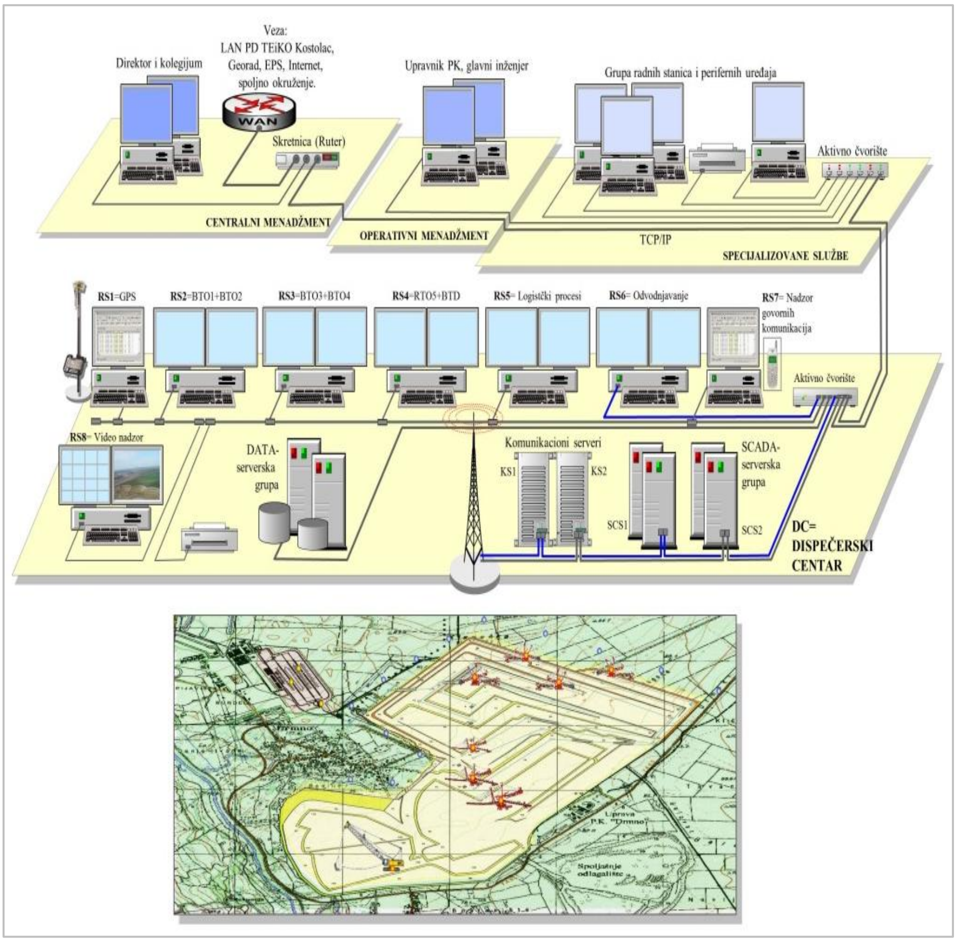

Figure 6 *Physical topology the remote control and management systems and management of the open coal pit Drmno, TPP-OPM Kostolac (2007)*

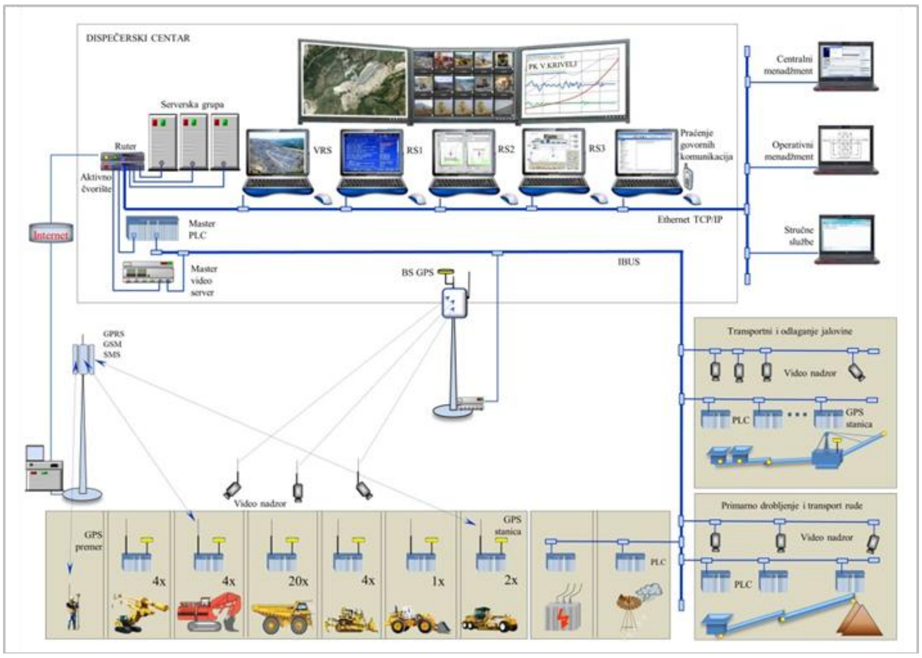

Figure 7 *Physical topology of the remote control and management systems of the open copper pit of Veliki Krivelj, Mining and Smelting Basin Bor (2012.)*

CONCLUSION

Findings, practice experiences and results of numerous engineering analysis in designing the computer supported systems for remote control and management in real time, confirm the profitability of these systems. The investment payout period ranges from a couple of months to two years. With establishment the computer supported control-management systems in mines, the metrics of profitability cannot be reduced to just material correlation of cost and benefits due to the significant exploitation benefits from technical, technological, innovative and safety aspects, it is not possible to quantify them with just monetary value. That is why the analysis and evaluation, when arguing for investments and benefits of control-management systems, must be based on multi attribute principles.

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Lidija Djurdjevac Ignjatović, Dragan Ignjatović*, Milenko Ljubojev*, Marko Mitrović**

CHANGE THE UNIAXIAL COMPRESSIVE STRENGTH OF PASTE BACKFILL DEPENDING ON CHANGE THE PARAMETERS**

Abstract

Traditional methods of ore mining in the underground mine Jama Bor cannot be applied to the ore body Borska reka for several reasons: the position of the ore body to the terrain surface, mining infrastructure and proximity of settlements, physical-mechanical properties of rocks in the ore body, economic viability, ore recovery, etc. In addition to the above, it should be taken into account that this is the ore body with low copper content, so it is necessary to choose a method for mass excavation. Due to these reasons, the mining method of ore was selected with backfilling of excavation area with paste backfill of certain physical-mechanical properties.

This work will be present the test results of uniaxial compressive strength of paste backfill, as one of the most important properties and how this parameter varies depending on the change in balance of components of which a paste backfill is made. Tests were conducted on a number of different recipes for paste backfills and the backfill with 5% cement, 24% water and 71% uncycloned flotation tailings was selected as the best one.

Keywords: *pasta backfill, flotation tailings, uniaxial compressive strength, ore body Borska reka, mass excavation method*

1 INTRODUCTION

Location of deposits and ore bodies for underground mining in Jama Bor, in relation to the terrain surface, old works in Jama and at the Open Pit in Bor in relation to the mining infrastructure limits the selection of methods for excavation. Previous technologies and methods can no longer be applied. Also, above the deposit, on the surface of terrain, there is a settlement (town Bor), roads, urban and industrial infrastructure, railway with the tunnel, collector for a few surface watercourses, as well as the industrial facilities.

Due to the mentioned reasons, it was necessary to select the method of exploitation that would be appropriate from the other aspects: adapted to the physical-mechanical properties of rocks, to achieve large capacity and productivity with minimum decrease of the ore grade, as well as better economy. The method of ore mining ore with backfilling of space with paste backfill of corresponding physical-mechanical and technological characteristics seems to be well selected method, which would meet all requirements listed above.

* *Mining and Metallurgy Institute Bor*
** *This work is the result of the Science Project No. 33021, "Research and Monitoring the Changes in Stress Deformation State in the Rock Mass "IN SITU" around the Underground Chambers with Model Development with a Special Emphasis on the Tunnel of the Bor River and Jama Bor", funded by the Ministry of Education, Science and Technological Development of Republic of Serbia.*

For conditions of making a paste backfill, it is given that the basic raw material for making is flotation tailings and as additives to use the other raw materials of RTB (slag, ash, mine waste) in order to improve the characteristics of paste backfill.

The laboratories of the Mining and Metallurgy Institute Bor have carried out the tests of physical-mechanical properties the samples of paste backfill of different recipes, as well as chemical and granulometric properties of flotation tailings, as its basic component.

The main objective of this test was to find an optimal recipe for the paste backfill, which will meet all essential properties, both in terms of physical-mechanical properties, but also in terms of economy.

The following text will show the changes of uniaxial compressive strength (Uniaxial Compressive Strength, hereinafter referred to as **UCS**) as one of the most important characteristics of the paste backfill.

2 PHYSICAL-MECHANICAL CHARACTERISTICS OF PASTE BACKFILL

In order to meet the basic physical and mechanical properties of paste backfill, and here primarily refers to consistency, compressive strength and load capacity, an appropriate recipe must be found, i.e. the ratio of its main components: **flotation tailings - cement - water**. One of the main requirements was that the uniaxial compressive strength after 28 days must be in the range of 1-1.5 [MPa]. For this purpose, as a starting point, the experienced data from the mines in the world (copper mine Čelopek - Bulgaria, Sweden, Canada, Zimbabwe) were used that already have used the paste backfills in their practice.

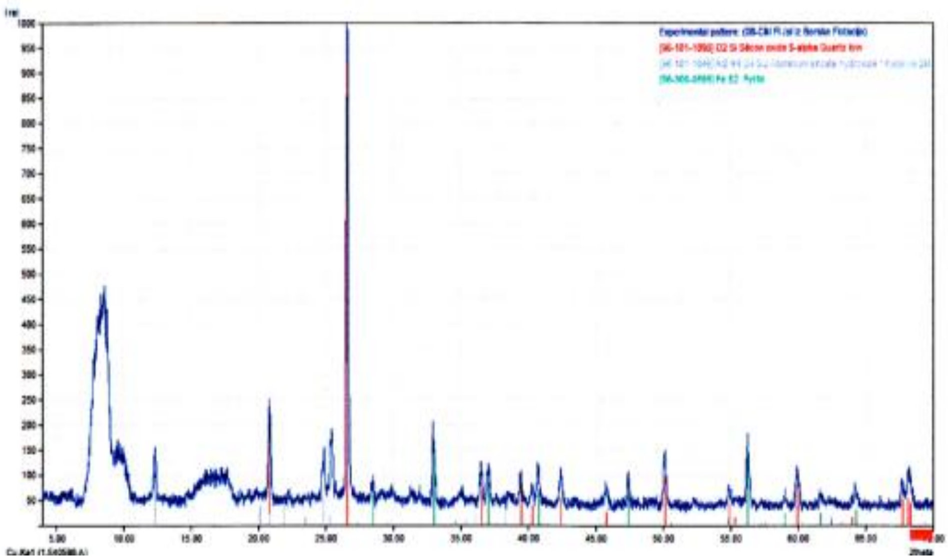
The samples obtained from different recipes were used for carrying out the following physical-mechanical tests:

- Determination the measure of paste consistency by the method of subsidence - Abrams cone,
- Testing the compressive strength of samples after 7, 14, 21, 28 and 90 days,
- Triaxial compression test of samples after 28 (90) days,
- Deformation characteristics of samples after 28 (90) days,
- Determination the cohesion and angle of internal friction of samples after 90 days of shearing test per discontinuity.

The main component of the paste backfill is flotation tailings obtained after ore processing. The original idea was to use the cyclone flotation tailings with the addition of cement and water. After the obtained results of such obtained paste backfill, tests were also carried out with uncycloned flotation tailings.

2.1 Testing the Uniaxial Compressive Strength (UCS) of Paste Backfill with Cycloned Flotation Tailings

The initial idea was to test the properties of paste backfill tailings with ***cycloned*** flotation tailings and cement from the factory "Holcim" - Popovac (cement designation 42,5R). Mineralogical tests and grain size distribution were carried out on it. Report on mineralogical analysis is presented in Figure 1.



Mineral name	Chemical formula	Content (%)
Quartz	SiO_2	46.6
Kaolinite	$Al_2Si_2O_5(OH)_4$	31.2
Pyrite	FeS_2	22.1

Figure 1 *Mineralogical analysis of cycloned flotation tailings*

After these tests, the recipes for obtaining the optimal characteristics of paste backfill were done. Total of 5 recipes were done with different content of cement and cycloned flotation tailings at the ***same water content of 25%,*** Table 1.

Table 1 *Recipes for making the paste backfills at water content of 25%*

Order No.	Amount of cement, %	Amount of cyclone flotation tailings, %
1.	4	71
2.	5	70
3.	6	69
4.	7	68
5.	8	67

USC tests of thus obtained paste backfill showed the following, Figure 2.

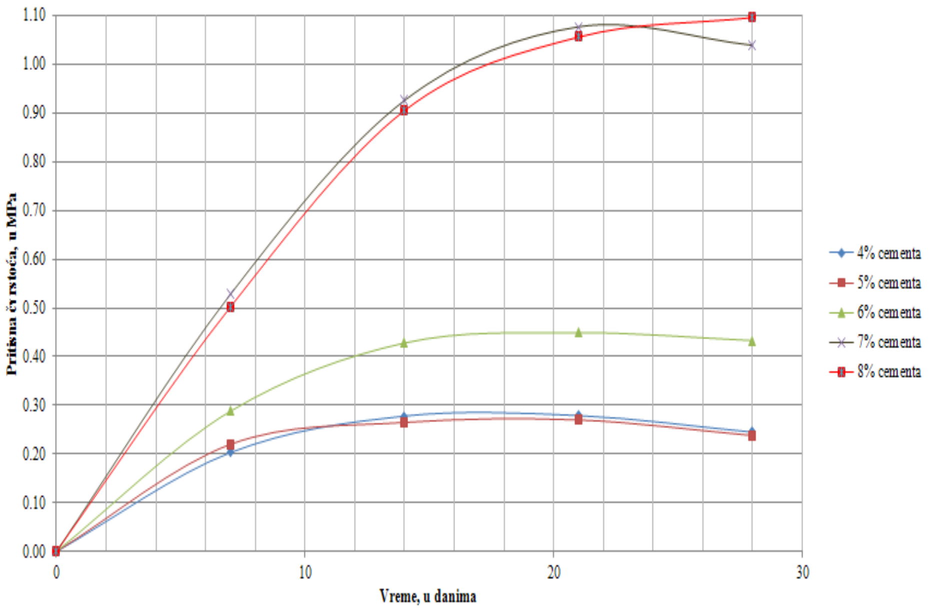


Figure 2 *Cumulative diagram of uniaxial compressive strength of paste backfill for different contents of cement and cycloned tailings and at the same water content of 25%*

Measuring of consistency for the method of subsidence has shown that thus obtained ***material is in the plastic limit.***

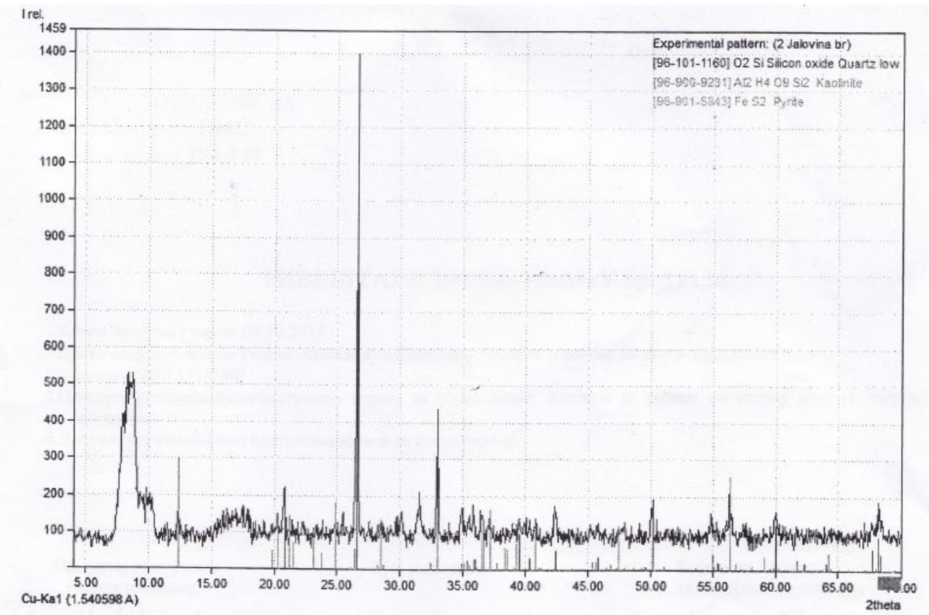
The binding time has shown that thus obtained backfills beginning with the binding after 10 hours.

As you it can be seen from the graph, the basic condition for UCS of 1-1.5 [MPa] is fulfilled by the paste which contain cement of 7 and 8%. However, in terms of economic viability, thus resulting paste backfills do not meet.

For this reason, it is passed on further investigation of paste backfill, which would contain in itself the ***uncycloned flotation tailings*** and small amount of water.

2.2 Testing the Uniaxial Compressive Strength (UCS) of Paste Backfill with Uncycloned Flotation Tailings

After unsatisfactory results obtained using the cycloned flotation tailings, preparing the new recipes was done, which would contain the uncycloned flotation tailings and small amount of water. The quality of cement remained the same as in previous tests. Figure 3 gives a graph mineralogical analysis of uncycloned tailings.



Mineral name	Chemical formula	Content (%)
Quartz	SiO_2	47.1
Kaolinite	$Al_2[Si_4O_{10}](OH)_8$	39.6
Pyrite	FeS_2	13.3

Figure 3 *Mineralogical analysis of uncycloned flotation tailings*

In the first case, 4 recipes were done with different content of cement and flotation tailings, at water content of 20%, Table 2.

Table 2 *Recipes for making the paste backfills at water content of 20%*

Order No.	Amount of cement, %	Amount of uncycloned flotation tailings, %
1.	3	77
2.	5	75
3.	7	73
4.	9	71

USC tests of thus obtained paste back fill showed the following, Figure 4.

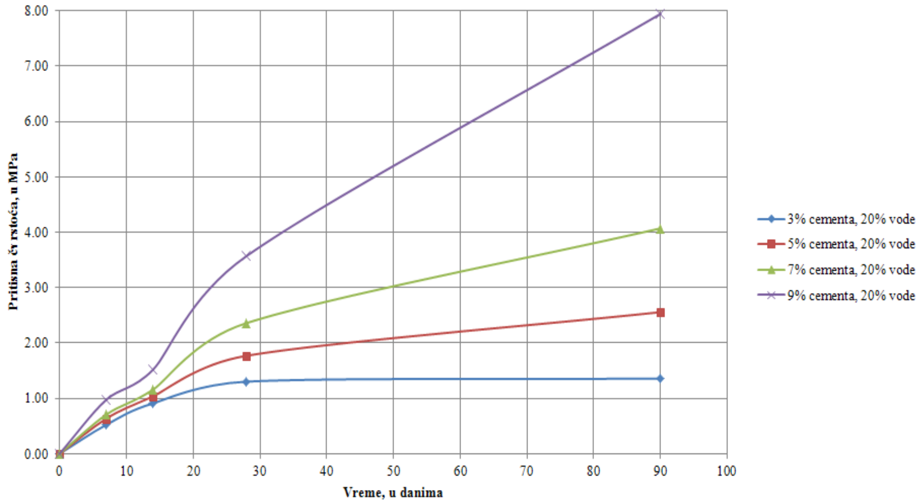


Figure 4 *Cumulative diagram of uniaxial compressive strength of paste backfill for different contents of cement and uncycloned tailings and at the same water content of 20%*

In terms of consistency, thus obtained backfills are quite thick, i.e. they belong to a group of ***poor plastic materials***.

The binding time was also only after 10 hours.

It can be seen from the cumulative graph that the basic condition for UCS of 1-1.5 [MPa] is fulfilled after 28 days even for the paste recipe which contain 3 % of cement.

Although of excellent compressive strengths, thus obtained paste backfills would require far more complex kind of material transportation to the place of installation, which raises the process of backfilling process.

Since it is necessary to achieve fluidity in these backfills, testing the paste backfills has started with the uncycloned tailings, but with a higher percentage of humidity (24%). Content of cement in paste is not changed (3, 5, 7, and 9% of the same quality cement), except the amount of water and uncycloned flotation tailings, Table 3.

Table 3 *Recipes for making the paste backfills at water content of 24%*

Order No.	Amount of cement, %	Amount of uncycloned flotation tailings, %
1.	3	73
2.	5	71
3.	7	69
4.	9	67

USC tests of thus obtained paste back fill showed the following, Figure 5.

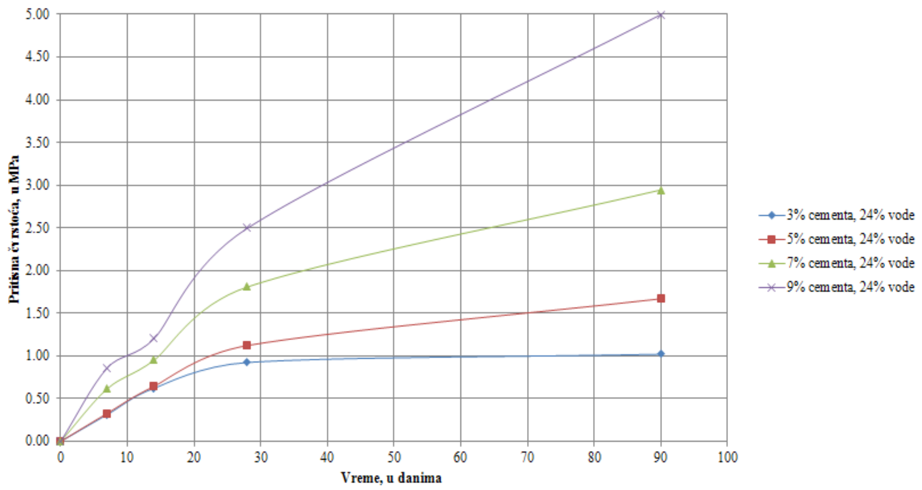


Figure 5 *Cumulative diagram of uniaxial compressive strength of paste backfill for different contents of cement and uncycloned tailings and at the same water content of 24%*

In terms of consistency, thus obtained backfills belong to a group of ***materials with liquid consistency.***

The binding time was also only after 12 hours.

It can be seen from the cumulative graph that the basic condition for UCS of 1-1.5 [MPa], after 28 days, is fulfilled for the paste recipes which contain 3 % of cement. Thus obtained paste backfills, besides the physical - mechanical, also meet the economic parameters.

Comparing the results obtained with cycloned tailings and 25% water and uncycloned tailings and 24% water, a significant difference can be seen considering UCS, Figure 6.

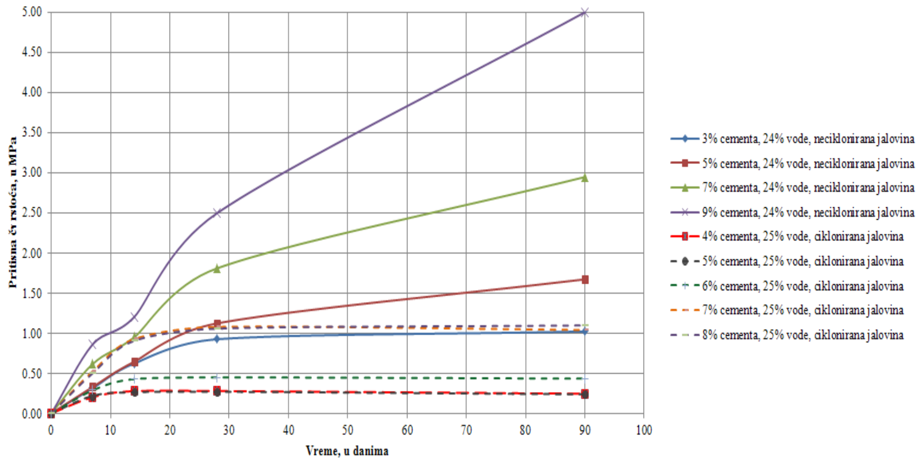


Figure 6 *Comparative analysis of paste backfills obtained from cyclone and uncycloned tailings*

CONCLUSION

Due to numerous difficulties that would occur in the traditional excavation of the ore body Borska reka, it is necessary to select the method of exploitation that would be suitable, both in terms of physical-mechanical properties as well as economically. For this reason, the selected method is the chamber pole method with back filling of excavation area.

Selected method also required a good selection of paste backfills, which must meet the demanding physical-mechanical characteristics of the structural elements in the stopes.

The laboratories of the Mining and Metallurgy Institute Bor have carried out the tests flotation tailings as the basic material for backfilling, but also the paste backfills obtained from different recipes of three basic elements: **flotation tailings – cement – water**.

Based on present tests of compressive strength of paste backfills with cyclone and uncycloned flotation tailings, as well as the other tested physical-mechanical properties, it can be concluded that the paste backfill, made with uncyloned tailings, has far better characteristics than the pate backfill with cyclone tailings. Since at first the opposite would be expected as the grain size distribution of uncycloned tailings is the most unfavorable due to the finest fractions, the test results showed that the thus obtained paste backfills satisfy the expected characteristics.

Due to such obtained good results, should further research should be continued as well as the application possibilities of the thus obtained paste backfill, but also to ttest the properties of paste backfill with additives to improve the quality of concrete.

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Saša Mitić, Nenad Makar*, Nenad Radosavljević*, Zoran Despodov***

EXCAVATION BETWEEN THE LEVELS H-910 AND H-830 IN THE "SVINJA REKA" MINE DISTRICT OF THE "SASA" MINE - MAKEDONSKA KAMENICA

Abstract

In this paper, the principles of mining between the levels H-910 and H-830 in the "Svinja reka" Mining District of the „Sasa" lead and zinc mine located in the M. Kamenica, are presented. Excavation has to be realized by implementation the sublevel caving mining method, while temporarily leaving the safety pillars, which have to be blasted in the second phase of excavation. The paper presents development works, as well as works for the ore blasting, based on the designed mining method. Also, the techno-economic parameters of mining method and work organization are presented.

Keywords: *lead and zinc, Svinja reka mine, sublevel caving method*

INTRODUCTION

Exploitation in the "Sasa" mine is done according to the approved documentation [1, 2, 3, 4]. In order to secure the amount of ore required for the designed capacity of 750,000 t/year, it was decided that the excavation should be intensified in the Mining District "Svinja Reka", primarily between the levels H-XIVb and H-830. This part of the mining basin was chosen for excavation intensification primarily due to the mining-geological ore qualities and waste rock, as well as due to the acceptable metal content in the ore. Having in mind the metal content in this part of the basin, as well as the present metals prices in the world market, it was concluded that intensifying the excavation of this part of the basin would provide the satisfactory economic effects. This paper presents the first excavation stage of the aforementioned part of the basin, between the levels H-910 and H-830.

ORE RESERVES BETWEEN THE LEVELS H-910 AND H-830

Geological explorations the ore mass between H-XIV and H-830 were done by the surface exploration drilling, pit exploration drilling and pit exploration operations. The shape and dimensions of the ore masses are changeable; by the way they extend and drop. Geological explorations identified three ore masses - shelf, middle and roof [3].

After conducting the explorations, the ore reserve amounts were calculated for levels between H-910 and H-830 in the mining district "Svinja reka", and displayed in the verified study of mining reserves, i.e. Table 1.

* *Mining Institute, Belgrade, podzemna@ribeograd.ac.rs*
** *Faculty for natural and technical sciences, University Goce Delcev, Štip, Zoran.despodov@ugd.edu.mk*

Table 1 *Ore reserves between the levels H-910-H-830 between the profiles 1350-400*

Category	Q (t)	Pb (%)	Zn (%)	Pb (t)	Zn (t)
A+B	1,795,582	4.83	3.28	86,727	58,895
C_1	3,572,025	4.41	3.27	157,526	116,805
Total A+B+C_1	**5,367,607**	**4.55**	**3.27**	**244,253**	**175,700**

OPENING AND BREAKING IN

Opening and breaking in of the "Svinja reka" basin was done with the Central mineral loose and export shaft, i.e. multiple excavation sites, ramps and raises. All opening and breaking in pit area are designed and made in accordance with the engineering standards, as well as law and regulations [1].

The pit areas are designed based on the dimensions of equipment to be used during excavation, designed production capacity for individual arts of the basin, as well as the necessary transport capacity and transport of blasted materials - ore and waste. The technology of construction of all pit areas of opening and breaking in was designed by application the classic technology of drilling-blasting operations, using loading and transporting powered by diesel fuel, in combination with rail transport - pit locomotive and cars [1, 2].

EXCAVATION BETWEEN H-910 AND H-830

Based on data obtained during the exploitation of higher areas of this basin, between H-XIVb – 990 and H-990-910 [2,3], this appears to be a stable work environment, with a solid ore, which is partially crushed, while the trace waste rocks cannot bear large loads so the building of supports is necessary.

Based on this data, in order to excavate this part of the basin, a **sublevel caving method was adopted from the lowest level to highest.**

Depending on the richness of ore mass (up to 10 m and over 10 m), the variants will be applied in stages - I stage is comprised out of excavation beneath the protective plate, while the II stage encompasses the ore excavation beneath the protection plate.

This paper presents the variant which will be used in the most of ore mass, and relates to the sublevels - intervals PE 910-7 to PE 910-70, which are beneath the protective plate.

The excavation geometry is shown in Figure 1 [4,5].

The excavation starts with the raise for kerf (RK), constructed between two sublevels, using a classic technology of mining-blasting operations. The construction is done in segments of 2 m. The segment number depends on the richness of ore vein and its geometry angle [4,5].

After the construction of U3, the ore blasting begins, with drilling-blasting operations. Excavation is conducted using the Rocket Boomer 281 (Atlas Copco, Sweden) car type from the raise for the kerf, in retreat, towards the access stope (PrH). Drilling of the sublevel plate is done in segments. The length of the mining drill holes is 3.0 m and the drilling is done with an angle of 40°. The number of production mine drill holes depends on the thickness of ore vein - width of the stope.

The stope-room b=15 m wide is located between two safety pillars (SS), and is divided into several excavation drifts (OH) between the excavation pillars are placed (OS). The excavation pillars are of minimal width b_{os}=3.0 m. All of these areas, as well as the demolishing of the excavation pillars is done with the drilling car Rocket Boomer 281 (Atlas Copco, Swe). Depending on width of the stope-room, the number of excavation drifts (OH) and excavation pillars will differ, and in this sense their proper designing is necessary, as well as constant geological and mining exploration works during excavation, so as to enable proper excavation. This is especially important due to maximum decrease in depletion and maximum use of ore. This means that the excavation drifts (OX) and excavation pillars (OS) do not have to be of equal dimensions [1].

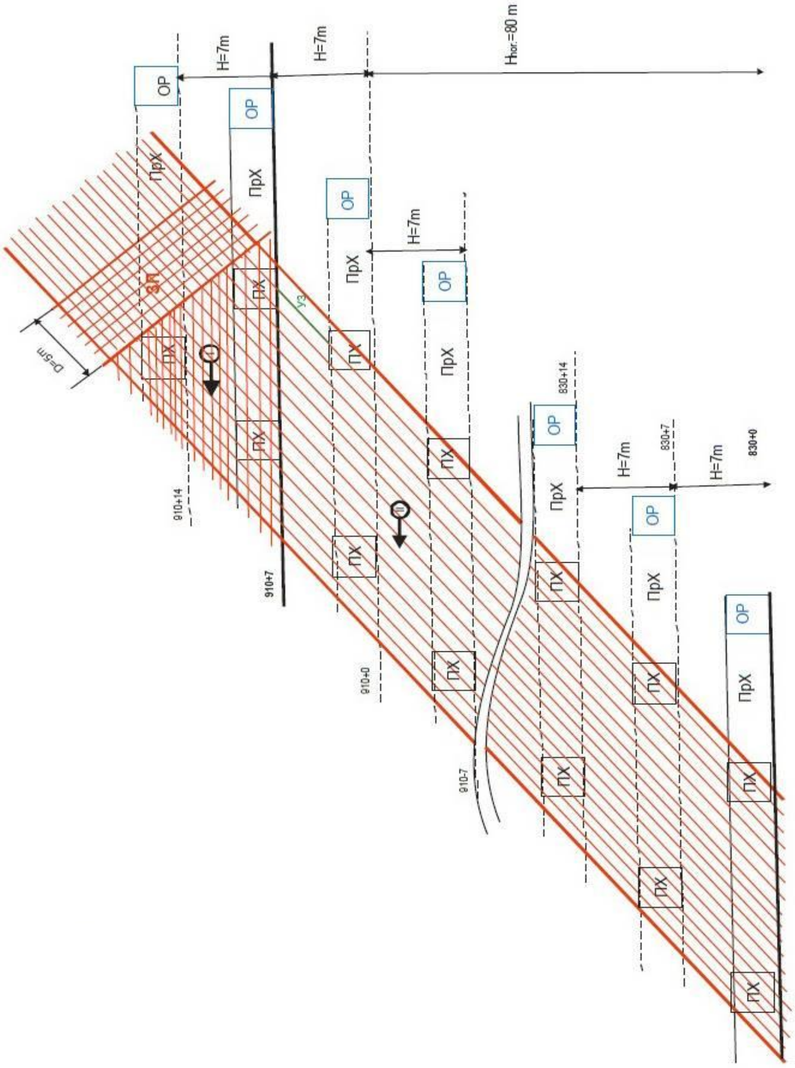

Figure 1 *Stages of preparation and mining block excavation*

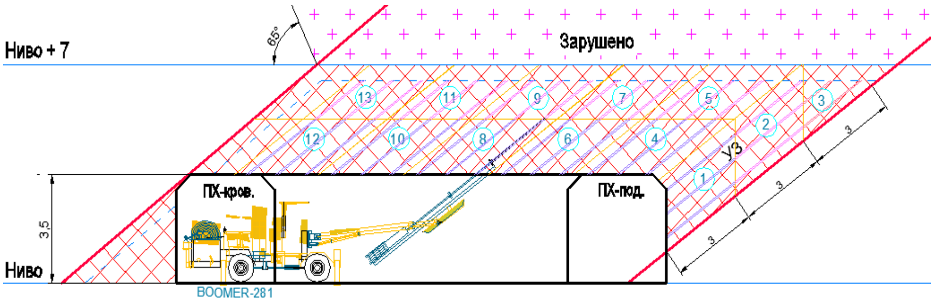

Figure 2 *The ore blasting schedule in applying the sublevel caving method for excavation the ore mass, thickness of over 10 m*

The further operations schedule of excavation is as follows:

1. With ore masses over 6.0 m thick, after construction the first excavation drift (OH), a floor sublevel drift is firstly excavated (PH), immediately next to the safety pillars (SS). This where special caution must be taken during the contour blasting in the CC zone, so as to properly form CC and avoid its weakening.
2. The second excavation drift is constructed OH, on an axial distance of 6.0 m which forms the excavation pillar (OS);
3. The ore gained through the construction of the raise for kerf (RK) is loaded and transported;
4. Drilling a fan of mine drill holes is done in the sublevel plate and the excavation pillar (OS), up to half of its width. It is recommended that in stope all of the excavation drifts (OH) be finished, and then to start the ore blasting in the sublevel floor, in retreat;
5. Mine holes are filled with explosives, according to the platform rule, for safety purposes;
6. The ore is mined-blasted;
7. The stope is ventilated;
8. Mine holes are drilled in the sublevel plate, the second half of the excavation pillar (OS). The width of the stope, in this case is b=6.0 m;
9. The ore is mined-blasted;
10. The stope is ventilated.

This is the manner in which the excavation of all stopes will be done in the individual mining block.

The demolition of roof rocks will be spontaneous, as the excavation progresses, from the roof towards the floor, and the angle under which the drilling and blasting is done does not allow large depletions of ore, i.e. a significant breach of waste rock from the roof of stope. In case that spontaneous collapse of waste rock does not occur, the secondary blasting will be conducted, after the loading and transport of previously blasted ore from the sublevel plate and excavation pillar (OS). In these cases, the remote controlled loaders are used so as to prevent the endangerment of employees.

After the separation ventilation of the work site, loading and transport of ore is done using a loader powered by diesel fuels. During loading, the amount of the blasted ore must be calculated in order to avoid increased dilution of the ore substance. The ore is transported to the mineral loose, and is then transported gravitationally to the transport level H-830 from where it is transported using trolley locomotives and "Gremby" carts to the bunker of the export-service shaft "Golema reka", and beyond, across the central transport system to the surface [1,6].

TECHNO-ECONOMIC EXCAVATION PARAMETERS

Techno-economic excavation parameters via the sublevel caving method from roof to floor in the Mining District “Svinja reka” of the “Sasa” mine - Makedonska Kamenica between H-910 and H-830 are shown in Table 2 [1].

Table 2 *Techno-economic parameters of excavation in the trial block*

Amount of ore in block ($T_č$)	1,795,582 t
Amount of ore to be excavated ($T_č$)	1,382,732 t
Exploitation coefficient (i_r)	0.77 (77 %)
Ore loss (g_r)	0.23 (23 %)
Dilution of ore substance (expected) (O_r)	0.22 (20 %)
Stope capacity (for production of Q_g=150.000t/year)	140.85 t/shift/block
Required daily production (Q_{dp})	1,268 t/day
Required amount of blasting per shift (N_{sf})	1
Excavation performance	10.3 t/wage
Excavation intensity	5.13 m/year
Required number of workers per day	36
Workforce norm	0.083 wage/t

CONCLUSION

Having in mind the mining - geological ore and waste rock characteristics between H-910 and H-830 levels, as well as based on the experiences gained so far during the excavation of the higher parts of the Mining District “Svinja reka” (between levels XIVb-990 and H-990-910), as well as the additional techno economic analysis, the conclusion is that the excavation of ore between H-910 and H-830 is technologically justified in using the *sublevel caving mining method from the roof towards the floor,* with temporary placement of excavation pillars. One of the main shortcomings of this method of excavation is the necessity of placement the protective plate for the purpose of support of surrounding rocks, in order to secure a safe and proper operation on the stopes - sublevels. The slanted protection plate is defined with the purpose that the excavated area can be properly filled with the blasted waste rock, for the purpose of reducing the loss of ore substance. According to the estimations, the slanted protection plate reduces the loss up to 50% compared to the horizontal plate. Also, the stability of stopes in its vicinity is also increased. Although the ore left in the protection plate cannot be excavated, the techno-economic analysis has shown that these losses are completely justified.

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Vladan Čanović, Saša Mitić*, Aleksandar Đerisilo*, Jasmina Nešković**

SUDOP1 SOFTWARE PACKAGE IMPLEMENTATION IN THE DESIGN OF DEWATERING OBJECTS AT THE OPEN LIMESTONE PIT MUTALJ NEAR BEOČIN

Abstract

The open limestone pit Mutalj is located on the southern slopes of Fruška Gora in the northern part of Serbia. Immediately beside the open pit, the Grabovački stream flows which has in its upper flow two artificial accumulations built to protect the settlements located downstream from floods. In relation to the "Lajtovački Limestone" that is in exploitation, a unique karst aquifer was formed and is hydraulically connected with the Grabovac stream and the artificial accumulations. Successful protection of the open pit of Mutalj from being flooded by surface and underground water was accomplished through designing the drainage facilities of proper dimensions that encompasses the floor canals, sumps, pumping station and a settling pond for which the SUDOP1 software package was used. The aforementioned program enables not only better designed objects of surface dewatering, but also provides a checkup of hydraulic functionality the existing drainage objects at the open pit.

Keywords: *open pit, surface dewatering, software package, sump, canal*

INTRODUCTION

At the open pit "Mutalj" which is a part of the Lafarge company - cement factory of Beočin (LBFC), limestone is exploited which together with the marl from the open pit "Filijala" makes the basis of mixture for cement clinker production.

In the area of the open pit "Mutalj, the limestone is exploited that has a high content of $CaCO_3$ which is over 96% and known in literature as the "Lajtovacki" limestone". Limestone thickness increases from the north to the south part of the basin. From a hydro-geological aspect these limestones present the largest collectors of underground water in the basin. A karst aquifer was formed in them which is fed from the atmospheric showers that infiltrate through the overlying permeable sediments, made out of loess and clayed graveled sands, as well as by the flow of temporary and constant surface waters which pour from northwest down the southern slopes of Fruška Gora, forming the Grabovac stream. This stream flows along the west ridge of the open pit "Mutalj" that was regulated by two dams in the mid XX century due to its flooding tendency.

At a part of the stream which passes near the open pit "Mutalj", the stream bed was buried until May 2014 and did not provide efficient and timely dewatering of the excess atmospheric water. Only after the catastrophic rain falls and flooding in May 2014 was the stream bed of the Grabovac stream near the open pit "Mutalj" finally dug up and cleaned.

The karst aquifer that was formed in the reef limestones is drained in the southwest

* *Mining Institute, Belgrade, Serbia, vladan.canovic@ribeograd.ac.rs*

direction, i.e. in the direction that the floor sediments made out of bentonite clay, basalt conglomerates and lake sediments drop and along with some presentations of coal make up the "Vrdnička serija".

The exploitation of limestone has in the past couple of years dropped beneath the level of ground water that was determined through hydro-geological exploration conducted in 2004 and was at the level of around 175 meters altitude. The present lowest level of limestone exploitation in the pit is around 157 meters altitude which is followed by the lowering the level of ground water as a consequence of continuous pumping of water from the floor canals and the main sump and are directed downstream in the Grabovac stream.

DEWATERING CONCEPT

Dewatering of the OP Mutalj is continuously done from June 2008 until today. The current dewatering is done through a system of deep drainage canals out of which one is the main water collector into which four transverse drainage canals lead. The canal depth goes from 3-5 m, width is 1.7 m, and total length is 756 m. The main water collection canal goes into a sump that is 20 x 10 m and 4 m deep which is located at the level of 157 meters altitude. Two sub-mersible pums are placed in the sump: Flygt 2670.180 18 KW and Flygt 2640.180 5.6 KW. The gathered water from the open pit Mutalj are then pumped via thrust pipelines Ø 105 mm into the Grabovac stream. The equipment required for dewatering consists of: pumps, pipelines, distribution boxes, pump powering cables, level-regulators, pump holder, etc.

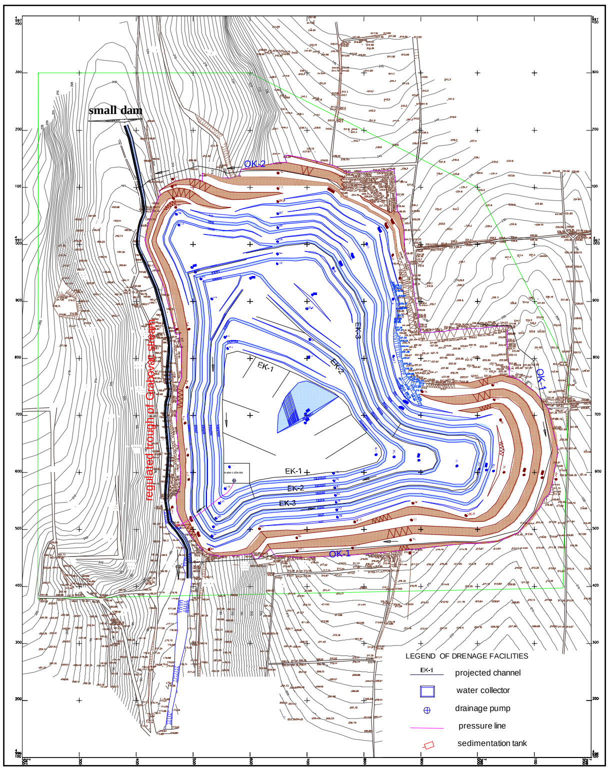

Figure 1 *Situation map of designed dewatering system at the OP Mutalj*

The adopted concept of dewatering and protection of the open pit Mutalj from water to the end of designed period of limestone exploitation consists of the following (Figure 1):

- Dewatering will be done through a system of floor canals which are designed to take the maximum influx of surface and outpoured underground water. This water will be gravitationally directed to the main sump which is always located at the lowest point of the pit. At the main sump on a pontoon, a pump will be installed of appropriate characteristics which will pump the gathered water to the settling tank located outside of the pit contour from where the water will be poured in the Grabovac stream.
- It is necessary to conduct a hydro-construction regulation of the Grabovac stream and build two ridge canals from the east and north side of the pit.

DESIGNING THE DEWATERING FACILITIES

SUDOP1 (Surface Dewatering of Open Pit) software was used to design the facilities of surface dewatering at the OP Mutalj. This program package contains modules for designing of canal, sumps, pumping stations, gravity pipelines and settling tanks and can be used for designing on the new or existing open pits of coal, metallic or non-metallic mineral ore. At the open pits that are exploited, as is the case with OP Mutalj, this program package offers the possibility of verification the existing facilities of dewatering or design the new ones in a function of development the mining operations. The program SUDOP1 was written in the program language Visual Basic 6.0.

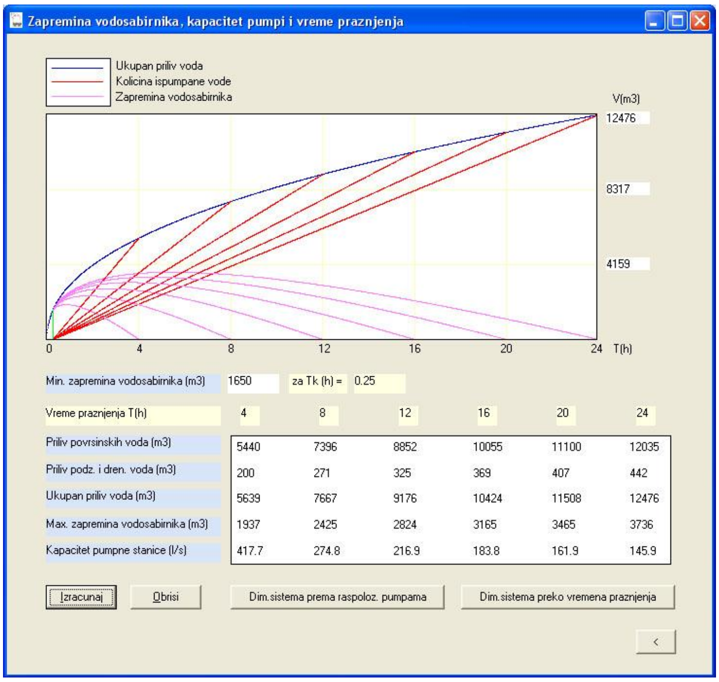

Figure 2 *Dependence the water influx, drainage capacity and sump volume*

The main sump and drainage pump are designed as a single system at the open pit Mutalj. The volume of the main sump is directly dependent on the characteristics of drainage pump. The design of entire system was done based on a rational method, using the methodology of Z. Ljubić, whose basics are given in the book "Dewatering Facilities at the Open Pits" by Prof. Dr Radomir Simić and M.Sc. Vladislav Kecojević (Faculty of Mining and Geology Belgrade, 1997). This methodology of calculation was applied in constructing the SUDOP1 program. Based on the formulae for calculating the maximum sump volume V_{max} , it can be concluded that the volume of sump as well as the time it takes to empty it depend on the capacity of drainage pump, the pump delay time and the rainfall intensity. Dimensions of the main sump and the drainage pump at the OP Mutalj are determined by taking into consideration the influx of surface and underground water and the time it takes to empty it (Figure 2). For the purpose of designing the bench channels, the sump and drainage pump, precipitation of 50 year return period were considered with 1 h of lasting intensity, i.e. i = 51.9 mm/h. This information was obtained from the Republic Hydro Meteorological Department of the Republic of Serbia and originates from the main meteorological station of Sremska Mitrovica which is closest to the OP Mutalj. In order to design the dewatering facilities, the entrance data about the influx of underground water into the pit of ~ 30 l/s was used. This data was obtained based on the hydro-geological exploration realized in the zone of OP Mutalj in the period from January 2012 to November 2013.

Table 1 displays the program calculated values of maximum volumes of the sump (V_{max}) and the drainage pump capacity (Q_p) for determined emptying time of 120 h for each designed year of exploitation.

Table 1

Year of exploitation	I	II	III	IV	V	Final
Max. sump volume V_{max} (m^3)	6697	6983	6983	7000	7080	7459
Drainage pump capacity Q_p (l/s)	53.0	55.3	55.3	55.4	56.0	59.0

Hydraulic calculation the permeability of floor and ridge canals of a trapezoid cross section at the OP Mutalj was conducted based on the Chezy-Manning equation which was applied in writing the SUDOP1 program. The program module for canal design which enables determination the optimal canal dimensions depending on hydrological and hydraulic parameters input consists out of five interconnected parts. In the first part of the program module, hydrological data is input, the second part of the program calculates the optimal proportions between the width of canal bottom, the height of canal and the surface of cross section of the canal. In the third part of the program, the material for canal construction is chosen, and the fourth part of the program enables the parameters of the canal lane to be defined, and the fifth and final part of the program conducts a graphic interpretation, i.e. determines the dependence of canal depth and throughput at certain sections of the canal lane. This paper displays only the calculation for the ridge canal RC-2. Input parameters for design of the ridge canal RC-2 at the OP Mutalj are shown in Table 2.

Calculated dimensions of the ridge canal RC-2 are shown in Table 3, where J- is the drop of canal lane (%), L-is the length of canal lane (m), H-is the depth of canal (m), B-is the width of canal bottom (m), R-is the hydraulic radius (m), U-is the wet circumference (m), C-is the coefficient which depends on material that insulates the canal, F-is the surface of canal cross section (m^2), V-is the speed at which water runs in the canal (m/s), V_k-volume of the material that is excavated (m^3). Calculated maximum throughput of the ridge canal RC-2 of the full profile is $Q_k = 2.4\ m^3/s$ and was increased by 20% for security reasons. Figure 3 shows a curve which depicts the dependence of depth change (H) and throughput of the canal (Q_k) while the blue lines on abscissa and ordinate of diagram mark the calculated values of the aforementioned parameters for the input ridge canal RC-2 lane drop.

Table 2

Parameter	Unit	Value
Runoff coefficient	-	0.35
Rainfall height	(mm)	58.2
Rainfall duration	(min)	60
Value of flank angle of canal slope	(°)	40
The roughness coefficient according to Basin	-	1.75
Mid drop of canal floor	(%)	0.48
Influx of underground and infiltration water	(l/s)	30
Size of drainage area	(m^2)	350000
Canal length	(m)	176

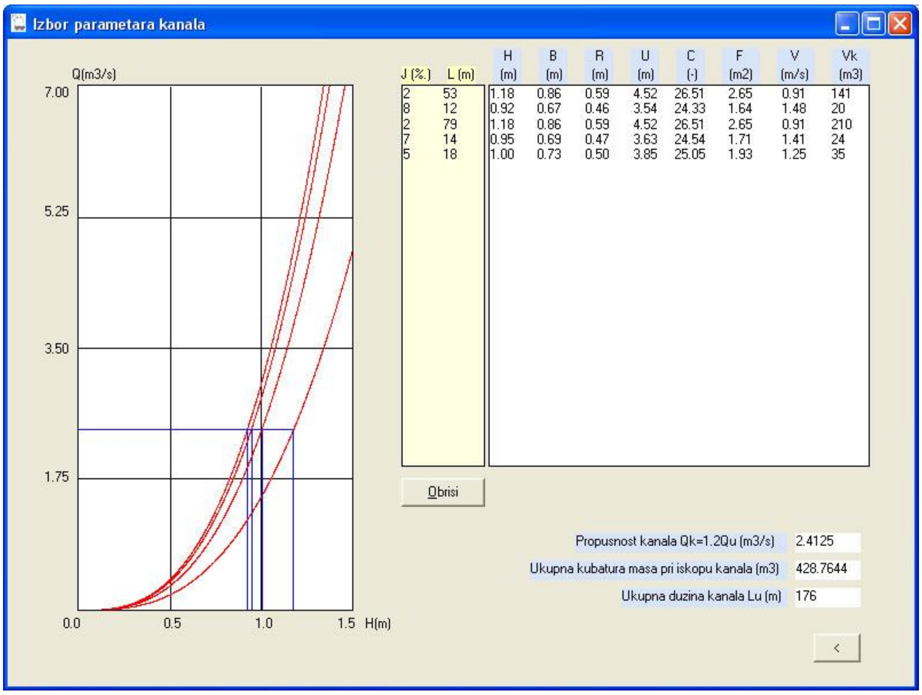

Figure 3 *Dependence diagram of throughput (Q_k) and depth (H) of the ridge canal RC-2*

Table 3

Lane number	J	L	H	B	R	U	C	F	V	Vk
	(%)	(m)	(m)	(m)	(m)	(m)	(-)	(m^2)	(m/s)	(m^3)
1	0.2	53	1.18	0.86	0.59	4.52	26.51	2.65	0.91	141
2	0.8	12	0.92	0.67	0.46	3.54	24.33	1.64	1.48	20
3	0.2	79	1.18	0.86	0.59	4.52	26.51	2.65	0.91	210
4	0.7	14	0.95	0.69	0.47	3.63	24.54	1.71	1.41	24
5	0.5	18	1.0	0.73	0.50	3.85	25.05	1.93	1.25	35

CONCLUSION

Protection of open pits from water influx in the mining industry becomes a great problem in the designing and exploitation stage. The successful designing of drainage systems at the open pits depends on the knowledge about the hydrological, hydro-geological and geomechanical characteristics of immediate and extended zone of the basin. The open pit Mutalj is mostly compromised by the influx of surface water from alluvium of the Grabovac stream and influx of ground water from the "Lajtovacki Limestone” that is in exploitation. The project solution predicts the hydro-construction regulation of the stream bed of the Grabovac stream and construction the ridge and floor canals, sumps, a pumping station and a water tank as the main protection facilities of the OP Mutalj from water. For designing the aforementioned drainage facilities, the software SUDOP1 was used. It consists of programs for designing the gravitational pipelines, canals, sumps and water tanks and pumping station calculations. The SUDOP1 program can also be used for verifying the hydraulic functionality of existing facilities for dewatering and equipment requisition planning.

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*Nedeljko Magdalinović**

TRUTHS AND MISCONCEPTIONS IN THE ORE COMMINUTION

Abstract

Inventions and now widely applied machines for ore comminution dated back to the second half of the 19th century. For the past 150 years, much theoretical knowledge was accumulated in the field of physics the solid bodies, physics of fracture, the theory of shock, etc., but they did not significantly affect the development of machines and technologies of comminution. Their development had a highlighted the empirical flow in the direction of increasing the unit sizes and capacities, while the main deformation mechanisms and technological principles remained unchanged. Development on practical experiences and insufficient physical justified hypothesis of the solid state physics inevitably had to result in some misconceptions that, despite recent theoretical knowledge and proven facts, continue to exist in practice. This paper just wants to point out some of these misconceptions.

Keywords: *comminution process, ore, comminution model*

THEORETICAL AND REAL COMMINUTION ENERGY

The most expensive stage in mineral processing is comminution (crushing and grinding), due to the high consumption of energy and metals. Due to the extreme complexity of structure and texture of ores, it is still not mathematically possible to model both an individual act of grinding and collective comminution a mixture of grains by crushing and grinding.

The Griffith model of stretching thin elastic plate (Figure 1), borrowed from the solid state physics, provides only a qualitative answer to the question why the real strength of minerals σr for 10^2-10^3 times if lower than the theoretical strength σ_T [2.5.] This fact is explained by the present defects and microcracks in the crystal lattice of minerals. Under the effect of external tensile strength, the incurred or already present microcracks grow and when they reach a certain "critical" length further expand by themselves (without further bringing the energy from the field), come to the surface, thus resulting into communition. Spontaneous expansion the microcracks of "critical" lengths occur because the strain of matter at the top of microcrack is several times higher than the average strain in other parts of grain (Figure 1).

The ores are polymineral inhomogeneous anisotropic materials, composed of many different mineral grains grown into each other. The strength of certain minerals greatly varies as well as the strength of binder with all possible present micro and macro cracks and defects in them. Hence, such complex system, for now, is still impossible to be mathematically modeled.

The Griffith model of comminution did not lead to redesigning the classic machines for comminution, but pointed to the possible directions of development the comminution technology. Namely, the formation of microcracks in grains may

* *19210 Bor, Timočke divizije 18/12, e-mail:magdalinovicnedeljko@gmail.com*

result, not only by the external mechanical forces, but also by heating, passing a flow of high stress, acting the electromagnetic waves and ultrasound. Although these studies are still only in the laboratory, they indicated that in the perspective of such treatment before the comminution of raw materials can expect a better release of minerals and, in general, lower power consumption.

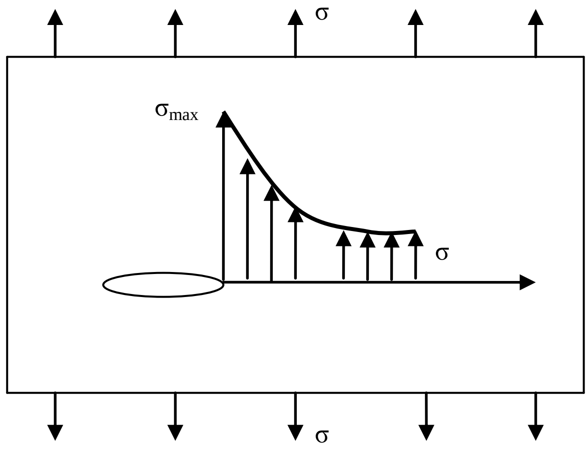

Figure 1 *Stress field of microcrack in a thin elastic plate*

Despite the fact that the real strength of minerals is lower 10^2-10^3 times than the theoretical strength, the energy needed for collective comminution of grain is about 10^3 times greater than the theoretical strength. It follows therefore, that the use of energy in comminution is very small, only 0.1 % [5]:

$$I_E = (\sigma_t/E)\ 100 = (\sigma_t/10^3\ \sigma_t)10^2 = 0.1\ \%$$

These theoretical considerations and conclusions confirm the results of measurements of industrial practice.

Inevitably the question arises where the most of the energy is consumed invested in comminution?

Measurements on an industrial mill for cement grinding have showed the following structure of energy distribution (Table 1):

Table 1 *Energy distribution in cement grinding* [2]

	%
Energy of newly created surfaces	0.6
Energy losses in the transmission system	12.3
Heat losses - on cylinder heating…................................ 6.4 - on raw materials heating...........................47.6 - pm environment heating............................31.0	85.0
Other losses	2.1
Total	100.0

Therefore, the most of the energy (85%) is transformed and lost in the form of heat energy, and only 0.6% of the energy is spent on formation the new areas. If the energy losses are excluded in the transmission system and other minor losses, it is concluded that from the energy directly invested in grinding, 0.7% is spent on formation the new areas, and even 99.3% is transformed and lost in the form of heat energy.

Large losses of energy through thermal effects are easily detectable in practice. Cement warms up to 120°C during grinding. The pulp at the exit of the mill has (10-20)°C higher temperature than at the entrance to the mill.

Although an undeniable fact of great energy losses through thermal effects in comminution has long been known, a lot of research and conclusions, based on them, often neglect this fact.

CAUSES OF THERMAL ENERGY LOSSES IN COMMINUTION

Thermal energy losses in comminution are the results of a large and intense friction between the individual grains and pieces of ore and their mutual friction with the working surfaces of grinding machine. Experiments of comminution the individual grains and grain comminution in a thin layer by removal of the comminution product after each act of comminution, which reduces friction to a minimum, indicate a significant increase in energy efficiency. These studies have shown that the energy losses to friction depend on the size of surface that participates in this process, the intensity and duration of friction. From these findings, the conclusion is clearly derived that the increase of energy efficiency in the comminution process can be achieved by reduction the contact surfaces, intensity and contact time of grains and piece with each other grains in the workspace of grinding machines.

Starting from these findings, Revnivcev [5] invoked the relationship of time of grain stay in the workspace of the grinding machine $\mathbf{t_2}$ and time of grinding act $\mathbf{t_1}$ as a qualitative criterion for assessing the energy efficiency of mechanical grinding machines and called it the "excess time coefficient" $\mathbf{k_t}$:

$$k_t = t_2/t_1 \qquad (1)$$

In classical grinding machine, $\mathbf{t_2}$ is several times larger than $\mathbf{t_1}$. Grinding machine is more energy efficient if the value of coefficient $\mathbf{t_1}$. is lower.

If **V** is the volume of raw materials in the workspace of grinding machine, and **Q** is the volume capacity of the machine, then the time of grain stay in the workspace of machine $\mathbf{t_2}$ is:

$$t_2 = V/Q \qquad (2)$$

If $\boldsymbol{V_m}$ is the expansion rate of the microcrack, after reaching the "critical" length, and **d** is the largest dimension of grain, then the time $\mathbf{t_1}$ of comminution act is:

$$t_1 = d/V_m \qquad (3)$$

It follows that the "coefficient of excess time"$\mathbf{k_t}$ is:

$$k_t = (V_m/d)(V/Q) \qquad (4)$$

The last expression clearly points to the conclusion that the crushing machine is energy efficient if it has a lower ratio of **V/Q**, which directly means a shorter time of stay tof raw material in the workspace of machine, and thus less friction time.

Revnivcev, in the case of quartz crushing, size **d** = 20 mm, the shape of cube, in the laboratory cone crusher, came to the conclusion that $\mathbf{k_t}$ ~ 2100. In other words, the time $\mathbf{t_1}$ for which the coarsest grain was comminuted is 2100 times shorter than the time $\mathbf{t_2}$ required that the comminution product of this grain goes out of the crusher workspace.

In comparison with the value of $\mathbf{k_t}$ for cone crusher, calculated by Revnivcev, here $\mathbf{k_t}$ will be calculated for quartz crushing of same grain size **d** = 20 mm in the roller crusher:

In the roller crusher, only one grain is in the workspace for crushing (between the rollers in intersection) (Figure 2).

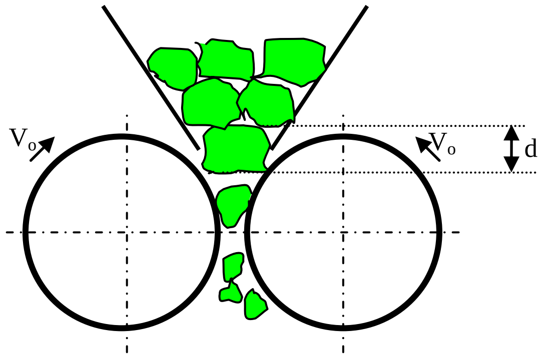

Figure 2 *Intersection of workspace of roller crusher*

The time of grain stay between the rollers $\mathbf{t_2}$ is equal to:

$t_2 = d/V_o = 0.02/5 = 0.004$ *s*

where

$V_o = 5$ *m/s* – circumferential velocity of rollers

Rate of communition act $\mathbf{t_1}$ is:

$t_1 = d/V_m = 0.02/1500 = 0.000013$ *s*

where

$V_m = 1500$ *m/s* – expansion rate of the critical length of microcracks in quartz

Coefficient of excess time $\mathbf{k_t}$ is:

$k_t = t_2/t_1 = 0.004/0.000013 = 308$

Therefore, the coefficient $\mathbf{k_t}$ in the roller crusher is about seven times smaller than that of cone crusher. If the hypothesis of Revnivcev that the "coefficient of excess time" $\mathbf{k_t}$ represents a qualitative measure of energy efficiency, then it means that the roller crusher should be significantly more energy efficient. The research results presented in Table 2 convincingly confirm this.

Table 2 *Energy utilization in some crushing machines*[5]

Machine	Raw material	Energy utilization,(%)
Cone crusher	Quartz	0.02
High speed impact crusher	Quartz	31.8
Roller crusher	Quartz	87.8
Ball mill	Cement	0.7

The presented theoretical analysis and research results clearly derive the conclusion that the reduction in friction compared to the same present in the classical machines for ore crushing (jaw and cone crushers and cylindrical mills) can be possible by redesign of their construction in the following directions:

- minimizing the relationship V/Q,
- increase in the frequency of stress (the act of grinding),
- minimizing the stroke of working parts of the machine as well as their speed in regard to the raw material grains, and
- faster evacuation of the finished product from the workspace of machine.

For now, these principles are largely achieved with the roller crushers.

THE RITTINGER LAW OF COMMUNITION

Communition reduces the coarseness of raw materials, as a direct consequence of its surface increase. Since the comminution process is formation the new surfaces, the literature argues that the Rittinger law of communition is only physically based because it starts from the hypothesis that the spent energy for comminution is directly proportional to the newly created syrface [4]. And that, as it was seen in previous chapters, is simply not true. In classical machines for ore comminution (crushers and mills, except crushers with rollers) a negligibly small part of energy is spent for formation the new surfaces, the order of 1%, while about 99% of directly consumed energy in commi-nution is lost in a form of heat effects, as a result of friction between the grains themselves and between the grains and the workspaces of grinding machine.

A lot of research was devoted in proving the Rittinger hypothesis in the field of grinding, especially a very fine grinding. Many works were published, even after learning that the most of energy in grinding is not spent on formation the new surfaces, but they are transformed into heat energy. This fact definitely refutes the Rittinger hypothesis and every possible proof of it is the fallacy.

CHARGING THE WORKSPACE OF CRUSHER AND ENERGY EFFICIENCY

It is usually believed in practice that the workspace of crusher should be backfilled with ore. Highlighting of such requirement is primarily associated with the uniform wear of linings, but also with the fact that a certain degree of cominution is realized in all zones vertically of the crusher workspace. To what extent is this energy beneficial is shown by the theoretical analysis derived for jaw crusher with the help of Figure 3a.

During the working stroke of movable jaw (half turnaround of eccentric), the pieces are ground a cross-sectional view of workspace is formed that is shown in Figure 3 [1,3]. During the idle (second half turnaround of eccentric), theoretically the final product is empieid from the workspace that is below the level 2-2. During this time, this space is charged with pieces that are between the levels 2-2 and 3-3, while in their place the pieces between the levels 3-3 and 4-4 come, and the cycle is repeated.

From this theoretical analysis of one cycle of crushing and discharinge the finished product, according to the criterion of "excess time coefficient" $\mathbf{k_t}$, it could be concluded that charging the workspace above the level of 4-4 is not desirable because it increases the coefficient $\mathbf{k_t}$, and thus adversely affect the energy efficiency.

Height $\mathbf{h_o}$ to what the workspace of crusher has to be charged can be called the optimal height, and it is:

$$h_o = h_1 + h_2 + h_3 \quad (5)$$

Angle $\boldsymbol{\alpha_1}$, when the workspaces of crusher are in the farthest position, is slightly higher than the gripping angle $\boldsymbol{\alpha}$, and further analysis will be carried out with the gripping angle $\boldsymbol{\alpha}$.

$$h_1 = s/tg\ \alpha \quad (6)$$

$$h_2 = a/tg\ \alpha \quad (7)$$

$$h_3 = a_1/tg\ \alpha \quad (8)$$

$$S/s = (S-l)/a \ \rightarrow a = s\ (1-l/S) \quad (9)$$

$$l = s/sin\ \alpha \quad (10)$$

$$(S-l)/a = (S-l-l_1)/a_1 \ \rightarrow a_1 = = a\ (S-l-l_1)/(S-l) \quad (11)$$

$$l_1 = a/sin\ \alpha \quad (12)$$

where:

α – gripping angle

s – stroke of movable jaw at the level of discharge opening (mm)

S - length of movable jaw (mm)

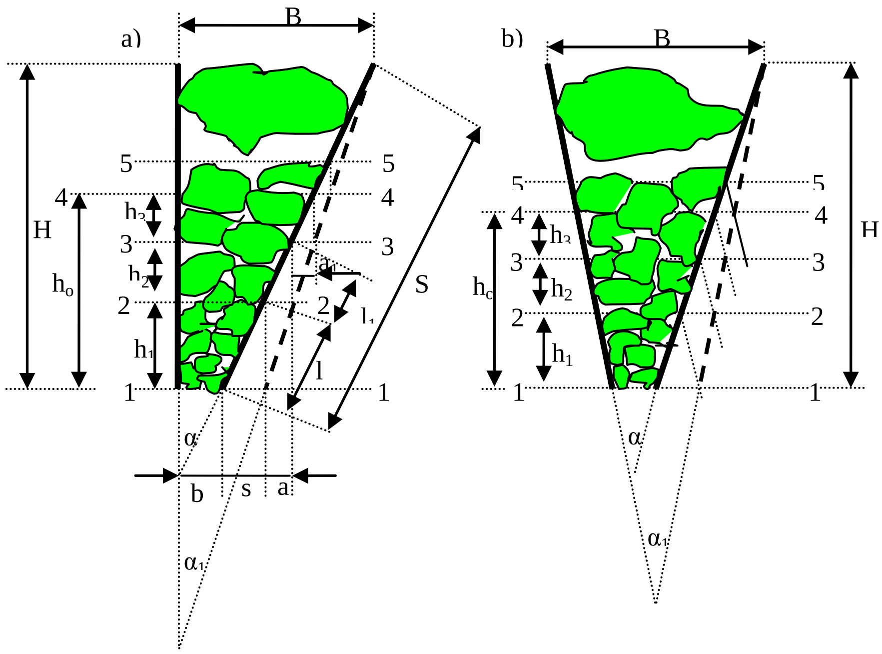

Figure 3 *Schematic view the workspace of jaw (a) and cone crusher (b)*

Relationship of optimum height $\mathbf{h_o}$ of charging the workspace and total height **H** of workspace will be denoted by the coefficient $\mathbf{k_H}$:

$$k_H = h_o/H \qquad (13)$$

Relationship of optimum volume $\mathbf{V_o}$ of charging the workspace and total volume **V** of workspace $\mathbf{k_V}$ will be named by the coefficient of charging the workspace:

$$k_V = k_H(2b + x)/(B + b) \qquad (14)$$

where:

$x = h_o\, tg\, \alpha$

Now let's see how much is the optimal charging height of crusher $\mathbf{h_o}$ and the coefficient of charging the workspace of crusher $\mathbf{k_V}$ in the example of the smallest C95 (index 1) and the largest C200 (index 2) jaw crusher from the catalog of the company Metso.

	Jaw crusher C95	***Cone crusher C200***
- width of inlet opening	B_1 = 580 mm	B_2 = 1500 mm, X =2.6
- length of inlet opening	L_1 = 930 mm	L_2 = 2000 mm
- height of workspace	H_1 = 755 mm	H_2 = 2800 mm, Y =3.7
- length of movable jaw	S_1 = 810 mm	S_2 = 2990 mm
- min. width of discharge opening	b = 60-175 mm	b = 175-300 mm
- gripping angle	$\alpha = 23^o$	$\alpha = 23^o$
- stroke of movable jaw at the level of discharge opening	s_1 = 30-85 mm	s_2 = 85-150 mm
$h_o = h_1 + h_2 + h_3$ [eq. (6)-(12)]	216-495 mm	623-1033 mm
$k_h = h_o/H$ (eq. 13)	0.29-0.65	0.22-0.37
$k_v = V_o/V$ (eq. 14)	0.09-0.46	0.08-0.20

Let's analyze what could be gained in the energy efficiency of crusher if its workspace is charged with raw material only to the height $\mathbf{h_o}$.

Let's in this workspace at full capacity of crusher the ore volume therein **V,** and the coefficient of excess time $\mathbf{k_t}$**,** according to equation (4):

$k_t = (V_m/d)(V/Q)$

Charging the workspace of crusher only to the height $\mathbf{h_o}$ has reduced the volume of ore in the workspace to $\mathbf{k_V V}$**.** Then, if the coefficient of excess time $\mathbf{k_{t1}}$ will be:

$k_{t1} = (V_m/d)(k_V\, V/Q)$

The quotient $\mathbf{k_{t1}/k_t}$ provides a qualitative change in energy efficiency, that is:

$$k_{t1}/k_t = [(V_m/d)(k_V V/Q)] / [(V_m/d)(V/Q)] \quad (15)$$

respectively:

$$k_{t1} = k_V k_t \quad (16)$$

Since $\mathbf{k_V}$ is less than one, than $\mathbf{k_{t1}}$ will always be less than $\mathbf{k_t}$**.** For the smallest crusher C95 is $\boldsymbol{k_{t1}}$ = (0.09-0.46) $\boldsymbol{k_t}$, and for the largest C200 is $\boldsymbol{k_{t1}}$ = (0.08-0.20) $\boldsymbol{k_t}$, depending on the size of minimum discharge opening of crusher.

Therefore, charging the workspace of crusher to the optimum height $\mathbf{h_o}$, the coefficient excess time $\mathbf{k_{t1}}$ is greatly reduced (in the smallest crusher C95 for 91-54%, and the largest crusher C200 for 92-80%). This will ultimately result in significantly higher energy efficiency of the crusher. In the same crusher, the coefficient $\boldsymbol{k_t}$ increases and energy efficiency reduces with a reduction in the crushing degree. A similar analysis leads to the same conclusion for cone crushers (Figure 3b).

Therefore, considering the importance of energy efficiency, the workspaces of crusher above the optimal height ho do not need to be charged with the raw material. Does this mean that the overall height of the workspace of crusher ***H*** could be reduced? No, due to a possible occasional arrival of large pieces of ore (Figure 4).

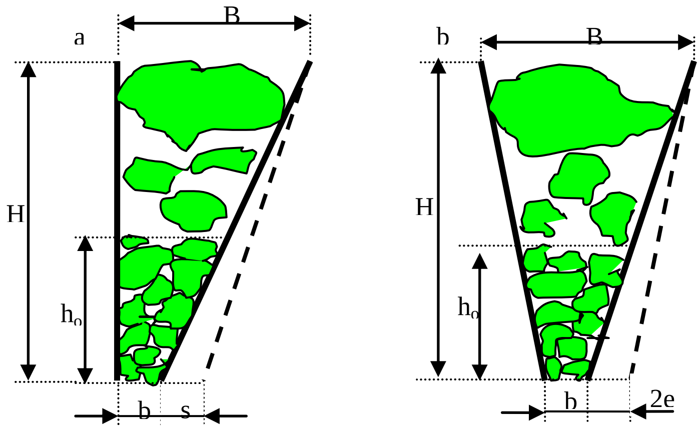

Figure 4 *Charging the workspace of crusher to the optimum height h_o*

Charging of crusher workspace only to the height $\mathbf{h_o}$ will lead to uneven wear of linings along the height of workspace. This problem is constructive workable as the linings along the the height would be of two or three parts. The bottom lining below the level $\mathbf{h_o}$ will be worn the fastest and they would be replaced more frequently.

CAPACITY INCREASE AND ENERGY EFFICIENCY OF GRINDING MACHINES

Inventions and nowadays widely applied machines for ore grinding dated back to the second half of the 19th century. For the past 150 years, development of grinding machines and technologies has the accentuated the empirical flow in the direction of increasing the unit size and capacity, while the main deformation mechanisms and technological principles remain unchanged. Therefore, it is interesting to accompany how this development affected the energy efficiency of machines for comminution?

Jaw and Cone Crushers

The analysis is carried out with two jaw crushers of different sizes, and with the same discharge opening and gripping angle. The sizes of the first small crusher have the index 1, and the sizes of the second larger crusher have the index 2. Two variants will be analyzed. In the first variant, both crushers work with a completely charged workspace **V,** and in the second variant with optimum charged workspace filled $\mathbf{V_o}$.

The analysis is based on the "excess time coefficient" $\mathbf{k_t}$ of Revnivcev, given by equation (4). To simplify the analysis, let's consider the crushing of a raw material of constant physical-mechanical characteristics ($\mathbf{V_m}$ = const.) and input of grain size (**d** = const.) in both crushers. Then, the equation (4) can be written as:

$$k_t = cV / Q \qquad (17)$$

where:

$$c = V_m / d$$

	Jaw crusher 1	***Cone crusher 2***
- width of inlet opening	B_1	$B_2 = XB_1$, $X>1$
- height of workspace	H_1	$H_2 = YH_1$, $Y>1$
- width of workspace	L_1	$L_2 = ZL_1$, $Z>1$
- min. width of discharge opening	b	b
* Volume capacity	$Q_1 = k \times L_1 \times b$	$Q_2 = k \times ZL_1 \times b$
* Volume of raw material in a full workspace	$V_1 = 0.5(B_1+b)H_1 \times L_1$	$V_2 = 0.5(XB_1+b)YH_1 \times ZL_1$
* Optimum volume of raw material in workspace	$V_{o1} = 0.5k_{V1}(B_1+b)H_1 \times L_1$	$V_{o2} = 0.5k_{V2}(XB_1+b)YH_1 \times ZL_1$
* Coefficient $k_t = cV/Q$	$k_{t1} = [0.5c(B_1+b)H_1]/kb$	$k_{t2} = [0.5c(XB_1+b)YH_1]/kb$
* Coefficient $k_t = cV_o/Q$	$k_{t1} = [0.5k_{V1}\,c(B_1+b)H_1]/kb$	$k_{t2} = [0.5k_{V2}\,c(XB_1+b)YH_1]/kb$

Variant with charged workspace of crusher:

The quotient $\mathbf{k_{t2}/k_{t1}}$ provides the qualitative insight into the change of energy efficiency:

$$k_{t2}/k_{t1} = [Y(XB_1+b)]/(B_1+b) > 1 \qquad (18)$$

which implies that:

$$k_{t2} = k_{t1}\{[Y(XB_1+b)]/(B_1+b)\} \qquad (19)$$

Variant with optimum charging of workspace of crusher:

$$k_{t2}/k_{t1} = [k_{V2}Y(XB_1+b)]/k_{V1}\,(B_1+b) > 1 \qquad (20)$$

which implies that:

$$k_{t2} = k_{t1}\{[k_{V2}Y(XB_1+b)]/k_{V1}\,(B_1+b)\} \qquad (21)$$

Let's illustrate this with a numerical example for already analyzed jaw crushers C95 and C200.

Variant with charged workspace of crusher:

The quotient $\mathbf{k_{t2}/k_{t1}}$ provides the qualitative insight into the change of energy efficiency:

k_{t2}/k_{t1} =[3.7(2.6x580+175)]/(580+175) = 8.2

which implies that:

$k_{t2} = 8.2\ k_{t1}$

Variant with optimum charging of workspace of crusher:

k_{t2}/k_{t1} = [0.08x3,7(2.6580+175)]/

/0.46 (580+175) = 1.44

which implies that:

$k_{t2} = 1.44 k_{t1}$

Thus, the derived analysis unequivocally shows that with increase of the jaw crusher capacity as a result of their increasing sizes of the workspace, the coefficient $\mathbf{k_t}$ increases. This increase is multi evident in the full crusher workspace (8.2 times), while the optimal charging of workspace is significantly lower and amounted to 1.44 times. Both leads to the conclusion that with increase of the jaw crusher capacity as a result of their increasing sizes of the workspace, the energy efficiency decreases. The same conclusion can be arrived also in the cone crushers. This is in direct contradiction with the current believes in practice and the assertions of the crusher manufacturers.

According to the energy unefficiency, the short cone crushers are particularly emphasized for the third stage of crushing, both due to the significantly higher coefficient $\mathbf{k_t}$ **,** and extremely large contact surface of friction between the grains.

Cylindrical Mills with Rods and Balls

The analysis for cylindrical mills with rods and balls is based on the "excess time coefficient" $\mathbf{k_t}$, identically as in the crushers.

The analysis is carried out for two mills of various sizes. The sizes of the first mill have the index 1, and the sizes of the second larger mill have the index 2.

	Mill 1	***Mill 2***
- mill diameter	D_1	$D_2 = XD_1$, X>1
- mill length	L_1	$L_2 = YL_1$, Y>1
- coefficient of raw material charging	s	s
* Volume capacity	$Q_1 = k\ D_1^{2.5}\ L_1$	$Q_2 = k\ (XD_1)^{2.5}\ YL_1$
* Raw material volume in mill	$V_1 = 0.25\ s\pi\ D_1^2\ L_1$	$V_2 = 0.25\ s\pi\ (XD_1)^2\ YL_1$
* Coefficient k_t = cV/Q	$k_{t1} = 0.25cs\pi/\ k\ D_1^{0.5}$	$k_{t2} = 0.25cs\pi/\ kX^{0.5}\ D_1^{0.5}$

The quotient $\mathbf{k_{t2}/k_{t1}}$ provides the qualitative insight into the change of energy efficiency:

$$k_{t2}/k_{t1} = 1/X^{0.5} < 1 \quad (22)$$

which implies that:

$$k_{t2} = k_{t1}/X^{0.5} \quad (23)$$

Since it is always $\boldsymbol{X > 1}$, the analysis shows that the increasing capacity of the mills as a result of their increasing sizes, the coefficient $\mathbf{k_t}$ decreases, which indirectly means that the energy efficiency increases. Also, it is noticed that the length of the mill **L** has no effect on the energy efficiency, but the impact has only the mill diameter D.

Let'S illustrate this by the numerical example for two ball mills from the catalog of the company Metso. The first smaller mill has a diameter D_1 = 4.0 m, and the second larger D_2 = 5.5 m (X = 5.5/4.0 = 1.37). It is understood that they work under the same technological conditions. The quotient

$\boldsymbol{k_{t2}/k_{t1}}$ provides a qualitative insight into the change of energy efficiency:

$$k_{t2}/k_{t1} = 1/X^{0,5} = 1/1.37^{0,5} = 0.85$$

which implies that:

$$k_{t2} = 0.85\ k_{t1}$$

CONCLUSION

For the past 150 years of invention the present machines for ore crushing, many theoretical knowledge has been accumulated in the field of physics of solids, physics of fracture, theory of shock, etc., but it did not significantly affect the development of machins and technologies of grinding. Their development had highlighted the empirical flow in the direction of increasing the unit sizes and capacities, while the main deformation mechanisms and technological principles remain unchanged.

From energy directly invested in comminution, only 1% is spent on formation the new surfaces and even 99% is transformed and loses in the form of heat. This fact definitely refutes the Rittinger hypothesis which argues that the spent energy on comminution is directly proportional to newly formed surface during grinding.

Thermal energy losses in comminution are the result of a large and intense friction between the grains themselves and pieces of ore and their mutual friction with the workspaces of grinding machines.

It is usually believed in practice that the workspace of crusher should be backfilled with ore. Highlighting of such requirement is primarily associated with the uniform wear of linings, but also with the fact that a certain degree of cominution is realized in all zones vertically of the crusher workspace. Theoretical analysis in this work shows that this is not energy favorable, but that the workspace of crusher has to be charged with raw material only to an optimum height.

With increasing capacity of crushers as the result of their increasing dimensions of workspace, the energy efficiency decreases. Decline in energy efficiency is much more pronounced in a completely charged workspace of crusher. This is in direct contradiction with the current believes in practice and assertions of the crusher manufacturers.

Unlike the crushers, the theoretical analysis shows that the growth capacity of cylindrical mills as a result of their increasing size, the energy efficiency increases.

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Violeta Čolaković, Dijana Vlajić*, Aleksandar Đerisilo*, Klara Konc Janković**

COMPLEX EXPLORATION OPERATIONS FOR THE PURPOSE OF MONITORING THE INFLUENCE OF THE FLYING AND BOTTOM ASH DEPOT SREDNJE KOSTOLAČKO OSTRVO ON THE ENVIRONMENT

Abstract

Based on the request by the Department of Environment Preservation, an order was given for the investment-technical documentation for implementation the remediation measures and to encompass the entire area of the flying and bottom ash depot on the location Srednje Kostolačko ostrvo. In order to define the project criterion for protection the ash depot, with the purpose of conducting the measures of remediation, it was necessary to make the Geotechnical Exploration Project. The Project is aimed to determine the exploration operations which will define the geological, engineering-geological and hydrogeological terrain characteristics. The designed exploration operations encompassed the field explorations and appropriate laboratory tests. The obtained results would enable to review the impact of the flying and bottom ash depot on the environment, and also to secure the relevant basis for further mining, construction, technological and all other projects at the locations in question.

Keywords: *exploration operations, ash depot, environmental impact*

1 INTRODUCTION

The ash depot of SKO is in the final exploitation stage and is going to be shut down in succession, one cassette at a time. The size of the flying and bottom ash depot of SKO and its geometry, upon cessation of operations, opens up the new possible usages for this area. The potential construction location requires permanent landscaping of this area which would ecologically be completely justified and acceptable. The problem of the SKO depot is made harder by its location between the Danube and the river Mlava, the impact of fluctuation of the surface and underground water levels, settlement proximity, climatic and meteorological factors, archeological culture monuments, etc.

The flying and bottom ash depot is located in the so-called Srednje Kostolačko ostrvo. From the north side, the depot is bordered by the defensive mound of the Danube, and from the east by a ridge mound placed in parallel with the regulated water flow of the river Mlava. On the south side, the depot is bordered by the tailing dump "Kipa Dunavac" and a ridge mound along the right bank of the Dunavac immediately next to the settlement Stari Kostolac, and on the west side, the depot is bordered by a the new brim channel of the cooling water, flowing into the river Danube.

In order to define the project criterion for the ash depot protection, it is necessary to conduct a series of geotechnical explorations to determine the geological-geotechnical, hydrogeological, geochemical, chemical, technological, pedological and other conditions at the actual location.

* *Mining Institute, Belgrade, e-mail: violeta.colakovic@ribeograd.ac.rs*

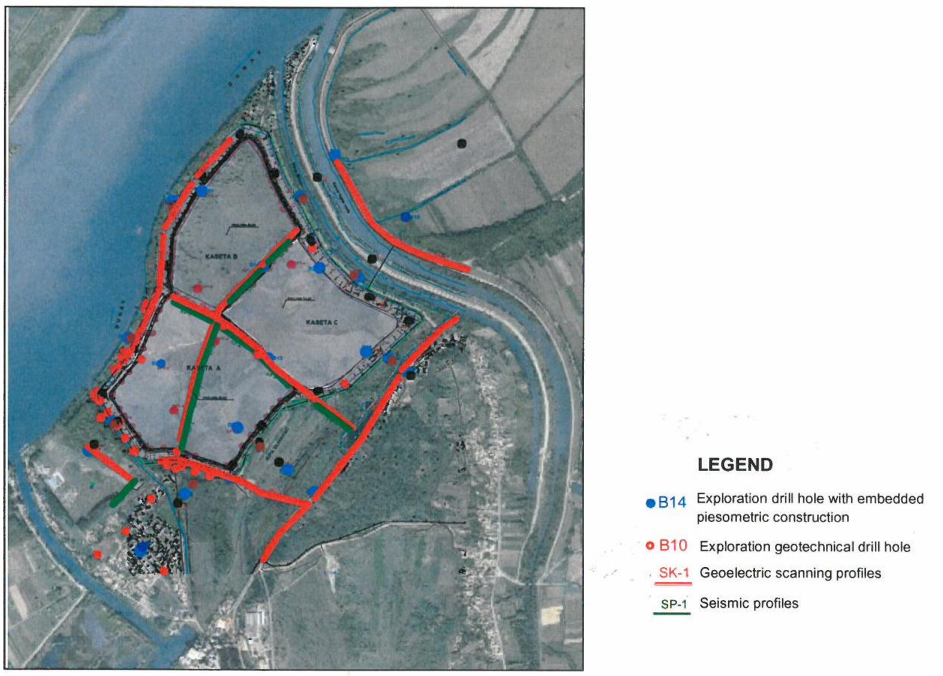

Figure 1 *Position of the SKO flying and bottom ash depot and the schedule of exploration operations*

2 CONCEPT AND RESEARCH METHODOLOGY

Types of exploration to be realized:

- Collection and analysis of all available data for the given location,
- Geodetic operations,
- Field exploration operations (geotechnical explorations in situ, hydrogeological explorations in situ, geophysical explorations, exploratory drilling with additional operations, mapping of the ash depot of immediate surround terrain, core of the exploration drill sites and exploration trenches, core and soil sampling, monitoring well construction),
- Laboratory testing (physical-mechanical, geochemical, water chemistry, pedology, etc.),
- Development the geotechnical and geochemical paper on the results of conducted exploration.

Engineering-geological Exploration Operations

In order to determine the geotechnical terrain model and to define the geotechnical conditions as well as permanent solutions for the slope stability at the whole depot area, a detailed engineering-geological field exploration is required, as well as a complex geo-mechanical exploration of the present lithological factors which encompasses the following:

- Making the engineering-geological field map,

- Conducting the exploration drilling with core sampling and the following geotechnical and geochemical operations,
- Construction and engineering-geological mapping of exploration pits,
- Detailed engineering-geological core mapping,
- Sampling from exploration drill and exploration holes for laboratory, physical mechanical and geo-mechanical testing,
- Penetration test SPT,
- Static penetration test CPT,
- Water permeability test,
- Laboratory parameter testing the physical-mechanical soil characteristics.
- Laboratory chemical and geochemical soil and water testing.

Pedological Tests

In order to completely define the surfaces of degraded land due to erosion, as well as chemical pollution, the indicators and sub indicators were defined as factors for this type of pollution. Based on this, the degree of land endangerment was defined through the chemical pollution, i.e. based on the values of pollutants and concentration values of dangerous and harmful substances (As, Ba, Cd, Cu, Zn, Pb, Mn, Ni) which could indicate a serious land contamination. As an indicator of degradation the endangerment of the land by alkalization will also be displayed through (active and substitution acidity), electric conductivity and as a fertility indicator the content of organic matter.

Geophysical Exploration

In order to obtain the engineering-geological data of the field for closing, sanitation and reclamation of the flying and bottom ash depot "Srednje Kostolačko ostrvo" in Kostolac, aside from the geomechanical and laboratory testing, it is necessary to conduct the geophysical tests. Geophysical test will be conducted in the form of refraction-seismic tests and geoelectrical tests.

The task of the refraction-seismic and geoelectrical tests consists out of:

- determination the depths and speeds of spreading elastic longitudinal ($\mathbf{V_p}$) and transverse waves ($\mathbf{V_s}$) in certain lithological environments,
- determination the thickness (**D**) of surface complex,
- determination the spatial setting and depth positioning of individual lithological factors,
- determination the value of specific electrical resistance ρ_p and ρ in **ohm** of certain lithological factors for the requirements of facility grounding design.

Refraction-seismic tests are based on determining the value of expansion speed of elastic longitudinal ($\mathbf{V_p}$) and transverse waves ($\mathbf{V_s}$) in different lithological environments.

Geoelectric tests will be conducted applying the method of specific electric resistance in the variant of geoelectric scanning. Through this methodology of geophysical tests and field reconnaissance on a wider area, the data on thickness the individual lithological factors and determination of compromised zones due to the underground water circulation would be obtained.

The purpose of these geophysical - (seismic and geoelectrical) testing is to determine the depths and speeds of elastic longitudinal ($\mathbf{V_p}$) and transverse wave ($\mathbf{V_s}$) spreading in the specific lithological environments, determination the spatial setting and depth of grounding the individual lithological factors.

Geochemical Tests

For the purpose of realization the Geotechnical paper of remediation and facility construction in the area of flying and bottom ash depot "Srednje Kostolačko ostrvo", with the project of geotechnical explorations among others, the geochemical and hydrochemical exploration of the immediate depot area are designed.

Sampling locations are designed by the project, and the precise sampling micro locations will also be defined by the project. In case of unpredicted circumstances, changes and choosing the new locations for sampling will be defined in accordance with the project designer.

In the set of these explorations, the following operations are designed:

- Sampling of soil for the geomechanical parameter testing,
- Sampling of underground water for quality control,
- Sampling of surface water for quality control.

CONCLUSION

Geotechnical explorations and results of these explorations will be the input data for the stability analysis of slopes of the existing depot as well for necessary geostatic stability and funding of the facilities of renewable energy sources, most likely the solar plant that can be built in the area of this ash depot.

The range and type of designed geotechnical exploration is planned so that it will secure the adequate base for making the previous Feasibility Study with the idea solution of construction a solar plant in the Srednje Kostolačko ostrvo.

The range and type of planned geochemical explorations should provide a sufficiently reliable base for remediation measures and protection of this area (ground and water).

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Branka Jovanović[*], Dragan Milošević [*], Vladan Čanović[*], Klara Konc Janković[*]

MANAGEMENT OF THE LIMESTONE MINING AND DEPOSITION PROCESS AT THE OPEN PIT "MUTALJ" FOR THE PURPOSE OF ACHIEVING THE NECESSARY QUALITY

Abstract

The open pit Mutalj is a part of the Lafarge Cement Factory of Beočin and its purpose is to supply the factory with limestone as one of the ore required for cement production. The required quality of the mined limestone is determined by the technological process of creating cement and, before all, the quality of marl as the essential ore. The Mutalj Basin is characterized by the fluctuating quality of limestone. The subject of this work is the methods applied in the mining and deposition in order to secure the required quality of limestone.

Keywords: *mining, depositing, limestone quality, open pit, Mutalj*

1 INTRODUCTION

The production of cement on Fruška gora had began in the beginning of the XIX century which was made possible due to the existence of primitive mineral ore, at first in a primitive manner and later in the Beočin Cement Factory (BCF), the oldest cement factory on the Balkans. Since the start of production up to today the factory has developed and improved the mineral ore exploitation and cement production processes.

Since 2002, the majority owner of the Beočin factory is the Lafarge company and LBFC had become a part of the Lafarge group with its headquarters in Paris. The Lafarge group is the world leader in the construction industry and employs 83,000 people in 75 countries.

The Beočin Cement Factory had a leading position in the local market even before, and with the Lafarge company behind it, the factory is positioned at the very top of the cement industry in Eastern Europe. Through modernization of production and solving the ecological problems, there is a possibility of taking the advantage of good experiences of the Lafarge group from all around the world and the world business standards.

The planned capacities and the business radius of the LBFC are based on the local market.

The basic mineral ore for cement production are marl and limestone.

LBFC has, besides the factory in Beočin, two active open pits for ore mining required to produce cement:

1. Limestone open pit Mutalj, 19 km away, and
2. Marl open pit Filijala, 3 km away from the factory.

Limestone and marl are mined at the open pits of LBFC, are then transported to the Beočin factory where they are crushed, dried, ground up and homogenized. After that, the ore is heated and introduced into a furnace from which it exits as cement clinker, which is then cooled down and ground up with gypsum, ash and other additives being added after which the final product -

[*] *Mining Institute, Belgrade, povrsinska@ribeograd.ac.rs*

the cement, is packaged and transported to the consumer.

The proportions of marl and limestone in the base mixture is defined by a technological production procedure, depending on the quality of the starting components, primarily the content of $CaCO_3$. According to the technological recipe at the start of production in the factory, the proportion was 3:1. Currently the proportion of marl and limestone is 60:40, with the limestone share tending to increase.

2 CHARACTERISTICS OF THE MUTALJ BASIN

The limestone basin of Mutalj is located on the south slopes of Fruška gora, between Sremska Mitrovica and Beočin.

The limestone basin Mutalj is explored on the surface of about 100 ha on in which thd limestone balance reserves were found in 43 ha.

The basin belongs to a group of sediment layered basins, with a roof and shelf configuration that is nearly the same as the terrain configuration. The altitude differences in the terrain of the contoured area of the basin, from NE to SW, are 60 m.

The useful minerals in the basin are carbonate sediments - real reef "Lajtovacki" limestone made out of gray sandy and sandy-marl limestone and reef white, yellow to red limestone.

The thickness of the "Lajtovacki" limestone in the ore mass of the basin caries, from 6 m in the northeast, to 136 m in the south and southwestern area, 53.5 m on average. In general, the ore mass goes along the lines of west-east to northwest-southeast with a slight drop towards south to southwest. Thickness of the overlaying waste sediments is 23.54 m on average, maximum 65 m in the east part of the basin.

The floor of the "Lajtovacki" limestone at the location of the Mutalj Basin is comprised of lake sediments, the so-called Vrdnik series, presented as colorful clastic clays with lenses of gravel-clay sandstones and layers of gray-green bentonite clays.

The overburden of the "Lajtovacki" limestone is made of quaternary deposits: the so-called Sremska series (of rusty red clay and sandy-gravel clay), loess formation and deluvial and colluvial alluvion of redeposited limestone Sremska series and loess.

According to the Mutalj Basin interpretation in the current study [3], two tectonic rift systems were established in the direction NW-SE and SW-NE, which separate parts of the basin in cascades (picture 1).

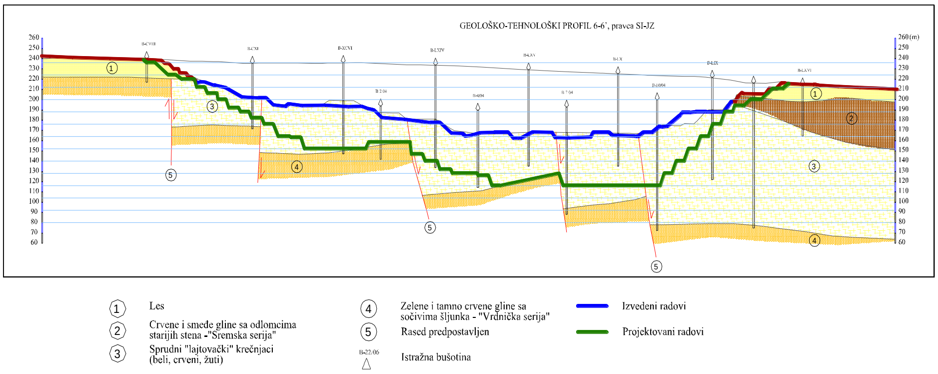

Figure 1 *Geological - technological profile 6-6' in the NE-SW direction*

Determination the limestone reserve boundaries [3] was done according to the content criterion of $CaCO_3 \geq 90\%$. Conducted geological explorations of the Mutalj basin have shown that the average content of $CaCO_3$ in the limestone reserves within the established contours is 93.91% and that within the same contour there are content changes of $CaCO_3$ to below 90 %. Figure 2 illustrates the map of the floor E-176 (h-12 m, altitudes from 176 - 188 m) with shaded surfaces with content of $CaCO_3$ below 90 %.

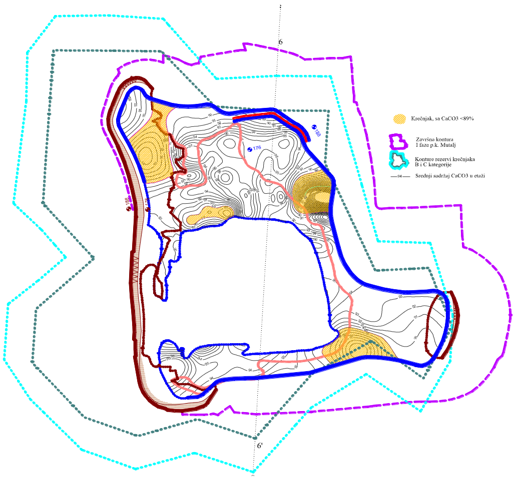

Figure 2 *Map of the floor E-176 (in the contour of stage I of OP Mutalj) with content of* $CaCO_3$ *isolines*

3 EXCAVATION PROCESS MANAGEMENT WITH THE OBJECTIVE TO EVEN OUT THE LIMESTONE QUALITY

In the contour of the I stage of OP Mutalj, the limestone quantities with the quality below border values are about 7%. In order to fully utilize the basin and to meet the technological process requirements in the cement factory of Beočin, exploitation at the open pit of Mutalj, selection and mixing of the lower quality limestone and higher quality limestone was carried out.

The basic equipment of limestone excavation at the OP Mutalj is a hydraulic dredge Liebherr 984C with a spoon E=4.7 m^3 and hourly capacity of Q_h = 420 t/h.

The daily capacity of the Liebherr R 984 C dredge, with two-shift operation and 6-hour shift is Q_d = 5,040 t/day and satisfies the daily factory requirements for limestone, Q = 5,000 t/day. Due to the changing quality of limestone in the basin, another hydraulic dredge is put into operation, Hyundai 450 LC with a “ripper” spoon E=2.2 m^3, and hourly capacity of Q_h=196 t/h and daily capacity of Q_d = 2,350 t/day.

In order to equalize the quality of the limestone and to fully exploit the basin, the production was conceived so that the limestone is excavated with two hydraulic dredges (Liebherr and Hyundai) at two different locations. The excavated limestone is loaded in two separate locations onto trucks and transported to the reloading plateau at the east side of the pit, where the limestone is loaded into tractor trucks with permitted carrying mass of M_{doz}=40 t for transport via public roads.

At the reloading plateau the limestone of lesser quality is temporarily set aside, so that it could later be mixed with a higher quality limestone, loaded onto trucks and transported to the reception bunker of the primary crusher in the cement factory of Beočin.

In accordance with the adopted concept of limestone excavation in the Technical Exploitation Project [1], the designed production dynamics of limestone production at the OP Mutalj was carried out in such a way that the annual limestone production is conducted by excavation in two different locations. As an illustration, Figure 3 presents the plan of excavation and overburden in the IV year of exploitation with the shaded locations of limestone excavation and location of reloading plateau (2).

Figure 3 *Plan of excavation in the IV year of the OP Mutalj - I stage*

4 CONCLUSION

Limestone of the Mutalj basin with the average content of $CaCO_3$ of 93.91 % is considered as the quality cement mineral ore which does not require special processing measures. A smaller part of limestone in the basin (around 7%) is of lesser quality and as such is unacceptable as the ore in the cement factory. At the open pit Mutalj, the methods of equalization the limestone quality are applied during exploitation: through excavation of limestone with two dredges, at two locations and mixing of the lower quality limestone with the high quality limestone at the reloading plateau. The application effects of described methods for limestone exploitation management are the continuous shipping of quality limestone to the factory reception bunkers and also greater basin utilization.

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Dejan Ivezić[*], Trajko Petrović[**], Dragan Zlatanović[***]

SEQUENTIAL DESIGN OF CONTROLLER FOR GRINDING CIRCUIT

Abstract

Multivariable controller design for grinding circuit is the subject of this work. Model of grinding circuit in Majdanpek Copper Mine is subjected to the newly proposed sequential design method, based on the Gauss-Jordan factorization and Nyquist-like techniques. Due to the nature of the applied design method, generalization to any other grinding circuit is straightforward. An extensive simulation analysis proved the effectiveness of proposed design in reference tracking, even in the case of some loop failures.
Keywords: *grinding process, sequential design, Nyquist array*

INTRODUCTION

Grinding of the ore to a fine product is the unit operation of crucial importance in the metallurgical extraction process. Its importance arises from the fact that it is the most expensive operation itself, but also, operation with the great influence to downstream extraction process. The ore exploitation with lower contain of useful component, as in the case of Serbian copper mines, make this importance even more significant.

The first major effort to automatic process control of grinding circuits made use of conventional PID-type controllers in many different single-loop control schemes. As a grinding process is not a single-variable control problem but is a multivariable system, conventional Inverse Nyquist Array (INA) method was favorite engineering tool for development a multivariable controllers for this kind of processes in last decades (Koudstaal et al., 1981, Hulbert et al., 1990). The direct extension of this approach, by introduction the model uncertainty and defines robust INA methodology, was presented by Ivezic and Petrovic (2003). Craig and MacLoad (1995, 1996) applied μ-controller synthesis and analysis methodology to design the robust controllers of grinding process. Duarte et al. (1999) and Pomerleau et al. (2000) made comparisons of different multivariable control strategies in grinding control.

For a number of reasons multi input - multi output (MIMO) control system, as for grinding circuits, could be designed, tuned and commissioned in sequential order, closing one feedback loop at the time. First, this sequential approach seems as an extension of well-known single input - single output (SISO) design methods into the area of generally much more complex MIMO systems. Second, a good sequential stability interpretation offers a possibility of obtaining a stable system at each stage of the design. A cautions use of this feature provides stability of the system even in the case of loop failure. Also, using of sequential design eliminate often a tedious work in direct decoupling of system but without loss of design quality and with faster solution obtain.

Basic concepts of the applied sequential method originate from Mayne (1979), as

[*] *University of Belgrade - Faculty of Mining and Geology, Belgrade, Serbia, ivezic@rgf.bg.ac.rs*
[**] *Alfa BK University - Faculty of Information Technology, Belgrade, Serbia, trajko.petrovic@alfa.edu.rs*
[***] *Innovation Centre, Faculty of Mechanical Engineering, University of Belgrade, Serbia, dr.dragan.zlatanovic@gmail.com*

suming that a pre-compensator is a full transfer function matrix, designed one column at each stage. Although such an approach simplifies a multivariable design into a series of simple multi input - single output design procedures, its implementation usually yields unnecessarily high order controllers within the pre-compensator. An improvement of the original sequential method was proposed in Bryant (1985), introducing Gauss elimination (GE) operations on the system return difference matrix. It was shown that the process of closing a system in sequential order could be formalized by GE, this way also avoiding afore mentioned controller order excess problem. Recently (Bryant and Yeung, 1996), additional method improvement is proposed the Gauss-Jordan (GJ) elimination instead of GE. Although both approaches offer a good sequential stability interpretation, using GJ elimination as a sequential transformation provides the full input - output transfer function of the partially closed system at each stage of design, whereas GE approach only provides transfer function associated with open-loop section of the system.

THE PLANT

A grinding circuit in the Majdanpek Copper Mine is shown in Figure 1. This circuit consists of two primary rod mills and two secondary ball mills closed by hydro-cyclones. Also, in the secondary loop are two unit flotation cells that prevent the over grinding of coarse, floatable material of high density.

The levels in each of sumps and the flow rates and density of the streams leaving sumps are measured. Thus, the discharge stream from the secondary mills can be characterized from balances round the unit-feed sump and the primary mills, and the feed stream to the secondary mills can be characterized from balances round the cyclones and flotation-feed sump. The level in each sump was controlled by separate system, which allowed them to act as a buffer against fluctuations in the incoming flow while the outgoing flow was kept under tight control. On the overflow line from the cyclone a particle size monitor (P.S.M.) measures the particle size distribution and the mass percentage of the solids in this product stream. Only one point on the particle size distribution is measured, namely the percentage of the solids smaller than 75 μm, and no indication of the particle size spread is given.

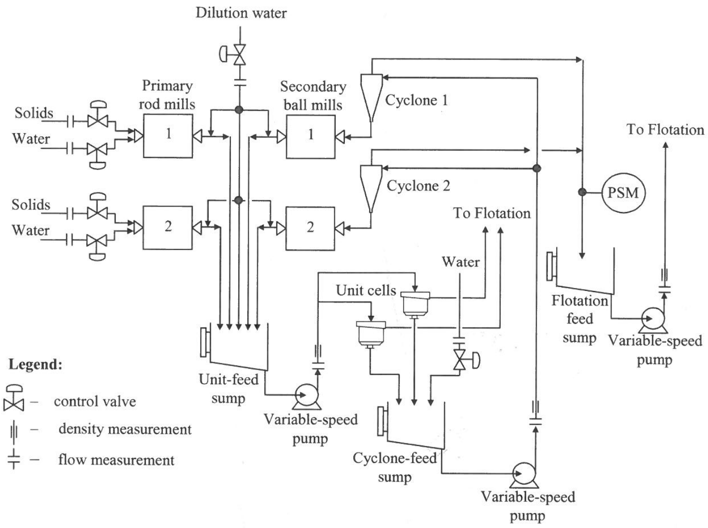

Figure 1 *Grinding circuit in Majdanpek Copper Mine*

The model of grinding circuit was obtained from its experimental identification around particular operating point, Grujic (1995). Transfer function of grinding circuit (consisting of the particle size of the product and the feed flow rate of the to the cyclone as outputs and the solid feed (together with its dilution) and the cyclone dilution as inputs) is given by Ivezic and Petrovic (2003), as 2x2 transfer function matrix. That transfer function matrix is here somewhat extended and modified for the reason of more adequate representing system dynamic but also for precise control. Namely, instead of total dilution as input variable, unit-feed sump dilution and cyclone-feed sump dilution are introduced as new input variables and density of the feed to the cyclone is introduced as new output variables. Also, sequence of input and output variables is rearranged so that model of grinding circuit is represented by 3x3 transfer function matrix:

$$P = \begin{pmatrix} \dfrac{36.49}{792s+1} & \dfrac{1.1405}{179s+1} & \dfrac{0.1866}{573s+1} \\ \dfrac{78.9}{1676s+1} & \dfrac{-4.51}{63.5s+1} & \dfrac{0.94}{683s+1} \\ \dfrac{-0.9362 \cdot e^{-350s}}{1164s+1} & \dfrac{(10.252s+0.003)\cdot e^{-200s}}{80218s^2+652s+1} & \dfrac{(132.7s+0.057)\cdot e^{-80s}}{36300s^2+358s+1} \end{pmatrix} \quad \begin{array}{l} y = Pu \\ y = [y_1\ y_2\ y_3]^T \\ u = [u_1\ u_2\ u_3]^T \end{array} \tag{1}$$

u_1 - solid feed (together with its dilution)
u_2 - unit-feed sump dilution
u_3 - cyclone-feed sump dilution

y_1 - cyclone feed flow rate
y_2 - cyclone feed density
y_3 - particle-size measurement of the products

The control objective of grinding circuit is to maintain the particle size of the product and flow rate and density of the feed to the cyclone close to set points $y_{ref} = [y_{1ref}\ \ y_{2ref}\ y_{3ref}]^T$.

SEQUENTIAL METHOD BACKGROUND

The applied sequential design method is described in detail by Bryant and Yeung (1996). A closed loop system structure is shown in Figure 2, where K(s) stands for controller (precompensator).

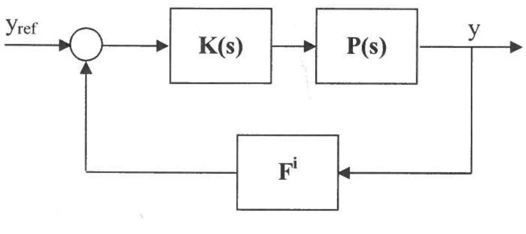

Figure 2 *The considered system structure*

It is assumed that K(s) is a full transfer function matrix, rather than just diagonal or triangular. Anyhow, K(s) can be written in the form

$$K(s) = K^{n+1}(s) \cdots K^2(s) K^1(s) \quad (2)$$

where $K^j(s)$, j = 1,..., n + 1, are constructed from an identity matrix by replacing j-th column with a column of transfer functions. The condition of feedback loops for system in Figure 2 is described by real matrix F^i

$$F^i = diag\{f_1 \cdots f_i \; 0 \cdots 0\}, F^{n+1} = F = I \quad (3)$$

where $f_j = 1$, j = 1,...i, stands for closed feedback loop. For precompensator structure (2) and a given plant transfer function matrix P(s), the j-th column of the return difference matrix R(s) of the system, defined as

$$R(s) = I + P(s)K(s)F \quad (4)$$

becomes a function of the j-th column of $K^j(s)$ only. Hence, the precompensator design is accomplish through a series of multi input - single output (MISO) designs, obtaining one $K^j(s)$ while simultaneously closing the associated feedback loop at each stage. The obvious advantage of this multistage sequential approach is a strong background in SISO design methods. On the other hand, a direct implementation yields precompensator of excessively high orders, and more seriously, the designer has no notion of the final form of K(s).

In order to overcome these problems, Bryant and Yeung (1996) proposed use of sequential transformation, which consists of series of GJ operation on R(s). These operations can be formalized as follows

$$N^{n+1}(s) \cdots N^2(s) \, N^1(s) \, R(s) = I \quad (5)$$

where $N^j(s)$ represents a row operation of the j-th stage of GJ elimination and is constructed from an identity matrix by replacing the j-th column with the transfer functions $n_{ij}^j(s), i = 1, \ldots, n+1$. If $R^{(j)}(s) = \left[r_{ik}^{(j)}(s)\right]$ denotes a return difference matrix after j stages of GJ elimination,

$$R^{(j)}(s) = S^j(s) R(s) \quad (6a)$$

$$S^j(s) = N^j(s) \cdots N^2(s) N^1(s) \quad (6b)$$

where S(s) stands for sensitivity matrix

$$S(s) = S^{n+1}(s) = R^{-1}(s) \quad (7)$$

then $n_{ij}^j(s)$ is defined as

$$n_{ij}^j(s) = \begin{cases} -\dfrac{r_{ij}^{(j-1)}(s)}{r_{jj}^{(j-1)}(s)} & \text{if } i \neq j \\ \dfrac{1}{r_{jj}^{(j-1)}(s)} & \text{if } i = j \end{cases} \quad (8)$$

Furthermore, using (6) and (3), (4) can be rewritten as

$$S^j(s)P(s)K(s) = P^j(s)K(s) = R^{(j)}(s) - S^j(s) \quad (9)$$

showing that after j stages of GJ operations on R(s), obtained transfer function Pj(s)K(s) represents partially a closed system with first j feedback loops closed; here Pj(s) is interpreted as a new open-loop system matrix with first j loops closed. It was already shown in Bryant (1985) that sequential loop closure can be formalized through a series of GE factorizations. However, in Bryant and Yeung (1996) it was provided that applying successive GJ operations on R(s) yields not only the transfer function of partially closed system, but also provides that columns of precompensator K(s) can be determined separately, one at each stage, even more directly than suggested by (2). In elements $r_{jj}^{(j-1)}(s)$ of R(j-1)(s), called sequential return difference functions, which is due to the nature GJ operations.

A sequential method via GJ factorization can now be outlined in the following procedure:

0. Let $K^0(s) = N^0(s) = S^0(s) = I$,

$$P^0(s) = Y^0(s) = P(s), \; j = 1.$$

1. Prepare the Nyquist array

$$Y^{j-1}(s)=\begin{bmatrix} I-S_{11}^{j-1}(s) & P_{12}^{j-1}(s) \\ P_{21}^{j-1}(s) & P_{22}^{j-1}(s) \end{bmatrix}$$

partitioned into the blocks $Y_{11}^{j}(s)_{j\times j}$, $Y_{12}^{j}(s)_{j\times(n+1-j)}$, $Y_{21}^{j}(s)_{(n+1-j)\times j}$ and $Y_{22}^{j}(s)_{(n+1-j)\times(n+1-j)}$.

2. Examine the Nyquist array $Y^{j-1}(s)$ and choose precompensator column $k_{ij}(s), i=1,\ldots,n+1$. Then compute $r_{jj}^{(j-1)}(s)$, plot its Nyquist contour and count the number of counter-clockwise encirclements about the origin, c_j. In order for closed loop system to be stable, it must be $\sum_{i=1}^{n+1} c_i = -p_0$, where p_0 is the number of unstable poles of the original P(s). Repeat this step until the stability margin requirements are satisfied.

3. Update $K^j(s)$ by substituting the j-th column $K_{*j}^{j}(s)$ with $k_{ij}(s)$, when update $R^{(j-1)}(s)$ according to the newest precompensator column, $R_{*j}^{(j-1)}(s)=P^{j-1}(s)K_{*j}(s)+e_j$, e_j representing the j-th column of identity matrix.

4. Compute $N^j(s)$ and perform one stage of GJ factorisation by updating

$$R^{(j)}=N^j(s)R^{(j-1)}(s) \tag{10a}$$

$$S^j(s)=N^j(s)S^{j-1}(s) \tag{10b}$$

$$P^j(s)=S^j(s)P(s)=N^j(s)P^{j-1}(s) \tag{10c}$$

5. Compute a transfer function of partially closed system, $H^j(s)=P^j(s)K^j(s)$. Submatrix $H_{11}^{j}(s)_{j\times j}$ should be close to the unity matrix I_j, and submatrices $H_{21}^{j}(s)_{(n+1-j)\times j}$ and $H_{12}^{j}(s)_{j\times(n+1-j)}$ should be as small as possible over the operating bandwidth. If not, repeat the procedure from step 1.

6. Increase j by 1. If still j ≤ n + 1, continue with the procedure from step 1, else the design is completed.

Since the Nyquist approach is utilized when choosing kij(s) in the stages 1 and 2 according to Bryant and Yeung (1996) this method is called the Basic Gauss-Jordan Nyquist design procedure.

CONTROLLER DESIGN

One of the most critical point in a grinding circuit is the cyclone underflow, since overloading at that point can lead to sanding-up of the secondary mills and to discharge of very coarse material to the product. Cyclone underflow is not measured direct in the Majdanpek Concentrator, but it is affected directly by flow and density of its feed. Therefore control of these two variables has priority so that loop closing procedure will start with one of these two loops.

According to steps 1 and 2 of the sequential procedure, the Nyquist array of $Y^0(s) = P(s)$ is examined. Its Nyquist plot is presented in Figure 3. It is evident that $Y_{21}^{0}(s)$ is relatively very large with respect to $Y_{11}^{0}(s)$, therefore the cross-couplings from loop-2 to loop-1 is high and therefore sequential design will start from loop 2. For further design is important to notify that $Y_{13}^{0}(s)$ and $Y_{23}^{0}(s)$ are relatively small with respect to $Y_{11}^{0}(s)$ and $Y_{22}^{0}(s)$, respectively.

Exploiting the last conclusions and using classical loop-shaping techniques, the second column of controller (precompensator) is chosen to be:

$$K_{*,2}^{1}(s)=\begin{bmatrix} \frac{1}{P_{21}(s)}\frac{k_1}{s(T_1s+1)} & \frac{1}{P_{22}(s)}\frac{k_1}{s(T_1s+1)} & 0 \end{bmatrix}^T \tag{11}$$

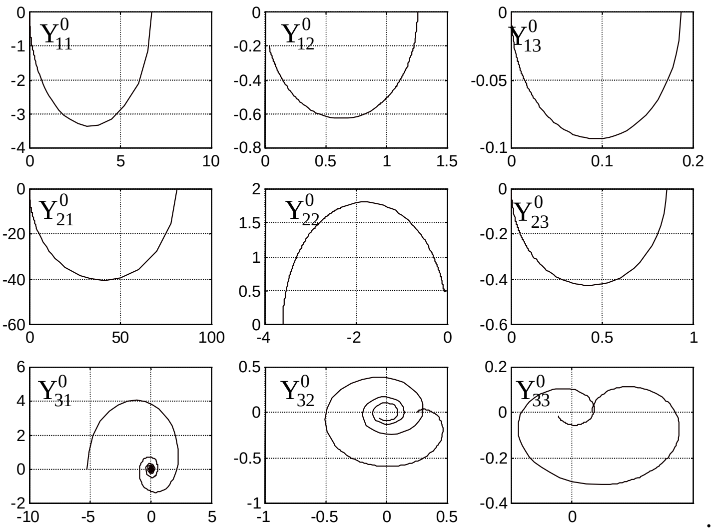

Figure 3 *Nyquist plot of $Y^0(s)$*

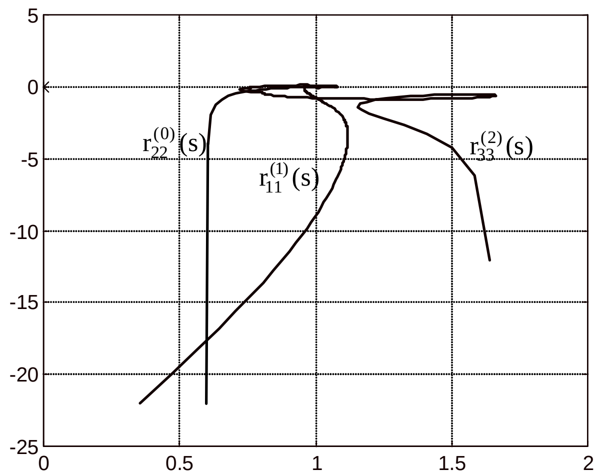

Figure 4 *Sequential return difference functions*

Time constant $T_1 = 1$ and gain $k_1 = 0.1$ are chosen according to step 2 requirement for number of encirclement sequential return difference function $r_{22}^{(0)}(s)$ about the origin. Since the open loop transfer function P(s) is stable, the sum of the origin counter-clockwise encirclements c_j of the sequential return difference function $r_{jj}^{(j-1)}(s)$ at each stage should be equal to zero, in order for a closed loop system to be stable. Figure 4 shows that this hold for $r_{22}^{(0)}(s)$. Physically, it means that cancel of loops 1 and 3 will not affected to regular feed density to cyclone. This ensure that sending-up of mills will be avoided

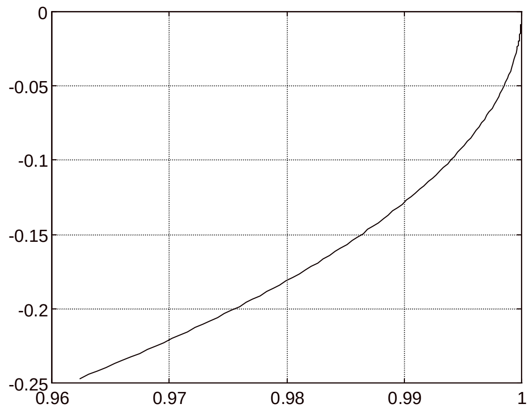

Figure 5 *Nyquist plot of* $h_{22}^{1}(s)$

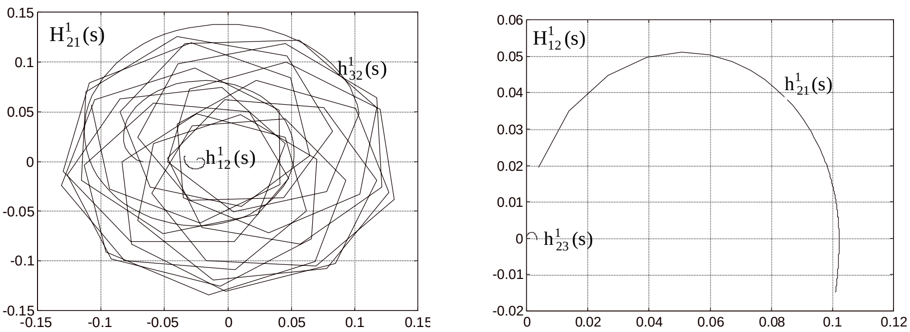

Figure 6 *Nyquist plots of* $H_{21}^{1}(s)$ *and* $H_{12}^{1}(s)$

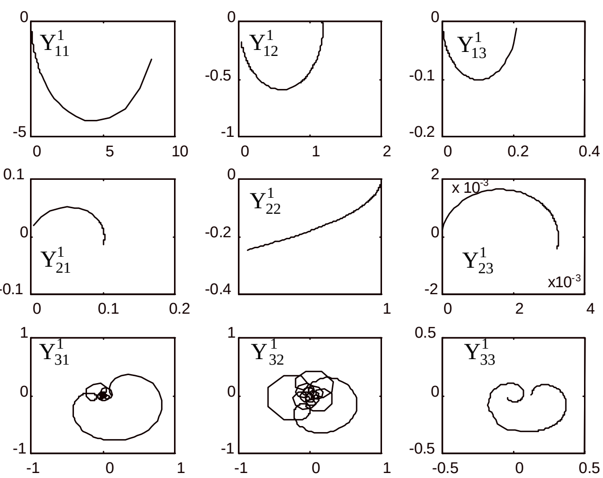

Figure 7 *Nyquist plot of* $Y^{1}(s)$

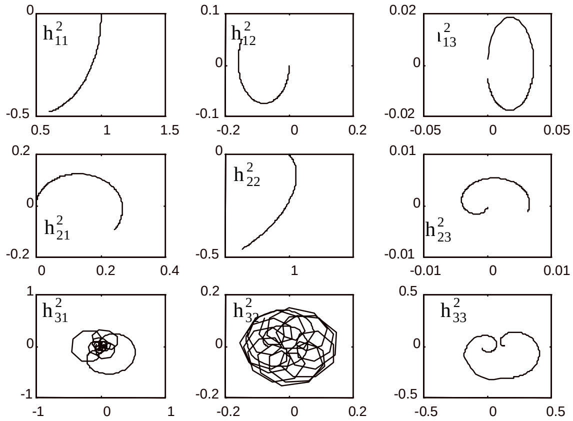

Figure 8 *Nyquist plots of* $\mathrm{H}^2(\mathrm{s})$

Following further steps 3 to 5 of the procedure, a partially closed system with only the second feedback closed $\mathrm{H}^1(\mathrm{s}) = \lfloor \mathrm{h}^1_{\mathrm{ik}}(\mathrm{s}) \rfloor$ is computed and reviewed. Thus Figure 5 shows that $\mathrm{H}^1_{11}(\mathrm{s}) = \mathrm{h}^1_{22}(\mathrm{s})$ is close to unity over the operating bandwidth, while

$$H^1_{12}(s) = [h^1_{21}(s) \quad h^1_{23}(s)]$$

and

$$H^1_{21}(s) = [h^1_{12}(s) \quad h^1_{32}(s)]$$

submatrices are insignificant comparing to $\mathrm{h}^1_{22}(\mathrm{s})$, Figure 6. This means that $\mathrm{K}^1(\mathrm{s})$ design so far yields both a good loop-shaping and decoupling, the later providing that the following stages of the design will not affect significantly the results of this one.

The second stage of design starts with putting j = 2 and examining the Nyquist array of $\mathrm{Y}^1(\mathrm{s})$, Figure 7, formed by the previously computed $\mathrm{S}^1(\mathrm{s})$ and $\mathrm{P}^1(\mathrm{s})$. It is obviously that $\mathrm{Y}^1_{22}(\mathrm{s}) = \mathrm{h}^1_{22}(\mathrm{s})$ and off-diagonal elements of $\mathrm{Y}^1_{11}(\mathrm{s})$ are significantly smaller of it. According to this fact, that is loop-1 is now insensitive to another, and using classical loop-shaping techniques, the first column of controller (precompensator) is chosen to be:

$$K^2_{*,1}(s) = \begin{bmatrix} \frac{1}{P_{11}(s)} \frac{0.005}{s(s+1)} & 0 & 0 \end{bmatrix}^T \tag{12}$$

Following further steps of the procedure, a partially closed system with the first and the second feedback closed $\mathrm{H}^2(\mathrm{s}) = \lfloor \mathrm{h}^2_{\mathrm{ik}}(\mathrm{s}) \rfloor$ is computed and reviewed. Figure 8 shows that $\mathrm{h}^2_{11}(\mathrm{s})$ is close to unity over the operating bandwidth, $\mathrm{h}^2_{22}(\mathrm{s})$ is unchanged, as expected and off-diagonal elements of these two terms are significantly smaller. From the point of view of overall system stability it is worth nothing that, just like in the previous stage of design, sequential return difference function $\mathrm{r}^{(1)}_{11}(\mathrm{s})$ has zero origin encirclements, $c_2 = 0$, Figure 4. Physically, it means that cyclone can work independently, regardless to loop-3 of system. This concludes the second stage.

The last stage of design starts with putting j = 3 and examining the Nyquist array of $\mathrm{Y}^2(\mathrm{s})$, formed by the previously computed $\mathrm{S}^2(\mathrm{s})$ and $\mathrm{P}^2(\mathrm{s})$, Figure 9. Existing the time delay in row 3 and somewhat large element $\mathrm{Y}^2_{31}(\mathrm{s})$ according to $\mathrm{Y}^2_{33}(\mathrm{s})$ require introducing of PI and phase-lag control in the precomensator third column:

$$K_{*,3}^{3}(s) = \left[\frac{1675s+1}{500s+1} \cdot \left(-0{,}019 - \frac{0{,}0002}{s} \right) \quad 0 \quad \left(1 + \frac{0{,}01}{s} \right) \right]^{T} \tag{13}$$

Final closed loop transfer function matrix $H^3(s) = H(s)$ has excellent frequency characteristics for all diagonal elements, together with strong decoupling of the off-diagonal elements, as shown in Figure 10. Precompensator choice defined by (11-13) gives zero origin encirclements of sequential return difference function $r_{jj}^{(j-1)}(s)$, $c_j = 0$, at each design stage, Figure 4. Together with the fact that the open loop transfer function matrix P(s) is stable, this guarantees stability for closed loop transfer function matrix H(s). Thus, the design is completed with stability and performance objectives satisfied. Also, it is interesting to note that controller obtained by GJ sequential design is simple (all transfer functions of K(s) are easy to implement) and that the controller is sparse matrix (four elements of nine of K(s) are zero).

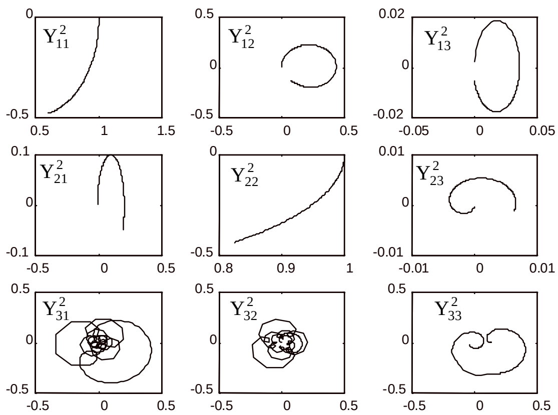

Figure 9 *Nyquist plot of $Y^2(s)$*

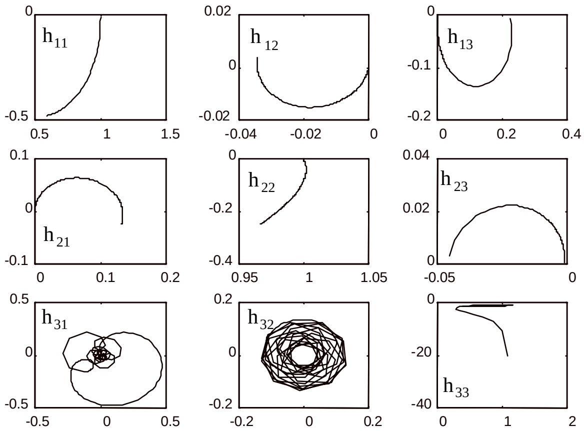

Figure 10 *Nyquist plots of H(s)*

THE TRANSIENT ANALYSIS

Extensive simulations have been performed to confirm the obtained results. Time responses of closed-loop system, using sequential controller (SC) and previously de signed, INA and decentralized (DC) controllers are compared in order to observe how the system tracks set points changes. A unit step is used as a test signal.

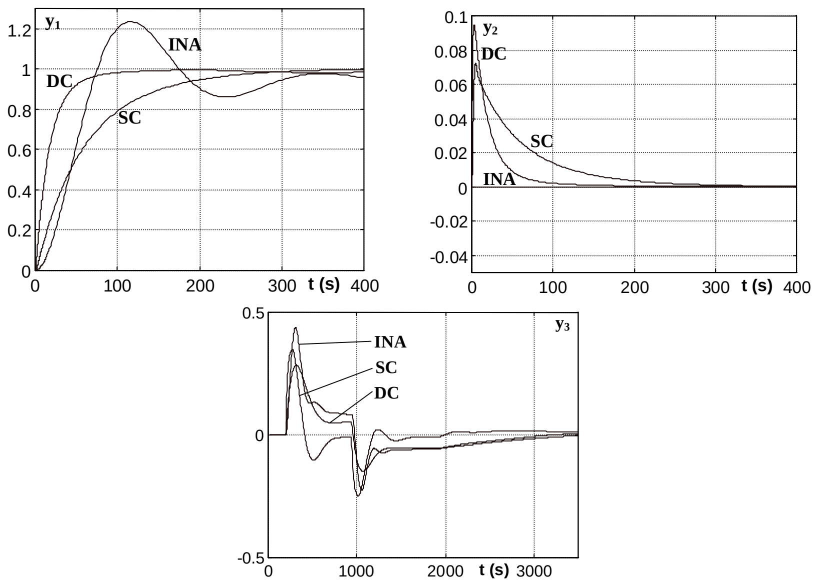

Figure 11 *Time responses of plant with different controllers to unity step signal in input u_1*

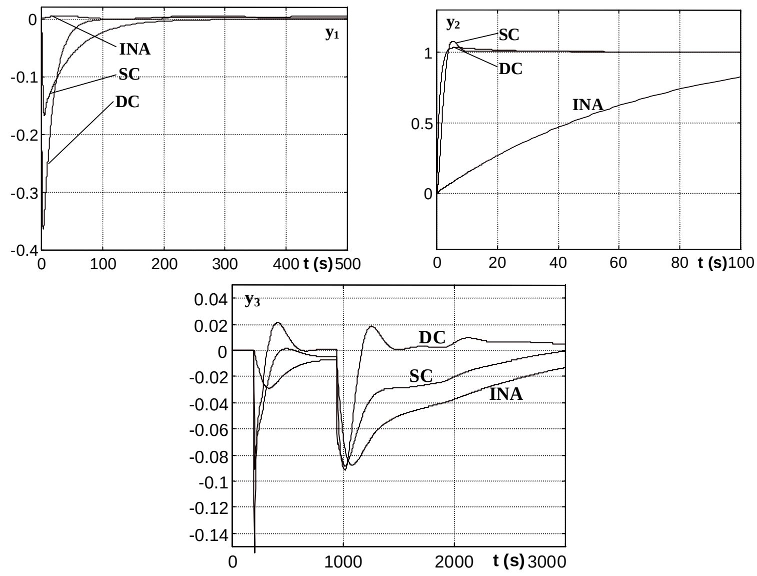

Figure 12 *Time responses of plant with different controllers to unity step signal in input u_2*

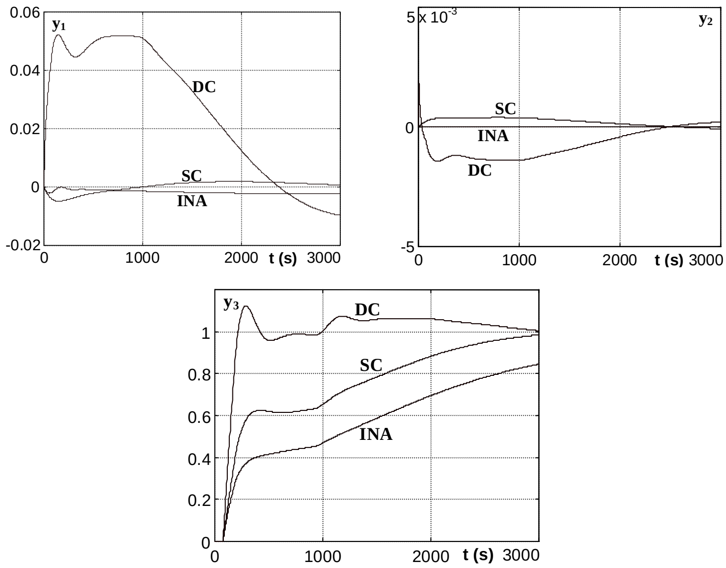

Figure 13 *Time responses of plant with different controllers to unity step signal in input u_3*

The most typical time responses of particle size of the product, cyclone feed flow rate and density on cyclone-feed sump dilution, solid feed (together with its dilution) and unit-feed sump dilution are shown in Figures 11-13. It is evident that characteristics of sequential controller lie somewhere between of characteristics of INA and DC controllers. Sequential controller has faster response compared to INA controller. The use of very complex structure of decoupling precompensator in INA controller design as a consequence has somewhat better decoupling characteristic. But simplicity of sequential controller, together with straight forward procedure for its derivation overcomes that advantage of INA design. Moreover, time responses of plant with failure of loops 1 and 3 show that decoupling characteristic of sequential controller is superior, even to INA controller, Figure 14 (responses in output y_2 are similar as in case of all loops closed - Figure 12). It is because special case, analogous to the first stage of sequential design, was considered. DC controller is with the similar responses characteristics as sequential controller but with the worst decoupling of the system which is more evident in the case of loop failures.

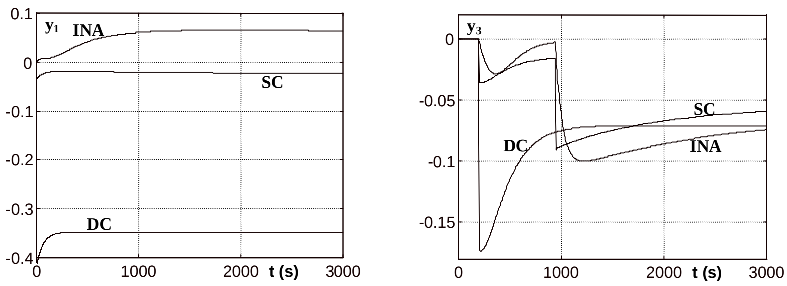

Figure 14 *Time responses of plant with failure of loops 1 and 3 to unity step signal in input u_2*

In this paper the design of a multivariable controller for grinding process is presented. The applied design method is the Gauss-Jordan Nyquist sequential procedure. Adequate sequential controller for the grinding circuit in Majdanpek Copper Mine is obtained, its structure discussed and variable parameters tuned. The simulation analysis revealed satisfactory closed loop performance. Comparison with INA and decentralized controllers justified use of this design method. Procedure is more straightforward than in case of INA design but with quite well decoupling result. Also, sequential design procedure provides in some degree integrity of the system, that is stability robustness in the face of loop failures. In the case of decentralized controller use, interaction between the loops is still significant and so unacceptable.

Regardless to different structure of grinding circuits used in different metallurgical extraction processes, mathematical description of physical dependences and phenomena is similar. As the utilization of Nyquist array plots in controllers' design is recognizable for the engineers in this branch and the proposed sequential design is based to Nyquist approach, given arguments and explanations for use of sequential design method, define this method as powerful tool for controllers design for this class of processes.

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*Branislav Rajković**, Zoran Ilić*, Radmilo Rajković**

VERIFICATION THE DRIVE UNIT COMPONENTS OF BELT CONVEYOR FOR ORE T.109

Abstract

This paper presents the methodology for verification the reducer and couplings as the components of drive unit on the example of belt conveyor for ore with capacity of 2000 t/h at the open pit "Veliki Krivelj".

Analysis was done by calculation according to the manufacturer instructions of the Falk Company, and its results are indispensable for checking the reliability of functionality the drive unit components in the given operating conditions.

Also, the technical characteristics of these components are given, as well as their graphical representation.

***Keywords**: belt conveyor for ore, verification of reducer and couplings, technical characteristics*

1 INTRODUCTION

The belt conveyor for ore T.109 was designed for transportation the primary crushed ore at the open pit "Veliki Krivelj" from the crusher (pos. T.102.100.2) to the open storage for ore [1]. To run the conveyor belt, two identical drive units are provided, each of which runs a single drive pulley. Each drive unit consists of an electric motor, a high-speed coupling, a reducer and a low-speed coupling.

Since the investor disposed of drive units with certain technical characteristics, it was necessary to check the functionality of drive unit components unit in the new operating conditions on the belt conveyor T.109, i.e. to do their verification.

2 TECHNICAL DESCRIPTION

The drive unit of the belt conveyor consists of an electric motor, a high-speed coupling between the electric motor and reducer, a reducer and a low-speed coupling between the reducer and drive pulley.

The reducer is of drive size 485 and type A [2] what means that it is with parallel shafts, horizontal and foot mounted, with solid low-speed shaft. Gears are helical and suitably machined to assure a full contact under load. The reducer is installed outdoors, and the position of reducer corresponds to assembly number 1. The reducer is with double reduction, nominal ratio 15.44 and actual ratio 15.24. The reducer is not equipped with any accessories what means that the reducer is cooled by natural circulation of ambient air. Since the transmission is through flexible couplings at both the high-speed and low-speed shafts, these shafts are not loaded with additional bending moments. The reducer image is given in Figure 1.

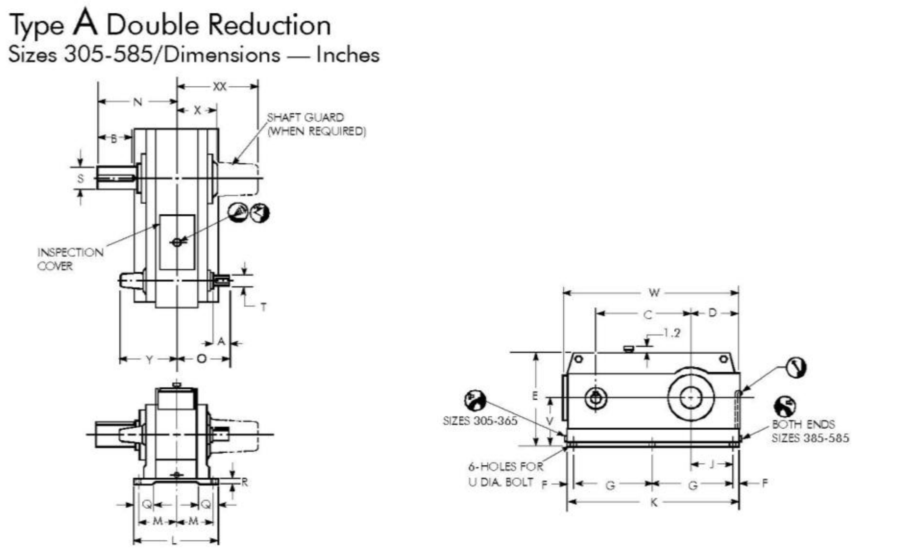
* *Mining and Metallurgy Institute Bor, branislav.rajkovic@irmbor.co.rs*

The high-speed coupling is 1110T10, and the low-speed coupling is 1180T10 [3]. The couplings belong to the group of Falk Steelflex Grid Couplings, type T10 what means that it is a double flexing, close coupled design for use in four bearings systems with horizontally split cover which allows for grid replacement without the movement of the connected equipment. View of couplings is given in Figure 2.

Technical characteristics of drive unit components are given in Table 1.

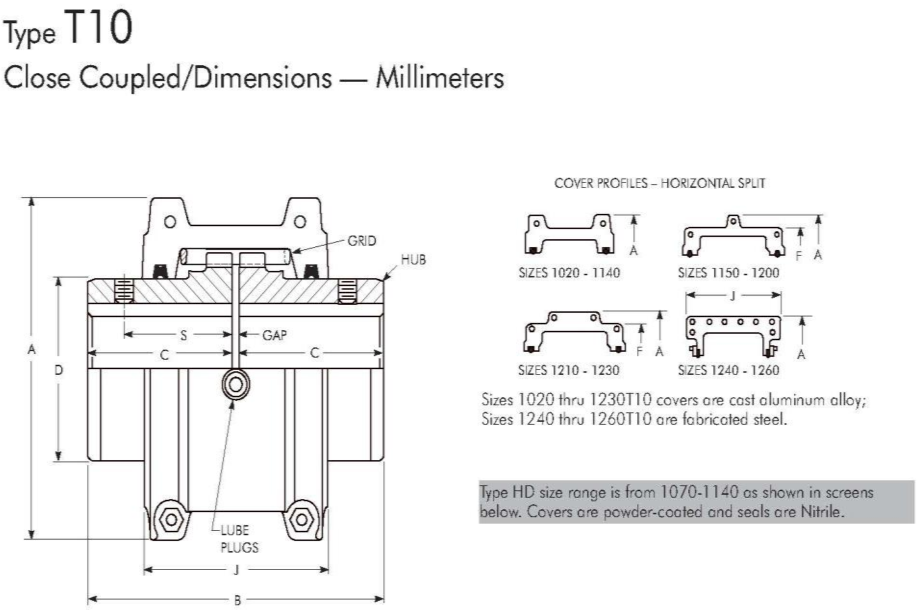
Table 1 *Technical characteristics of drive unit components*

Item	**Electric motor**	**High-speed coupling**
Technical characteristics	- Manufacturer: Sever Subotica - Label: ZPN 6170 - Power: $P = 450\ [kW]$ - Speed: $n = 982\ [min^{-1}]$ - Mass : $m = 4200\ [kg]$	- Manufacturer: Falk - Label: 1110T10 - Torque rating: $M_n = 9320\ [Nm]$ - Maximum speed: $n_{max} = 2250\ [min^{-1}]$ - Bore diameter: $d = 42 \div 120\ [mm]$
Item	**Reducer**	**Low-speed coupling**
Technical characteristics	- Manufacturer: Falk - Label: 485-A2 - Power: $P = 788\ [kW]$ - Ratio: $i_n = 15.44\ [-]$ - Mass: $m_{red} = 2857.6\ [kg]$	- Manufacturer: Falk - Label: 1180T10 - Torque rating: $M_n = 103000\ [Nm]$ - Maximum speed: $n_{max} = 1100\ [min^{-1}]$ - Bore diameter: $d = 153 \div 300\ [mm]$



A	B	C	D	E	F	G	J	K	L	M	N	O
9	14.88	30	19	35	2.5	22.5	15	58.5	33	14.4	30.7	23
		Low-speed shaft		High-speed shaft								Mass in lb
Q	R	S	Key	T	Key	U	V	W	X	XX	Y	
8	1.5	8.5	2x1.5x13	4.5	1x1x8	2	18	61.12	14.36	31.24	23.9	6300

Figure 1 *Reducer 485-A2 with assembly dimensions*



Size	A	B	C	D	F	J	Gap
1110T10	269.7	259.0	127.0	160.3	...	161.5	5
1180T10	629.9	483.6	238.8	393.7	554.7	321.1	6

Figure 2 *Couplings T10 with assembly dimensions*

3 CALCULATION

3.1 Calculation of Reducer

Calculation of reducer has for a purpose of verification the technical characteristics of reducer in given operating conditions and it is given according to [2]. The reducer is selected according to the required ratio of reducer and equivalent power which has to be less than the power rating of reducer. Checking the thermal rating of reducer is done according to the application adjusted thermal rating which must be greater than the actual power transmitted by reducer.

3.1.1 Reducer low-speed shaft rpm required

$$n_2 = \frac{60 \cdot v}{D \cdot \pi} [min^{-1}] = 64\ [min^{-1}]$$

Where:

$v = 3.8 \left[\frac{m}{s}\right]$ –conveyor belt speed

$D = 1{,}132\ [m]$ - lined drive pulley diameter

3.1.2. Reducer ratio required

$$i = \frac{n_1}{n_2} = 15{,}34$$

Where:

$n_1 = 982\ [min^{-1}]$ - reducer high-speed shaft rpm

3.1.3 Reducer equivalent power

$$P_{EQ} = s_f \cdot P_{EM}\,[kW] =$$
$$= 675\,[kW] < P_{RED} = 788\ [kW]$$

Where:

$s_f = 1.5$ - service factor for heavy duty belt conveyors working over 10 hours per day

$P_{EM} = 450\ [kW]$ - electric motor power rating

$P_{RED} = 788\ [kW]$- reducer power rating

3.1.4. Application adjusted thermal rating

$$P_{TA} = B_1 \cdot B_2 \cdot B_3 \cdot B_5 \cdot P_T\,[kW] =$$
$$= 395\,[kW] > P_{EF} = 329\,[kW]$$

Where:

$B_1 = 0.822$ - ambient temperature factor for ambient temperature $t = 40\,[°C]$ and for reducer with or without shaft or electric fan

$B_2 = 1.00$ –altitude factor for altitude above sea level $H = +350{,}00\,[m]$ and for reducer with or without auxiliary cooling

$B_3 = 1.90$ –ambient air velocity factor for sustained ambient air velocity $v > 3{,}683\ \left[\frac{m}{s}\right]$, outdoors installed environment and reducer without shaft or electric fan or cooling tubes

$B_5 = 1.00$ - duty cycle factor for operating time per hour of $E = 100\,[\%]$ and for reducer with or without auxiliary cooling

$P_T = 253\ [kW]$ –basic thermal rating for reducer type A, AR and AXV with double reduction, for nominal ratio 11.39÷20.93, reducer size 385÷585, high-speed shaft rpm $n_1 = 982\ [min^{-1}]$ and for reducer with no auxiliary cooling

$P_{EF} = 329\ [kW]$ - actual power transmitted by reducer for belt conveyor drive [4]

3.2 Calculation of Couplings

Calculation of couplings has for a purpose of verification the technical characteristics of high-speed and low-speed coupling in given operating conditions and it is given according to [3]. Couplings are selected according to the torque rating of specific coupling which has to be greater than the minimum coupling rating obtained by multiplying the service factor and system torque rating, having in mind coupling dimensions and maximum speed.

3.2.1 System torque rating of high-speed coupling

$$T_{HS} = \frac{P_{EF} \cdot 9549}{n_1}\ [Nm] =$$
$$= 3199{,}2\ [Nm]$$

3.2.2 System torque rating of low-speed coupling

$$T_{LS} = \frac{P_{EF} \cdot 9549}{n_2}\ [Nm] =$$
$$= 49087{,}8\ [Nm]$$

3.2.3 Minimum coupling rating of high-speed coupling

$$T_{HSmin} = s_f \cdot T_{HS}\ [Nm] =$$
$$= 3199{,}2\ [Nm] < T_{HSnom} =$$
$$= 9320\ [Nm]$$

3.2.4. Minimum coupling rating of low-speed coupling

$$T_{LSmin} = s_f \cdot T_{LS}\,[Nm] = = 49087{,}8\,[Nm] < T_{LSnom} = = 103000\,[Nm]$$

Where:

$s_f = 1.00\,[N]$ – service factor for belt conveyors for coupling calculation

$T_{HSnom} = 9320\,[Nm]$ – torque rating of high-speed coupling

$T_{LSnom} = 103000\,[Nm]$ – torque rating of low-speed coupling

4 DISCUSSION

The selected reducer 485-A2 meets in terms of power because the equivalent power of reducer is less than the reducer power rating, i.e. the reducer selected from the manufacturer's catalogue is the first size that exceeds the required power rating. Ratio of selected reducer corresponds approximately to the necessary ratio. Also, the selected reducer meets in terms of thermal rating because the application adjusted thermal rating is greater than the actual power transmitted by the reducer for conveyor belt drive. In case that reducer did not meet in terms of thermal rating, the alternative would be the selection of adequate method of auxiliary cooling and its calculation verification. Auxiliary cooling of reducer may be via shaft or electric fan, via cooling tubes in the reducer itself for oil cooling by water or via separate installation for oil cooling in which oil circulates by pump through air or water cooler placed near the reducer.

The selected couplings 1110T10 as high-speed coupling and 1180T10 as low-speed coupling meet in terms of load because the torque rating is greater than the minimum coupling rating in both cases. It should be noted that at coupling selection, the care was taken about the shaft diameter of connected equipment that corresponds to the bore diameter of coupling, which on the other hand has to be within the certain limits given in the manufacturer's catalogue data (see Table 1). Therefore, although i according to the load criterion, one size smaller coupling can be adopted for both high-speed and low-speed coupling, it was not possible due to the limitations in maximum dimension coupling size.

5 CONCLUSION

The results obtained by calculation have shown that the components of drive unit, i.e. the reducer and couplings are properly designed.

In this case, the verification was done by procedure given in the manufacturer's catalogue data of the Falk Company that, with minor differences, corresponds to the calculation of reducers and couplings of the other manufacturers. Selection the components of drive unit is an integral part of design not only belt conveyors, but for many other devices on the mechanical drive.

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Dragan Zlatanović, Mile Bugarin**, Vladimir Milisavljević***, Vukašin Zlatanović*****

FORECASTING THE FINANCIAL DISTRESS OF MINING COMPANIES: TOOL FOR TESTING THE KEY PERFORMANCE INDICATORS

Abstract

There are numerous studies and research work related to the forecasting of financial distress of companies. Developed theoretical and practical models were used for forecasting such problems. Application of specific model is relatively novel analytical approach and represents an indicator which sometimes could have large importance for decision makers. Indicators for production and business aspects are represented by one of the most suitable synthetic parameters – Altman Financial Distress Ratio, which is sum of weighted individual parameters. The aim of this paper is to present a method for forecasting the financial distress, mainly based on financial parameters of a company. Calculation of financial parameters was based on the public annual financial reports of companies included in the example. Authors applied the Altman Z-score model on sample of two mining companies, to establish accuracy of this model and possibility for application on other mining companies.

Keywords: *financial distress, mining company, Altman Z-score, performance indicators*

INTRODUCTION

Importance for examination of such problems and application of suitable model for forecasting business distress or mining company failure forecast is very important in our commercial situation, since a number of failed companies, including those from mining sector, increases permanently. The research has indicated that there are more bankrupt companies than those which are re-industrialized. Bankruptcy of mining company has a negative impact on overall national economy, therefore this area requires a special attention since it affects numerous beneficiaries. Beside the management team, there are various parties interested in the company business and its future, such as employees, current and potential creditors, suppliers and other users of financial information.

Rescue for failed companies can be found in searching the available founding and interests of potential investors, through privatization or some other form of association which would enable rehabilitation of mining industry.

Mining and metallurgy are extensive industries with low productivity in almost all transition countries, as left behind by the socialist economy. Also, economy transition during last 15 years did not yield the positive effects. Privatization of such companies is complex and specific, mainly due

* *Innovation Centre, Faculty of Mechanical Engineering, University of Belgrade, Serbia; e-mail: dr.dragan.zlatanovic@gmail.com*
** *Mining and Metallurgy Institute, Bor, Serbia; e-mail: mile.bugarin@irmbor.co.rs*
*** *Faculty of Mining and Geology, University of Belgrade, Serbia; e-mail: vladimir.milisavljevic@rgf.bg.ac.rs*
**** *FEFA - Faculty of Economy, Finance and Administration, Singidunum University, Belgrade, Serbia; e-mail: vukasinzlatanovic@gmail.com*

to the exposure to large social pressure. Such companies also have larger national impact, additionally contributing to complexity of privatization [1].

Due to these reasons, the Serbian Government has taken the steps in finding solutions establishing which companies have the "strategic importance" [2].

Strategic importance of these companies is established according to:

1. Positive impact on employment in the region,
2. Positive impact on business of other companies,
3. Market share and market potential,
4. Feasibility of production and business,
5. Product/service of strategic importance for Serbian economy,
6. Privatization certainties in period of one year and
7. Company potential for attracting significant investment.

As it can be seen, these aspects consider both external and internal social and business environment.

There are two mining complexes among the companies for privatization with strategic importance and those are: Public Company for Underground Coal Mining (JP PEU) in Resavica and RTB Bor Group with subsidiaries.

Any further analysis must include the basic assumption that any mine is not feasible without investments. This criteria evaluates the possibility for investment in improvement of technology, transport, infrastructure and similar, i.e. privatization of the mine.

Investment potential, which includes the amount of investment for execution the project, is the result of technical and economic analysis which should be considered as separate criteria due to lack of cheap funding and difficulties to obtain them. The existing mines and mining companies in Serbia have huge problems in finding the investors in environment with controlled selling price of coal, historical debts, outdated infrastructure, inherited environmental issues, variations on metals market, etc.

Some of the previously privatized mines have exhibited the efforts to utilize chances at the open market. In order to use business opportunities, these mines attempted to utilize the controversial market effects without any safety measures. This includes incompetent utilization of resources and exploitation of only those parts of deposit which generate high profit. Such approach resulted in difficult situation for further operation of these mines.

Described conditions had a serious social impact on mining, hence it should be considered in time. Threats to individual industries and their parts in Serbia are not equally distributed, where mining industry is probably in the most difficult situation, resulting in allocation of capital toward industries with larger profit. All mentioned indicators suggest that during privatization process of mining companies the governmental institutions must have significant regulating role. This means that these institutions must be supplied with suitable information which defines the business and social status of each mine and their importance for local community [3].

Approach presented in this paper uses a relatively simple tool for condition assessment and sustainability of production capability of mining company in the domain of financial distress.

SUSTAINABILITY OF CURRENT PRODUCTION AND BUSINESS ACTIVITY

Subject of this paper is related to the sustainability of business activities of the companies in the mining industry. Finances of the mining company can be analyzed by determination of so called Z value of the company. Z value represents quantity indicator of probability for bankruptcy of the company, i.e. it describes the financial strength of the company. Calculation of Z value for two mining companies is given below.

This approach can be justified by the fact that bankruptcy of a company is a foreseeable consequence of economics. Economy strengths and weaknesses are related to the stimulations created by solutions of who takes the burden in case of bankruptcy. Real life suggests that market economy has relative advantage since market identifies troubled companies quickly and cheaply and also distributes the generated losses in manner which does not hamper the economic development. Of course, subsidies to companies which generate losses are usual in developed economies as well, but in strictly controlled manner. Such attitude is also applied in the mining industry, especially to companies related to energetics and metallurgy [4].

Zeta Altman Model for Forecasting the Financial Distress

One of the most common synthetic indicators is the Altman synthetic financial distress ratio, which represents the sum of weighted individual parameters.

Altman Z score is the first model for forecasting the financial distress, which uses model of iterative Multiple Discriminant Analysis. This model is still one of the most popular models for forecasting the bankruptcy. Original model was developed in 1968 for production companies and it forecasts bankruptcy if Z value sets in the specific ranges [5].

Research performed by Edward Altman included 33 failed and 33 successful companies. He developed the following relation, as a result of iterative Multiple Discriminant Analysis:

$$Z = 0{,}012*X1 + 0{,}014*X2 + 0{,}033*X3 + 0{,}006*X4 + 0{,}999*X5 \quad (1)$$

where following ratio are:

- X1 = Current assets / Total Assets

Measure of net liquidity in relation to total assets. Company that generates the operational losses will have reduced current assets in relation to total assets.

- X2 = Retained earnings / Total assets

Fairly new companies will have lower value of this ratio, since they did not have time to generate assets.

- X3 = Operating Profit/ Total Assets

Since the future of company is based on capability to generate profit with own assets, this ratio is suitable for conclusion the possibility for bankruptcy.

- X4 = Market value of Equity / Book Value of Total Debt

This ratio indicates the level of assets value losses before liabilities surpass the assets and company become insolvent.

- X5 = Sales / Total assets

Lower limit is 1.81, meaning that companies with Z score below this limit will go to bankruptcy, while the upper limit is 2.99, meaning that companies above this value will not go to bankruptcy. For values be tween 1.81 and 2.99 the original sample of companies showed mistakes. Accuracy of model discrimination between successful and unsuccessful companies for the period of one year before bankruptcy is 95%, and for the period of two years is 82%.

Altman revised original model in 1993, in such manner to develop a model for private companies also. This was achieved by replacement the Market value by Book value in ratio X4.

Result was the following Z'-score model:

$$Z' = 0{,}717*X1 + 0{,}847*X2 + 3{,}107*X3 + 0{,}420*X4 + 0{,}998*X5 \quad (2)$$

In this case, lower limit is Z'=1.23. Companies below the score value bellow are considered to be those that will surely go bankrupt, and the upper value is Z'=2.90 for the companies with above score value considered as successful. Scores between these values are in "grey" area, and same rules are valid as in the previous case.

Probability for company bankruptcy can be calculated by applying the following formula: 1-EXP (Z score value)/(1+EXP (Z score value)).

Since this assessment is based on financial reports and business information provided by companies itself. These can be arbitrary corrected in relation to the real situation, therefore validity and accuracy of method can questionable.

Table 1 *Mining company "X"*

			Mining company "X"				
	rev.	**Z' = 0.717*X1 + 0.847*X2 + 3.107*X3 + + 0.42*X4 + 0.998*X5**	2010	2011	2012	2013	2014
		Probability for bankruptcy	1.16%	0.54%	0.05%	0.04%	0.07%
		Scores between 1.23 and 2.90 (grey zone)	**4.449**	**5.214**	**7.683**	**7.935**	**7.291**
(Private) production companies	X1 -	Working Capital / Total Assets	X1	X1	X1	X1	X1
			0.333	0.254	0.245	0.348	0.342
	X2 -	Retained Earnings / Total Assets	X2	X2	X2	X2	X2
			0.155	0.227	0.262	0.154	0.225
	X3 -	Earnings before Interests and Taxes / Total Assets	X3	X3	X3	X3	X3
			0.151	0.216	0.239	0.137	0.201
	X4 -	Book Value Equity / Book Value of Total Liabilities	X4	X4	X4	X4	X4
			3.755	4.324	8.598	11.120	8.722
	X5 -	Sales / Total Assets	X5	X5	X5	X5	X5
			2.038	2.358	2.938	2.462	2.573
	Y1 -	Index of fuel and energy costs in total costs	13.01%	13.67%	12.47%	12.55%	13.36%
	Y2 -	Index of labor costs in total costs	25.78%	27.47%	26.94%	29.51%	38.08%
	Y3 -	Average costs per employee (gross) per month (RSD)	42,277	52,460	61,153	62,804	84,308

Table 2 *Mining company "Y"*

			Mining company "Y"				
	rev.	**Z' = 0.717*X1 + 0.847*X2 + 3.107*X3 + + 0.42*X4 + 0.998*X5**	2010	2011	2012	2013	2014
		Probability for bankruptcy	22.8%	12.9%	17.1%	22.6%	39.6%
		Scores between 1.23 and 2.90 (grey zone)	**1.222**	**1.912**	**1.578**	**1.231**	**0.423**
(Private) production companies	X1 -	Working Capital / Total Assets	X1	X1	X1	X1	X1
			0.271	0.441	0.087	0.143	0.049
	X2 -	Retained Earnings / Total Assets	X2	X2	X2	X2	X2
			0.013	0.052	0.050	0.002	0.064
	X3 -	Earnings before Interests and Taxes / Total Assets	X3	X3	X3	X3	X3
			0.006	0.052	0.039	0.010	0.090
	X4 -	Book Value Equity / Book Value of Total Liabilities	X4	X4	X4	X4	X4
			1.156	1.158	0.588	0.609	0.338
	X5 -	Sales / Total Assets	X5	X5	X5	X5	X5
			0.512	0.905	1.107	0.904	0.651
	Y1 -	Index of fuel and energy costs in total costs	11.29%	13.85%	13.11%	11.90%	16.00%
	Y2 -	Index of labor costs in total costs	13.89%	14.44%	15.00%	19.56%	34.67%
	Y3 -	Average costs per employee (gross) per month (RSD)	47,608	57,399	60,229	65,021	97,229

Application of Altman Z'-score Model

Application of Z'-score model is given further on in this chapter, for two privatized mining companies. The purpose was to establish the accuracy of this approach. Z'-score model is applied for the mining company "X" which is successful and the mining company "Y" which is in bankruptcy. Values of Z' were calculated for previous 5 years according to the data publicly available at the Serbian Business Registers Agency and Statistical Office of the Republic of Serbia websites.

Results of Analysis the Altman Z'-score Model

Since Z' value for company "Y" is significantly lower than limiting value of 2.90, it can be considered as the unsuccessful in period of analyzed 5 years. This was proved in real life. Bankruptcy probability of company "Y" 2014 was 39.7%.

Our opinion is that the additional explanations are necessary for ratios used in Z' model and their analysis:

X1 – is a liquidity indicator representing portion of Working capital in Total assets. Research indicated that companies operating with losses show decline of short-term assets in total assets. Company "Y" had this indicator in last year in negative values, meaning that total assets are larger than the operating capital (which represents difference between short-term assets and short-term liabilities). Net working capital is also negative as a consequence of larger short-term liabilities in relation to short-term assets.

X2 – is a ratio of Retained earnings and Total assets of the company, and it indicates a portion of reinvested assets during one calendar year. Retained earnings are considered as best indicator of company growth and sources of financing. This indicator for company "Y" is low, even negative in the last year, meaning that retained earnings are very low and negative in relation to total assets.

X3 – ratio which indicates the profit. This ratio for company "Y" is negative since it has business losses.

X4 – ratio indicates how much the value of company assets could lose its value before company becomes insolvent. When this ratio is lower than 1 total then total liability are smaller than book value equity, as is the case for company "Y".

X5 – ratio is a coefficient of total asset turnaround. It is obvious that company "X" is far better than company "Y" since it turns much of its asset into the business income.

Possibility for Using other Financial Indicators

Tables 1 and 2 are also providing indexes Y1, Y2 and Y3 (beside ratios required for calculation of Z' value). These indexes also can be calculated with publically available information. As already mentioned, these sources are mainly annual reports such as the Balance sheet and profits and losses, which are available at website of the Serbian Business Registers Agency[†] (APR) and statistical annual and monthly bulletins available at website of the Statistical Office of the Republic of Serbia websites[‡] (RZS).

The mentioned Balance sheets of companies are also having following data: cost for fuel and energy, cost for gross salaries and remunerations, as well as the average number of employees. Therefore, it is fairly simple to determine their ratios:

Y1 - Index of fuel and energy costs in total costs;

[†] *http://www.apr.gov.rs/Регистри/Привредна друштва/ДруштваПретрагаподатака.aspx*
[‡] *http://webrzs.stat.gov.rs/WebSite/*

Y2 - Index of labor costs in total costs;

Y3 - Average costs per employee (gross) per month (RSD).

It should be mentioned that same data was obtained for JP PEU and RTB Bor, strategic companies mentioned in the introduction of this paper. Analysis of Z' model for these companies are not presented in this paper. These two companies are for long time having total liabilities larger than total assets. This is the reason why they are in restructuring process and not in bankruptcy. Instead, the indexes Y1, Y2 and Y3 will be presented and compare their values with analyzed companies "X" and "Y" and analyze those.

Table 3 *Indexes of fuel and energy costs, labor costs and salaries in total costs*

	JP PEU – Consolidated balance sheets	2008	2009	2010	2011	2012	2013	2014
Y1 -	Index of fuel and energy costs in total costs	4.37%	4.30%	5.01%	5.12%	4.91%	4.95%	6.38%
Y2 -	Index of labor costs in total costs	49.36%	48.55%	44.67%	43.96%	46.45%	45.82%	69.36%
Y3 -	Average costs per employee (gross) per month (RSD)	49,740	53,730	54,164	61,294	68,981	72,559	94,745
	RTB BOR GRUPA – Consolidated balance sheets							
Y1 -	Index of fuel and energy costs in total costs	24.36%	26.13%	23.08%	24.28%	23.76%	23.90%	27.96%
Y2 -	Index of labor costs in total costs	26.34%	26.27%	22.68%	23.12%	22.57%	21.23%	28.47%
Y3 -	Average costs per employee (gross) per month (RSD)	52,257	55,471	62,722	82,504	95,918	98,689	123,296

As it can be seen in table 3, index Y1 (cost of fuel and energy in total costs) varies for all analyzed companies but is consistent without large deviations from the average values for each individual company.

However, in case of index Y2, which ratio of labor cost and total costs, it is unacceptable that this ratio is up to 70%. This means that there is no production and consequent income which could compensate such a large share of labor cost in total costs. This is additionally confirmed by 5% value of Y1 (share of fuel and energy cost).

Comparing this value for various mines this index is generally between 20 and 30%, which is the reference range for this ratio.

Finally, the largest paradox can be seen in the analysis of gross cost per employee per month (index Y3). The analyzed company "X" has the best financial indicators and it is furthest away from any financial distress, but it also has the lowest average gross salaries in comparison to other companies. Company "Y", which is in bankruptcy, together with other two strategic companies, which are protected from bankruptcy so far, are having negative capital and fairly high average gross salaries. Proper review of this situation shows that on one hand there is a responsible salary policy and on the other hand there is a situation that salaries are higher in proportion to poorer condition of the company. This situation is more typical for social companies than for mining ones. Therefore, it is very dangerous situation to have the bankruptcy of such company, since it could lead to high social turmoil both at local and national level.

Since two strategic companies are employing almost one third of workers in the Serbian mining industry, then they would have significant impact on gross salaries at the industry level. Hence, the question what is the realistic mining salary in the Serbian mining industry remains. Data given in Table 4 clearly show that the salaries in the mining industry are 50% higher than the average salary at the level of the Republic of Serbia.

Table 4 *Indicators of gross salary shares in the Republic of Serbia and mining industry*

Indikatori/Indicators*	**2008**	**2009**	**2010**	**2011**	**2012**	**2013**	**2014**
AVERAGE GROSS SALARY *Average gross salary in RS*	*45,674*	*44,147*	*47,450*	*52,733*	*57,430*	*60,708*	*61,426*
Average gross salary in mining	55,835	61,226	69,582	80,605	89,521	96,051	97,900
- Nominal Growth Index	14.00%	10.00%	13.60%	15.80%	11.10%	7.30%	1.90%
- Real Growth Index	0.40%	1.30%	6.30%	4.30%	3.10%	-0.50%	-1.00%

* *Annual statistical journals*

Graph with comparative overview the gross salaries in Serbian dinars (RSD) at monthly level is shown in Figure 1.

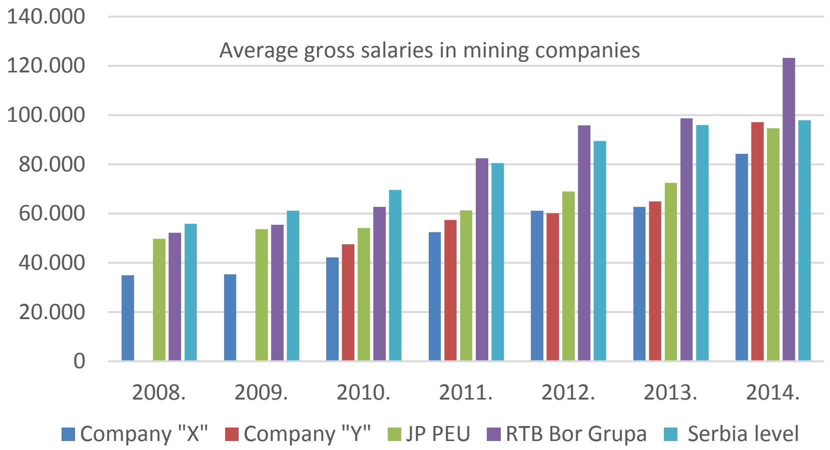

Figure 1 *Average gross salaries in selected mining companies and national level*

CONCLUSION

This paper provides one approach for forecasting the financial distress of mining companies, mainly based on their financial indicators. Analysis was based on publicly available balanced sheets of companies and statistical information. Calculated indexes and values can provide satisfactory presentation about the company, as well as the establishment of key performance indicators that have the importance in decision making process.

This work presents application of Altman Z' score model and sufficient accuracy for its application is shown in mining industry.

Analysis included two privatized mining companies "X" and "Y". Indicators of company "X" showed the business success with very high Z' value which means the financial stability. On the other hand, company "Y" had poor results, with Z' value below lower limit, thus confirming bankruptcy which happened in real world.

Such analysis raises the question why management of company "Y" did not notice the business problems and/or why they did not acknowledge it before it was too late?

As an answer to this issue is a fact that most commonly proper methods and techniques are not available, to provide the red flag on possible problems and upcoming business difficulties, threating the overall business and generate to bankruptcy. Therefore, it is necessary to establish simple and reliable forecasting system of success or failure of mining company. Altman Z' score model surely can be used for this purpose.

Further research could include the analysis of numerous mining companies, as well as possible correlation of Z' value with some technical indicators and other internal and external factors which are not directly related to financial indicators.

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Mladen Radojković, Aleksandra Kokić-Arsić*, Marijola Božović*, Bojan Prlinčević**

CIRCULAR HOLES AS SOURCES OF STRESS CONCENTRATION IN THE PARTS OF MACHINES AND DEVICES USED IN MINING

Abstract

This work discusses the impact of circular forms on stress distribution in the parts of machines and devices used in mining, since it represents a typical source of stress concentration. The stress distribution around the hole, as a source of stress concentration, is significantly affected both by the hole size and type of material and its shape and position relative to the direction of external load effect. The aim of this work is to analyze the impact of circular holes, as a source of stress concentration, on the stress distribution since a large number of parts of which the machines and devices used in mining are made, is impaired by the stress. The work discusses only the parts made of isotropic material, because the practice has shown that most of the parts of which machines and devices used in mining are assembled, are made of isotropic materials. To obtain the results of stresses distribution in a plane isotropic field weakened by the holes, analytical and numerical methods were used.
Keywords: *circular hole, stress distribution, stress concentration, parts, machines and devices*

1 INTRODUCTION

The intention of each country is, as far as possible, to develop and expand its raw material base. Mining industry has a task of supplying an economy with the raw material in various forms. The supply of these raw materials is realized both by underground and surface mining exploitation the deposits of raw materials. Underground mining of mineral deposits involves obtaining the mineral raw material by the underground operation using the appropriate machines and devices and appropriate excavation method. Machines and devices in the underground mining are used for excavation, loading, transport, supporting elements of excavated area and its filling, and for making the underground chambers. Surface mining of deposits is significantly different from the underground, both in technological process of mining deposits, and in the application of appropriate machinery. Surface mining represents a set of all works from the ground level to the unearthing of deposits for useful mineral raw materials, and a facility that is thus formed is called an open pit mine. Surface mining involves two basic groups of works: works on stripping waste rock (tailings) and works on useful mineral raw materials. Work on the overburden consist of the excavation, transportation and disposal of overburden tailing mass that cover, i.e. prevent free access and safe exploitation of useful raw materials. Works on useful raw material consist of the excavation, transport, transfer or storage of useful mineral resources. For all these works, it is necessary to use the appropriate machinery, machines and equipment [1].

Mining machines and devices have the built-propulsion systems, power transmis-

* *High Technical School of Professional Studies Zvečan,* mladen.radojkovic.zp@gmail.com

sions, working bodies, control devices and brake systems which are composed of a large number of mechanical parts that make up a functional unit. In most cases these parts are complex geometric shapes. Bearing in mind the fact that each of these parts may have at least one zone in which there is a discontinuity, and then the issue covered by the title of this work becomes even more relevant.

In engineering practice, the knowledge of stress distribution and deformations is of great importance in the areas of geometric discontinuity the elements of machine design. Tests have shown that the stresses on the change of cross-sectional areas of machine parts increase, and all the more so if the change of cross-section is more intense. Such an occurrence of increasing stress on positions the cross-section change is called the stress concentration. Notches, grooves, passes from one width to another or from one diameter to another, holes and openings are typical sources of stress concentration.

The basic and most important task in design of complex mechanical structures, or parts of it, is the proper consideration of all parameters that influence the stress state. This approach involves defining the critical zones within the structure or its parts. These are, among others, the source zones of stress concentration around the holes of different shapes and positions. Solving problems in this case comes down to resolving differential equations, ordinary and partial, while satisfying the appropriate boundary and initial conditions [2-5].

For a more detailed calculation the stress, in order to take into account all influential factors in the field of machine elements weakened by the openings numerical methods, were used for research in this work. To simulate the real operating conditions (load, boundary conditions), the numerical methods were also used. Given the nature of the problem that is being addressed in this paper, finite element method (FEM) was chosen and two-dimensional (plane) finite elements were used [6-7]. As numerical methods involve the use of computers and related software packages for generating a finite element mesh ANSYS software package was used herein [14].

A large number of structures, in order to ensure the conditions and meet the demands of strength, were made of isotropic materials. Isotropy is a feature of some bodies in different directions to show the same physical and mechanical properties, for example elasticity, stress, strain, etc. Many construction materials from which the parts of machines and equipment in mining are made of have the isotropic characteristics.

When it comes to research the phenomenon of stress concentration around a circular hole in the majority of cases, a thin plate weakened by a circular hole loaded with a certain load is analyzed [8-13]. The problem narrows down to the exact determination of stress distribution around the hole, as well as determination the stress concentration factor.

In this work, a special attention will be paid to the study of distribution the stresses in areas weakened by the circular hole, made of isotropic material and exposed to the static load.

2 DETERMINATION THE STRESS IN PANEL USING THE THEORY OF ELASTICITY EQUATIONS

In practical problems, it is often the case that a part of structure is burdened with external load so that only stresses occur in it and then it is spoken about the flat (planar) strain, or only deformation in two mutually perpendicular directions when it comes to flat (planar) deformation. Solving problems in this case comes down to resolving differential equations, ordinary and partial, while satisfying appropriate boundary and initial conditions [4, 5].

Starting from the aforementioned equations of the theory of elasticity, the expressions are derived in literature [3, 8], which define the stress state as the uniaxial strained homogeneous isotropic plate weakened by a circular hole in polar coordinates:

$$\sigma_\rho = \frac{p}{2}\left[\left(1-\rho^2\right)+\left(1-4\rho^2+3\rho^4\right)\cos 2\theta\right] \tag{1}$$

$$\sigma_\theta = \frac{p}{2}\left[\left(1+\rho^2\right)-\left(1+3\rho^4\right)\cos 2\theta\right] \tag{2}$$

$$\tau_{\rho\theta} = \frac{p}{2}\left(1+2\rho^2-3\rho^4\right)\sin 2\theta \tag{3}$$

where:

p – surface forces of plate stretching,

ρ – polar pull measured from the hole center,

θ – polar angle.

The expression for calculation the stress to the contour of circular hole is obtained if in (1), (2) and (3), $\rho = 1$:

$$\sigma_\theta = p\left(1-2\cos 2\theta\right) \tag{4}$$

From (4), it is shown that maximum values of stress $\sigma\theta$ are obtained at cos $2\theta = -1$, i.e. at $\theta = \pm\,\pi/2$:

$$\left(\sigma_\theta\right)_{\max} = 3p \tag{5}$$

According to the literature [3, 8], the expressions to calculate the stresses σ_{max}, σ_{min} and τ_{max} will be determined by the following formulae:

$$\sigma_{\max} = \frac{\sigma_\rho + \sigma_\theta}{2} + \sqrt{\left(\frac{\sigma_\rho - \sigma_\theta}{2}\right)^2 + \tau_{\rho\theta}^2} \tag{6}$$

$$\sigma_{\min} = \frac{\sigma_\rho + \sigma_\theta}{2} - \sqrt{\left(\frac{\sigma_\rho - \sigma_\theta}{2}\right)^2 + \tau_{\rho\theta}^2} \tag{7}$$

$$\tau_{\max} = \pm\sqrt{\left(\frac{\sigma_\rho - \sigma_\theta}{2}\right)^2 + \tau_{\rho\theta}^2} \tag{8}$$

Based on the expressions (1-8), an algorithm is formed in the software package Microsoft Office Excel and the values of stresses σ_{max}, σ_{min} and τ_{max} are calculated. Figure 1 shows: *a)* lines of equal σ_{max} and σ_{min}, *b)* lines of equal τ_{max}.

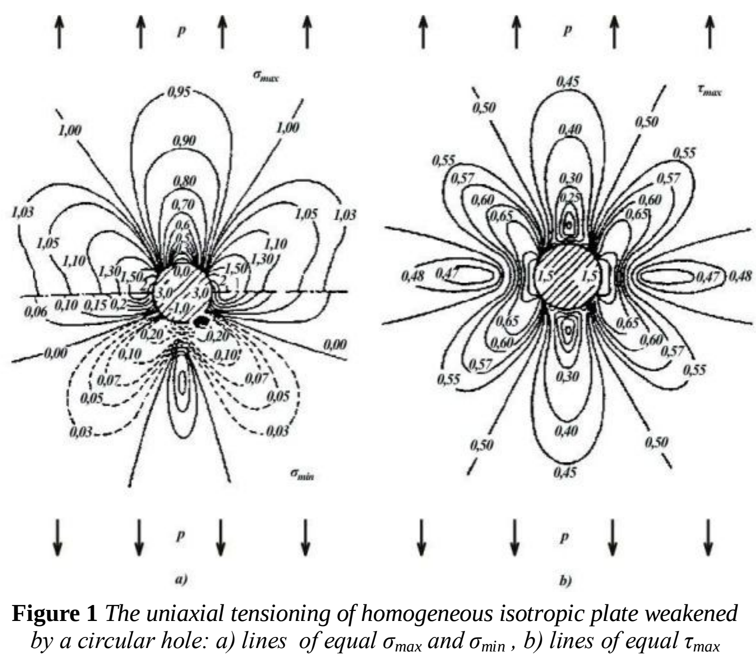

Figure 1 *The uniaxial tensioning of homogeneous isotropic plate weakened by a circular hole: a) lines of equal σ_{max} and σ_{min} , b) lines of equal τ_{max}*

3 DETERMINATION THE STRESS ON PANEL USING THE FINITE ELEMENT METHOD

To determine the value of the stress and in order to take into account all the influential factors in the field of machine elements weakened by the circular opening, as well as to simulate real operating conditions (load, boundary conditions), numerical methods can be successfully used. Given the nature of the problem that is addressed in this work, the finite element method (FEM) was chosen. The model was discretized using 2D triangular solid finite elements. A detailed overview of equations is given in [7, 8], and here some of the most important relationships will only be given.

Components of surface forces are the vector F_n elements:

$$F_n = \begin{Bmatrix} F_{nx} \\ F_{ny} \end{Bmatrix} \tag{9}$$

At any point of observed body, the shift vector *s* with the displacement components *u* and *v* in the direction of coordinate axes *x* and *y*, is shown as:

$$s = \begin{Bmatrix} u \\ v \end{Bmatrix} \tag{10}$$

The links between the displacement vector *s* and the deformation vector ε can be represented by the Cauchy's revolutionary kinematic equations in the form:

$$\varepsilon_x = \frac{\partial u}{\partial x}, \quad \varepsilon_y = \frac{\partial v}{\partial y},$$

$$\varepsilon_{xy} = \frac{1}{2}\left(\frac{\partial u}{\partial y} + \frac{\partial v}{\partial x}\right) \tag{11}$$

According [7, 8], the relation between the deformation vector ε and displacement vector *s* can be displayed in the matrix form:

$$\varepsilon = d\ s. \tag{12}$$

The matrix of differential operator *d* and its transposed matrix d^T have the forms that can be found in [7, 8].

Stress state in the given point of stressed body is determined by three component stresses as follows: two normal (σ_x and σ_y) and tangential ($\tau_{xy} = \tau_{yx}$) which operate at that point. Conditions of balance between the internal and external forces at the part of contour where the contour conditions are specify by the surface forces are given by the Cauchy's revolutionary equations (Cauchy's boundary conditions):

$$d_s^T \sigma = F_n \tag{13}$$

where: d_s^T- the transponed matrix of the matrix d_s [7, 8].

The general form of constitutive equations, i.e. the relationship between the matrix of the components of stress tensor and matrix components of strain tensor of elastic material is given by the formula:

$$\sigma = D\ \varepsilon, \tag{14}$$

where: *D* - is the matrix of material stiffness, which for homogeneous isotropic elastic materials can be expressed through the Young's modulus of elasticity *E* and Poisson's ratio μ hence the name for this matrix “the matrix of elastic constants, or matrix of the tensor of elasticity” [7, 8].

The basic equation of finite element, i.e. equilibrium equation, which gives the relationship between the nodal displacements and nodal forces, can be written in the form of:

$$F = K\ S \tag{15}$$

where:

K - finite element stiffness matrix.

The following section presents the results of stress distribution σ_{max} and σ_{min} in a plane isotropic field weakened by a circular opening. The results refer to the element of plate type, whose sizes are in this case 2 m × 5 m × 0.1 m, and the same is weakened by the circular opening of diameter d = 100 mm. The load is uniaxial, in the direction of y axis, and is p = 1 [N/m^2]. Steel is the material of which the plate is made with the elasticity modulus E = 2.1x10^5 [MPa] and Poisson's coefficient of μ = 0.33. In the case 2D triangular solid finite elements are used.

Figure 2 shows distribution of the highest stresses σ_{max}.

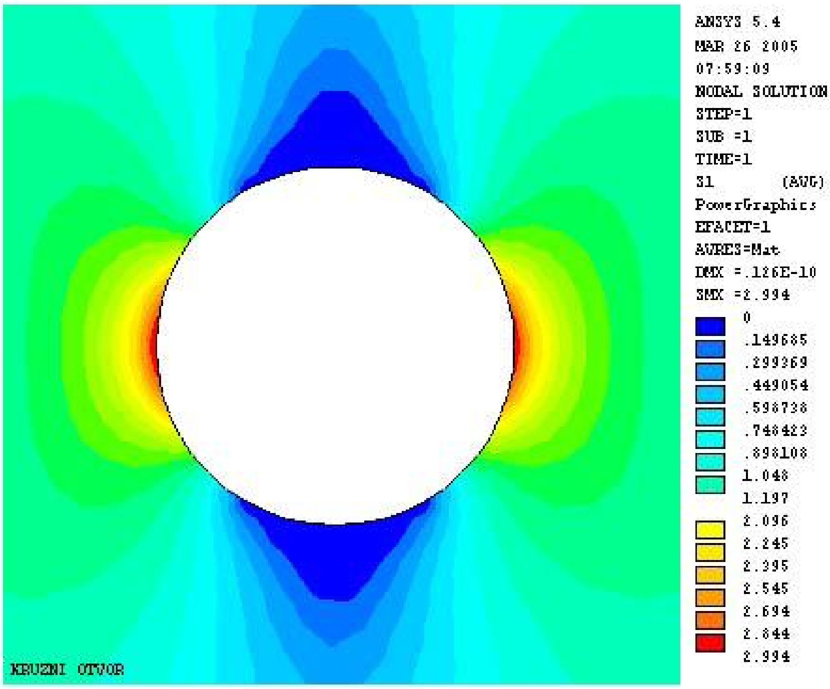

Figure 2 *Distribution of stress σ_{max} on the panel weakened by the circular hole of diameter d = 100 mm and uniaxially strained along the y axis*

It can be seen from Figure 2 that the greatest stress values are at the intersection of contour of hole and x-axis (red color of the stress field in Figure 2).

Maximum stress value in this case is σ_{max} = 2,994 [N/m^2].

Figure 3 shows distribution of normal stress σ_{min}. It can be seen from Figure 3 that the highest stress values are at the intersection between the contours of hole and y-axis (blue stress field in Figure 3). Maximum value of this stress is σ_{min} = -0.994333 [N/m^2].

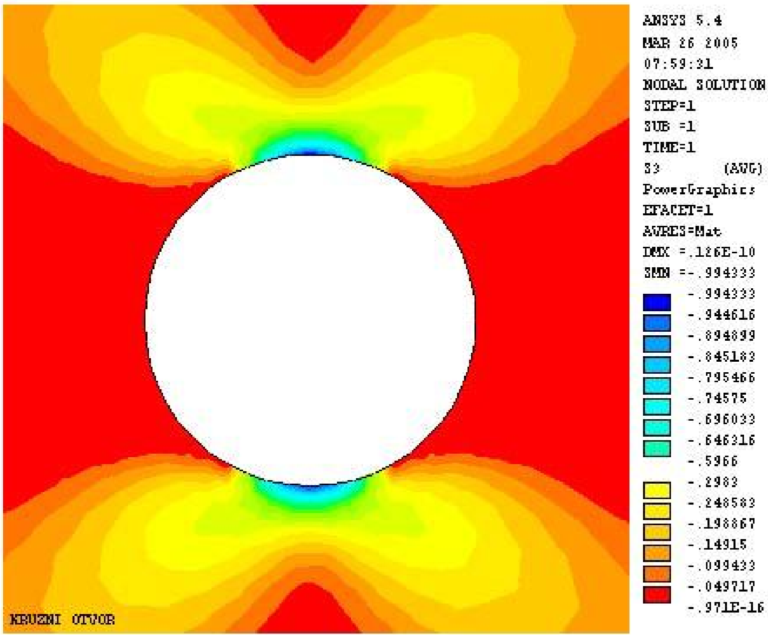

Figure 3 *Distribution of stress σ_{min} on the panel weakened by the circular hole of diameter d = 100 mm and uniaxially strained along the y axis*

4 COMPARISONS OF ANALYTICALLY AND NUMERICALLY OBTAINED RESULTS

In order to verify the numerical results obtained by the finite element method (FEM), the results obtained analytically were compared. Comparison of the obtained values of normal stresses σ_{max} and σ_{min} for uniaxial strained homogeneous isotropic plate along the *y* axis, weakened by the circular hole of diameter *d* = 100 mm was done.

Table 1 presents the highest values of components of the normal stress σ_{max} and σ_{min} obtained by finite element method (FEM) and by analytical manner.

Table 1 *Values of stress σ_{max} and σ_{min}*

Method	σ_{max}	σ_{min}
FEM	2,994	-0.994
ANAL	3,000	-1.000

Based on the results obtained by the finite element method (FEM) and analytical way, it can be seen that the highest values of stress σ_{max}, as expected, are obtained on contour of hole and in the points of contour intersection of hole and x-axis. In those points, the cracks will first occur during uniaxial strain of parts weakened by the circular hole, which can later lead to the breakage of parts, and even accidents on machines and equipment used in mining. The highest obtained values of stress σ_{min} are on contour of hole, the points of contour intersection of the hole and the y-axis, which should be taken into account in designing a uniaxial pressed parts weakened by a circular hole.

5 CONCLUSION

In this paper to determine the image of the stress state in mechanical parts of which machines and equipment used in mining are assembled analytical and numerical methods are used. Analytical problem solving, for more complex problems may be difficult or even impossible often due to bulky mathematical procedure and deadlines that are set in solving problems. Therefore, in this work, the analytical methods are used only for obtaining the results of stress distribution in simpler problems. Numerical method, i.e. finite element method, presented in this paper, has enabled to reach certain results which are certainly important for the future work in the field of research the stress concentration. The analysis of the results obtained by the finite element method and using the computer program ANSYS, confirms the usefulness and justifies the use for solution the similar problems in design.

These studies demonstrated the influence of circular holes on the stress state, particularly on a very contour of the hole. Based on the results, reached in this paper, it can be concluded that any change in shape causes a disruption in the stress state of strained element and stress concentration. Therefore, in designing the machinery and equipment used in mining, special attention should be paid to the construction of parts weakened by the holes, prestressed both at the strain and pressing, as well as to the other types of strain.

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Stojance Mijalkovski, Zoran Despodov*, Nikolinka Doneva*, Vancho Adjiski***

MODERN TRENDS OF GEODETIC MEASUREMENTS IN THE UNDERGROUND MINE "SASA" OF LEAD AND ZINC ORE

Abstract

This paper gives an overview of development the modern trends of geodetic measurements in the underground mine of lead and zinc ore "Sasa" in Makedonska Kamenica. The precise geodetic measurements are of crucial importance, especially in the mines with underground exploitation. The precise geodetic data have a very big impact on safety in preparation the underground mine facilities, as well as in every working process in the mine exploitation.

Keywords: *geodesy, mining, measurements, instruments*

INTRODUCTION

The application of geodesy in mining, especially in mines with underground exploitation, with all its specificities is a big challenge for any surveyor in every sense of the word. All the measurements, calculations, tools and displaying the measured data in the underground mines, are not very different from those methods applied on the surface. The difference is that the conditions, in which the measurements are carried out in the underground mines, require full attention of the surveyor both in terms of safety for him and his co-workers as well as for the equipment.

1 DEVELOPMENT OF MODERN TRENDS OF GEODETIC MEASUREMENTS IN THE UNDERGROUND MINES

The faster development of computer technology, as in all areas of everyday life, inevitably brought revolutionary changes in geodesy. These development changes are not only in geodetic measurement technology, but also in processing methods and how the data is displayed. Surveying instruments are themselves mini computers with their speed and accuracy allow skipping and aceleration of many steps leading to the final product (maps and plans). All this developments greatly facilitate the work of surveyor and reduce the required to spend in the underground mining facilities to carry out the necessary measurements [3]. As the main representative of the new technology, the total station LEICA TCR (800) (Figure 1), which fully meets all needs of the surveyor in performing the geodetic measurements in the underground mines [1]. The underground mine "Sasa" use this total station LEICA TCR (800) for geodetic measurements (Figure 1).

* *University "Goce Delcev", Faculty of Natural and Technical Sciences, Mining Institute, Stip, R. Macedonia*
** *Lead and Zinc Mine "SASA", M. Kamenica, R. Macedonia*

Figure 1 *Total Station "LEICA TCR" in the underground mine "Sasa"*

2 APPLICATION OF TOTAL STATION "LEICA TCR" IN THE UNDERGROUND MINE "SASA"

The big advantage of using this surveying instrument is that it does not require any trigonometric forms for registration the measured values (angles, lengths), because this instrument stores, processes and outputs the obtained data coordinates and elevations of all measured parameters [2]. For the purposes of "translating" the measured data, the instrument uses the appropriate software "Leica Office Tools", which is actually a part of the whole package. The measured data from the instrument to the computer are received in the form as in Figure 2.

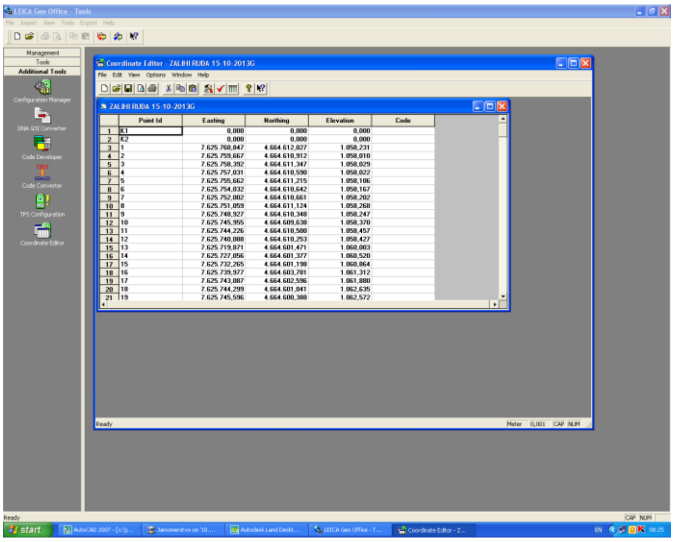

Figure 2 *Processing data from instrument using the software "Leica Office Tools"*

Furthermore, data are processed in the "Text format" and as such are imported using the CAD program; in our case it is "Autodesk Land", where in the working area of software, data are obtained in the form of: symbols, point numbers and elevations of the items in the appropriate coordinate system (Figure 3). It should be noted that in data importing, an option can be selected that includes:

1. Plane surface by omitting the elevation (H), which still remains as numerical data visible on screen
2. Three dimensional coordinate system with three coordinates (X, Y, Z).

If the elevations of points are not necessary, it is best to choose the first option because if the second option is chosen, the lengths that are measured will be inclined (not reduced) and it can give a false representation in the plane of the map or plan.

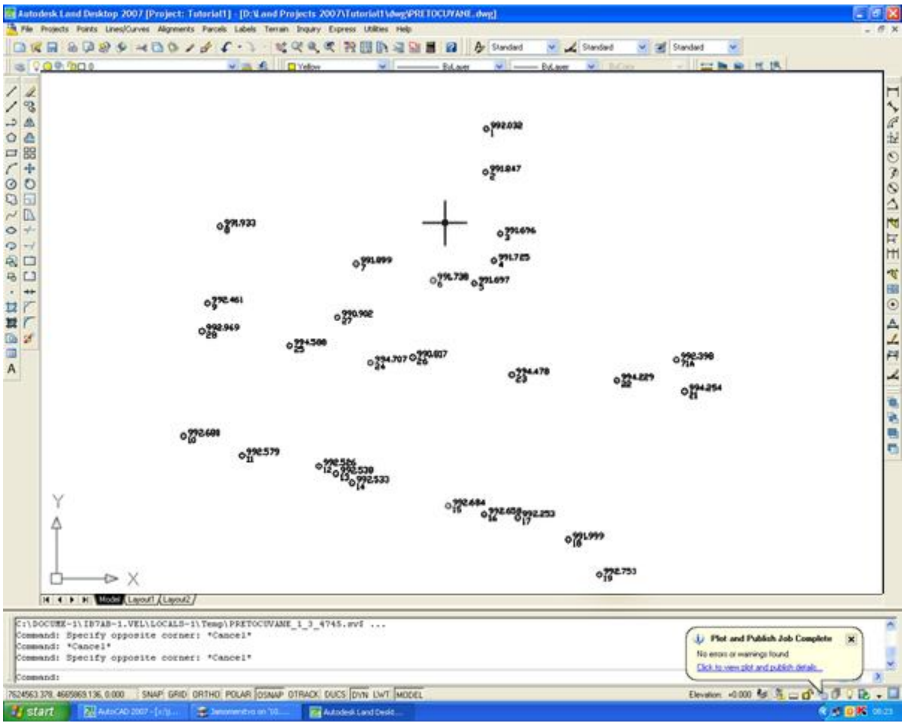

Figure 3 *Importing data into CAD program "Autodesk Land",*

With the points obtained in graphical shapes, the objects are formed simply by connecting them and thus getting a complete view of measured object, as shown in Figure 4. To connect the points with the obtained elevations and shape, the software "AutoCAD" can be used.

This method actually achieve the main goal of geodesy and its application in the underground mines, and that is to give more realistic and accurate representation to all underground mining facilities to all engineers working in the mine and to receive the quality geodetic maps in the electronic form for carrying out their obligations under the set tasks.

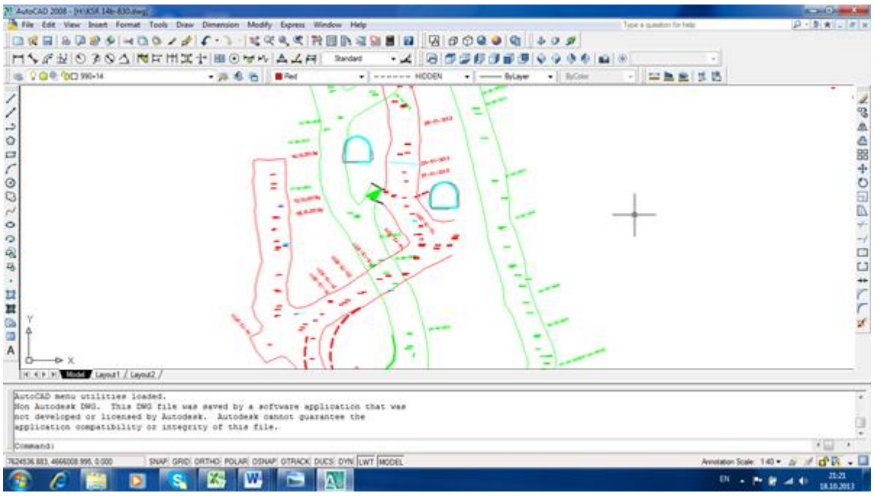

Figure 4 *Formation of objects in "AutoCAD"*

2.1 Development of Plans and Maps

Advances in preparation of maps and plans as the end products of survey are primarily the graphic accuracy of the map or plan, which was always questionable due to many reasons such as: the scale, type of paper, the method of storage, deformations of plane surfaces, geodetic instrument for drawing, etc. The electronic representation of maps or plans (Figure 4) is free from all these problems both in the preparation and in their further use.

The historical development of these plans and maps is exactly the same as the preparation of maps and plans for surface measurements, such as cadastral, urban planning, construction and other needs, starting from different types of paper, tracing paper, paper pasted on aluminum foil, to today's electronic programs for displaying maps and plans, and their quick and easy reproduction in any size (Figure 5).

Figure 5 *Maps and plotter for printing*

Although perhaps with a little dose of mistrust it may be talked about maps and plans of the past, however, these are wonderful works which are made with a lot of effort, precise standards and rules so with all of its advantages and disadvantages they are also used as needed in everyday operation [4, 5].

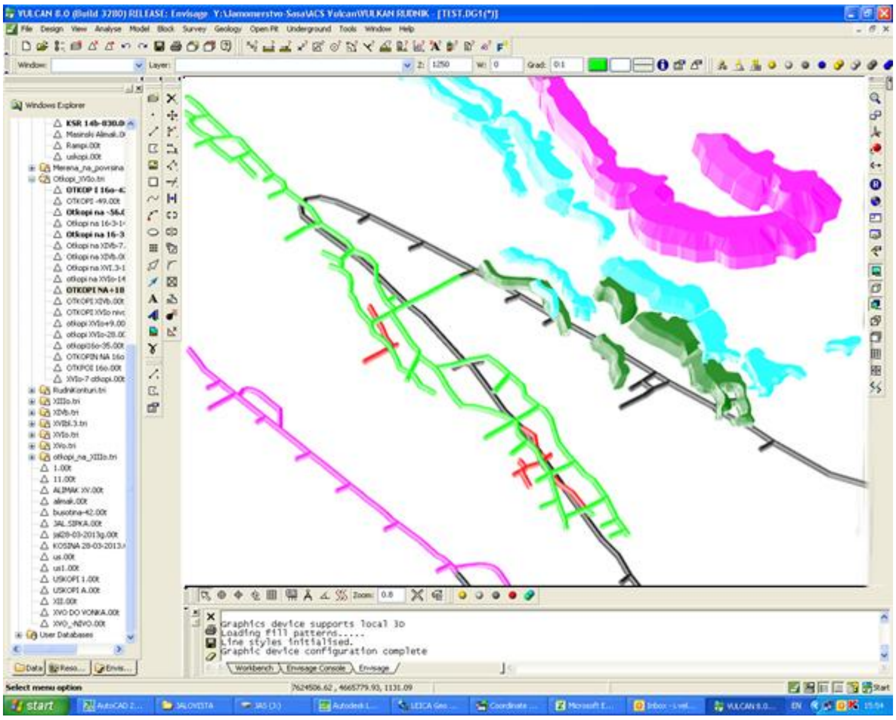

Figure 6 *Three dimensional representation of underground mining facilities*

3 THREE DIMENSIONAL REPRESENTATION OF UNDERGROUND MINING FACILITIES

Another step further to more realistic representation of underground mining facilities is their three dimensional representation (Figure 6).

From a large range of software packages designed for this purpose (PROMINE, VULCAN, SURPAC, DATAMINE and others), the initial experiences promise still more revolutionary changes in display the underground mining plans and maps. This is of particular importance because the actual representation of underground mining facilities is very important in planning of any new mining development. The underground mine "Sasa" used for this purpose in the past the software "Promine", in addition to the "AutoCAD", and today the upgraded to the software "VULCAN" [2].

Proper development of these 3D models in greatly facilitated the preparation of profiles in any given direction, as well as calculation of surfaces and volumes of the ore zones. When it talking about making the correct three dimensional representation of underground mining facilities, it is emphasized that a care must be paid because theprograms work on their own algorithms and mathematical relations and there are often inappropriate choice of the points on which the model can give a misleading view of the object.

In the last few years, geodetic measurement technology went even further with GPS technology, which uses satellites for positioning anywhere on the earth with pretty solid accuracy (Figure 7), which, however, unfortunately cannot be used for measurements in the mines for underground exploitation.

Figure 7 *GPS satellites for positioning objects*

CONCLUSION

Despite great advances in geodetic equipment and computer technology, the fact remains that the underground mine surveyor works in very specific conditions. Today, the underground mines are much safer as opposed to years ago in terms of reliable ways for safe support of underground mining facilities, ventilation, safety measurements at work, etc., but we must unpredictable nature of all underground mines must never be forgotten.

However, the thought of being able to see parts of the planet that were created millions of years in the past, causing a certain amount of excitement and privilege to see things that few people can see. In some cavities between the ore zone there are beautiful crystals that cannot be described in words and forms that geometry that does not know.

It is interesting that even in the mines working people of different professions, but they are primarily the miner, and then everything else.

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Simeun Marijanac, Nenad Makar*, Branka Jovanović*, Pavle Stjepanović**

MAINTAINING THE FUNCTIONALITY OF THE COAL SETTLING BASIN AT THE UNDERWATER PIT KOVIN

Abstract

Underwater coal exploitation is carried out by the use of underwater excavator "Kovin I". The mined coal is transported through a pump system as a hydro mixture via the pipeline Ø900 mm to the facility for irrigation and classification of the mined coal. From this facility, the undersieve goes to the coal settling basin, where in the area of over 32 ha the complete sediment settling is done. Considering that the material durability is limited, this paper presents a way to empty, i.e. maintain the coal settling basin.

Keywords: *coal, underwater exploitation, settling tank, Kovin*

INTRODUCTION

Exploitation of mineral ore in the coal mine Kovin began in the second half of 1991. Work began with the excavation of overburden (sand, gravel and clay), and since 1995 the coal exploitation has begun. All mineral ore are excavated under water by an underwater dredge and deposited on the island Dubovačka ada through a pipeline system. The settling tank of coal and other under sieve sediments made during the coal exploitation is one of the base infrastructure objects in the Kovin mine. Its function is to take and settle the under sieve fractions of coal and other sediments that are a product of the underwater excavation method and coal classification. The settling tank was designed in the original variant of underwater mine operation as an integral part of the coal depot from which it was separated by a dividing mound and connected by overflow pipes. Its function was to direct the excavated coal hydraulically directly in the depot, and to settle the sludge and water created by decantation from the overflow pipes.

By giving up on discharging of the excavated coal without previous technological equipment and construction the facility for taking and coal classification in 1995, the role and function of settling tank have, besides its surface of 32 hectares, changed. The present role of settling tank is to take in, select and drain small granulations (-5+0 mm) created as the under sieve fractions from the hydro mixture of coal, sand and clay during coal exploitation. This paper presents a method of emptying the settling tanks. After an analysis of deposited material in settling tanks that was conducted in the period from November 2013 to February 2014, the settling tanks were divided into zones. This paper presents the solution for material exclusion for the purpose of maintaining the functionality of the settling tanks. This solution should define the possibilities of regular settling tank maintenance. Technical solution of partial settling tank discharge creates the possibility of further mine operations and taking in the new under sieve

* *Mining Institute, Belgrade, Serbia, simeun.marijanac@ribeograd.ac.rs*

sediments. This method of settling tank discharge increases the basin utilization, and more positive financial effects are achieved through it.

TECHNOLOGICAL EXCAVATION PROCESS OF MINERAL ORE IN THE COAL MINE KOVIN

Overburden Excavation

Overburden excavation is done by the underwater dredges. Sediments (sludge, sand, gravel and clay) which go 22 m deep are excavated by the underwater bucket dredge, and the excavated material is laded and transported to a location designated by the project. The sediments that are at a greater depth are excavated by an ejector reffiler "Kovin I", and transport is done through pipelines and deposited in already prepared cassettes.

Coal Excavation

Coal exploitation is done exclusively by the underwater – ejector reffiler "Kovin I". The excavated coal is exclusively transported by the dredge through its pumps as a hydro-mixture through a pipeline Ø900mm to the primary dewatering and classification facility.

The basic characteristics of the "Kovin I" dredge are:

- cutting wheel diameter (4.5 m.);
- maximum excavation depth (45.0 m);
- installed power (11,3 MW);
- pylon stepping (4.5 m);
- speed of circular motion (0-20 m/min).

The guaranteed hourly dredge capacities, given by the manufacturer are:

- sand (2,300 m^3/h);
- gravel (1.300 m^3);
- coal (600 t/h).

Dewatering and Coal Classification

The primary dewatering and coal classification facility (EHS III) is a part of the basic exploitation system, and connected to the pipeline via the "Kovin I" dredge, so that it presents a unique technological-production whole. Dewatering and classification of trench coal hydro-mixture is done in the facility, and commercial assortments are provided:

- piece-cube, i.e. "Marl piece" of the class size -300+30 mm;
- nut, class size -30+15 mm;
- bean, size class -15+5 mm;
- under sieve product (-5 (7) + 0.0 mm), which represents the mixture of small coal, clay, sand and water.

The commercial product makes up to 57 % of entry mass (trench coal), while the rest is made up of the under sieve product 31 % and excavation losses 12%.

Additional Coal Refining Facility "Kučuk Plant''

This facility refines (the under sieve product), mixture of coal and tailings from the EHS-III facility. The average annual capacity of this facility is 40 t/h on entry, i.e. up to 20 t/h of the final product. The coal refinement facility in technological terms presents a facility for gravity concentration in a hydro-cyclone with a heavy environment. The maker of this equipment is the "Kučuk Makina", a company from Turkey. The material is brought via trucks from the settling tank and unloaded into a bunker of around 20 m^3 in volume. The bunker is covered by a grid with openings 400×400 mm in order to prevent insertion of pieces that are larger than 400 mm into the process. The bunker is emptied through a belt feeder which brings the coal onto a vibrofeeder. The class of +100 mm presents the tailings (clay pieces) which are taken in by a conveyor belt and deposited on depot.

Settling Tank Function

Settling tank is a hydro-construction facility where the under sieve product of the process of primary dewatering and classification is deposited for the purpose of deposition (settling) the solid stage and water overflow into the Danube through five overflow pipes. The hydro-mixture enters the settling tank in its western part, and clear water flows over in the east part of settling tank. The settling tank was originally designed and constructed as an integral system of the coal depot from which it is separated by a divider mound and connected with overflow pipes. Its function was to dewater the excavated coal that is hydraulically directly deposited in the depot, and settles the sludge and water made by dewatering from the overflow pipes. By giving up on discharging the excavated coal without previous technological equipment and construction the facility for taking and coal classification in 1995, the role and function of settling tank were, besides its surface of 32 hectares, changed. The present role of the settling tank is to take in, select and drain small granulations (-5+0 mm) created as the under sieve fractions from the hydro mixture of coal, sand and clay during coal exploitation. The constructed depot was retrofitted and serves to take in and deposit the produced coal classes.

Current Settling Tank Condition

The settling tank is a facility in which the under sieve product of dewatering and classifying process for the purpose of deposition (settling) of the solid stage and water overflow in the Danube through five overflow pipes is deposited. The hydro-mixture enters the settling tank in its west part, and clear water overflows at the east part of the settling tank. The settling tank can be divided in the following zones, shown in Figure 1, and characteristics are given in Table 1.

For the purpose of maintenance the settling tank, it is necessary to obtain the following mechanization: dragline dredge, trench excavator, loader, bulldozer, tipper and a grader.

Table 1 *Characteristics of individual settling tank zones*

<table>
<tr><td rowspan="10">Wet part of settling tank</td><td rowspan="2">Active part of settling tank</td><td>Zone 1</td><td>Drainage canal. The materials with a high sand content whose abrasion negatively impacts the refining facility. The material from this zone is treated as tailings.</td></tr>
<tr><td>Zone 2</td><td>Canals in the settling tank. The zone from which the material refined in the Kucuk Makina is excluded</td></tr>
<tr><td rowspan="5">Unavailable part of settling tank</td><td>Zone 3</td><td>The zone in the immediate extension of drainage canals. With the existing mechanization material exclusion is impossible.</td></tr>
<tr><td>Zone 4</td><td>The zone between the canals and overflow pipes. The materials inaccessible to the existing mining mechanization. Zone covered with cane.</td></tr>
<tr><td>Zone 5</td><td>Zone between the canals and overflow pipes. Material inaccessible to the existing mining mechanization. Zone covered with cane and weed.</td></tr>
<tr><td>Zone 6</td><td>Zone of water mirror</td></tr>
<tr><td>Zone 7</td><td>Zone in which the material from EHS and overflow from the new settling tank is indirectly settled</td></tr>
<tr><td rowspan="3">New part of settling tank</td><td>Zone 8</td><td>Part of the new settling tank with a great sand content. The material is treated as definite tailings.</td></tr>
<tr><td>Zone 9</td><td>Part of the settling tank from which coal dust is taken and sold.</td></tr>
<tr><td>Zone 10</td><td>Zone of the new settling tank with a high clay content - inaccessible part, covered in cane. Currently treated as waste.</td></tr>
<tr><td colspan="2">Dry part of settling tank</td><td>Zone 11</td><td>Location in which the materials, which will be refined during the year, is temporarily deposited.</td></tr>
</table>

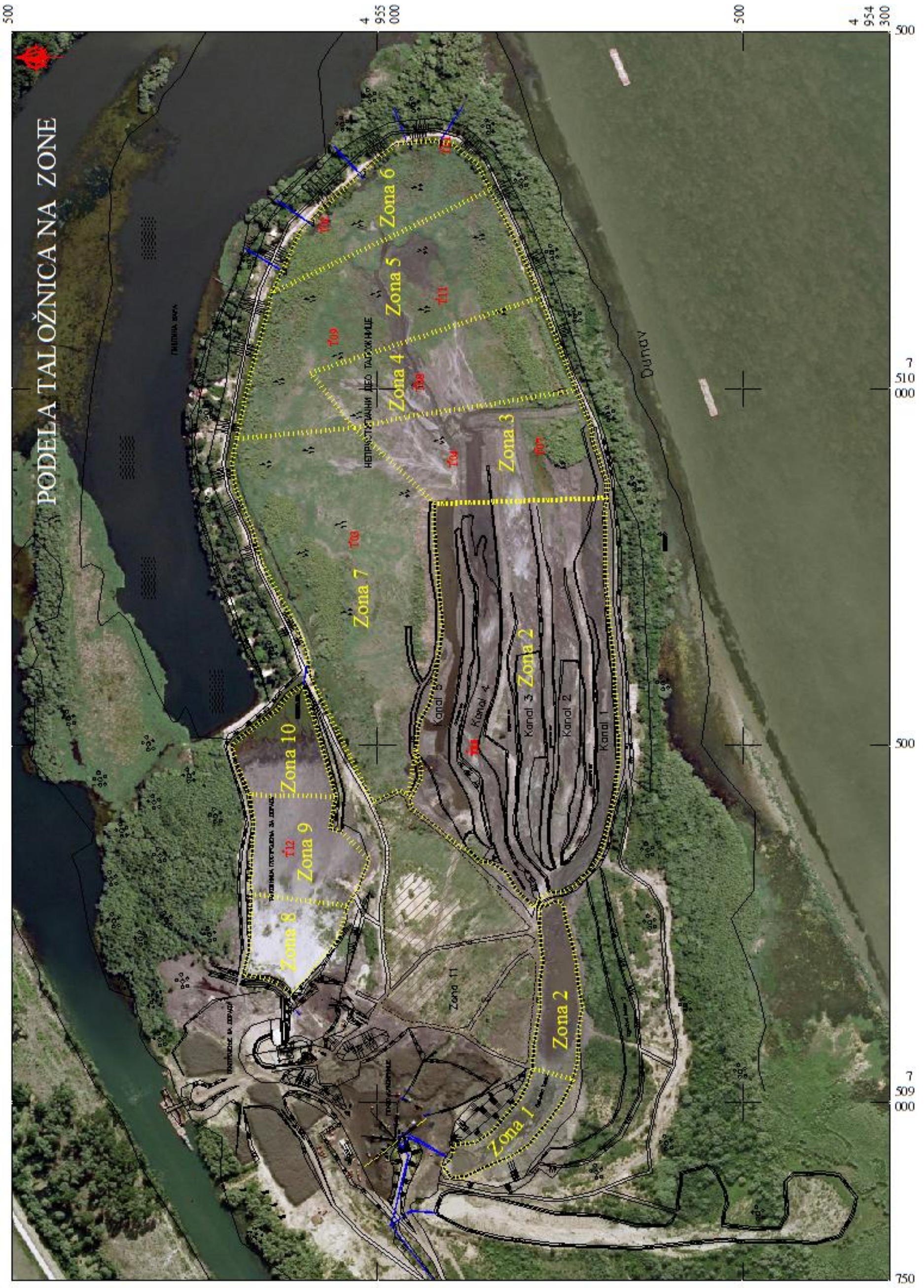

Figure 1 *Settling tank with zoned locations*

SEPARATION AND MATERIAL TRANSPORT FROM SETTLING TANK

This paper presents the operation technology of the excavation transporting mechanization at the separation of settled sediments in the settling tank area, in zones 5, 6 and 7. Material excavation from the settling tanks consists out of three stages.

First stage: *Excavation via spoon dredge with truck transport*

In this stage the hydraulic spoon dredge on the mound excavates the material from settling tank in full width and unloads it onto trucks. The material is deposited in a specially prepared area – the cassette.

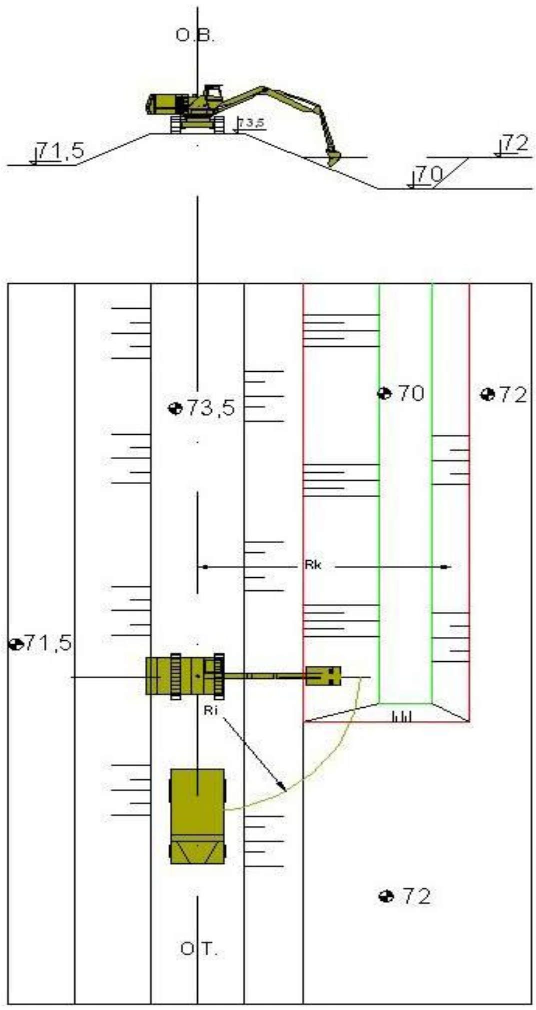

Figure 2 *First stage of separation and material transport*

Second stage: *Dragline and spoon dredge excavation with loading onto trucks*

In the second stage when the spoon dredge and trucks are removed, the dragline dredge goes into operation with digging range is 25 m. The dragline also operates from the mound of settling tank and transports the material from the full width into the already excavated area that was excavated by the spoon dredge in the first stage. The material is deposited along the internal slope of settling tank mound. The dragline is used up to the level of +70 m. The material that the dragline has transferred at the internal slope of the settling tank as additionally excavated by a trench dredge and loads it onto trucks for material transport to the designated location.

In the second stage, the material from the settling tanks is excavated in width of 25 m. The construction of auxiliary mounds (combs) within the settling tank was planned made up of sturdy materials 25 m in length and 5 m wide at the crown. These auxiliary mounds are constructed in parallel between the second and third stage. The distance between the auxiliary mounds is 40 m.

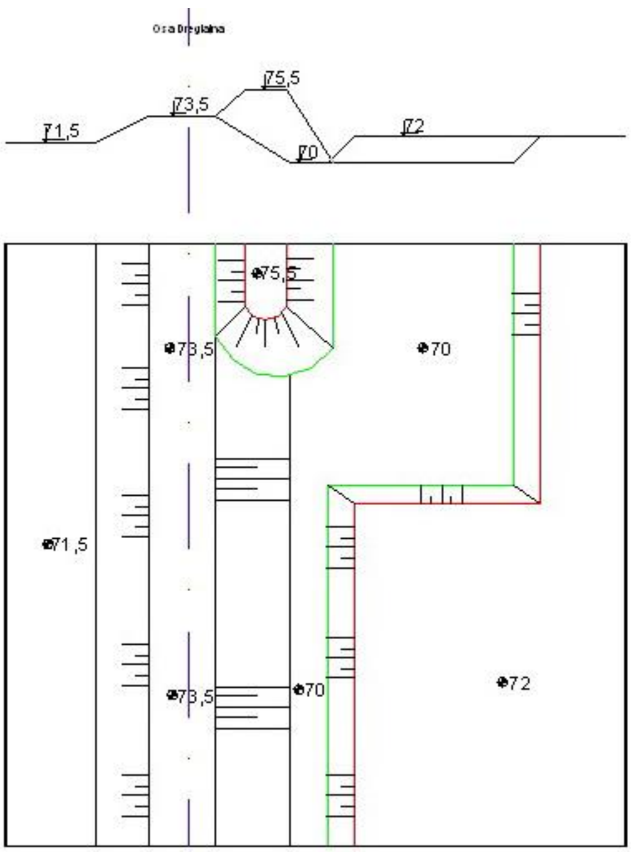

Figure 3 *Second stage of material separation and transport.*

Third stage: *Dragline excavation from the auxiliary mound and spoon dredge operation with loading onto trucks*

In the third stage, the dragline goes to the auxiliary mounds and in full width (25 m) transfers the material from settling tank to the internal slope of the same into the reach of the spoon dredge. The spoon dredge does additional excavations and loading onto trucks using standard technology.

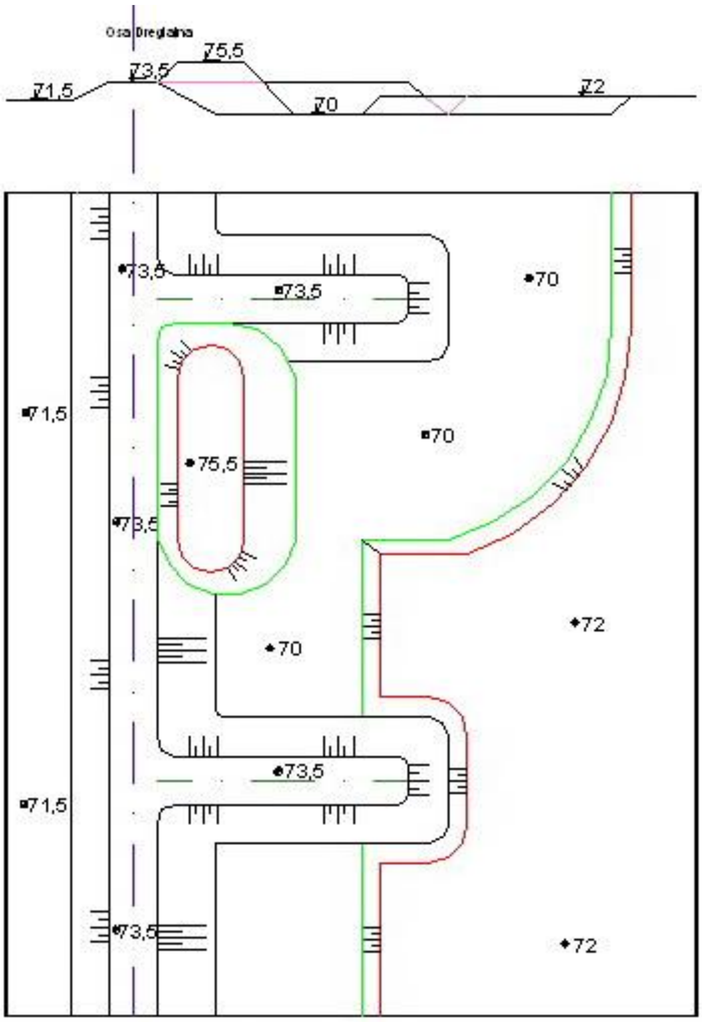

Figure 4 *Third stage of separation and material transport*

CONCLUSION

The process of cleaning the settling tank is technologically demanding. The mine possesses the mechanization, apart from the dragline dredge, that can regularly maintain the settling tank's functionality. This displays only one of the settling tank discharge methods that needs to constantly be in operation. By discharging this area of the big settling tank, the operation of the facilities for additional coal processing is enabled. Exclusion of materials from the big settling tank and discharging the same gives positive financial effects, as well as greater basin utilization. The significance of the big settling tank is great in environmental term because clean water pours out of it.

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Slavko Torbica, Veljko Lapčević**

FRAGMENTING OF CYLINDRICAL ROCK SPECIMENS UNDER EXPLOSIVE LOAD – COMPARISON BETWEEN MODEL AND LABORATORY RESULTS

Abstract

Achieving optimal fragmentation of the blasted rock is common task for mining engineers. During the years many models for fragmentation prediction were developed and are mostly based on empirical relations. While there are theoretical expansions of the empirical models, one could note the lack of the purely theoretical models based on the rock breakage theory. Herein, validation of such model is presented. Model considers only the fragmenting of the cylindrical monolith rock specimens and compares the results with the laboratory tests that were carried out in same manner. Model provides results through the definition of the fragment shape and size and then geometrical fragmenting of the whole specimen. Comparison of the model and laboratory results shows high level of agreement between sieving curves and also confirm the right constitution of the rock blasting theory that this model is based on.

Keywords: *rock fragmentation, rock breakage by explosives, fragmentation model, laboratory tests, blasting*

1 INTRODUCTION

Achieving optimal fragmentation of blasted rock is one of the main tasks in mining production. This means that after blasting muck pile will contain minimum possible amount of non-blasted blocks and fines. If these criteria are not satisfied, there is need for secondary fragmenting of large blocks which is time consuming. On the other hand, if there is large amount of fines it could be impossible to load these fractions and it leads to greater loss of blasted material. Kuz-Ram empirical model was the first to provide estimate of the fragment size distribution. This model was proposed by Kuznetsov [1] and is based on the Rosin-Rammler distribution. Later on, model was expanded by Cunningham [2, 3] and up to these days it is the one most used and modified model for fragment size estimation. Many researchers were considering differrent statistical distributions and their application inside the Kuz-Ram model. As for the example, Ouchterlony [4] proposed the Swebrec function that fits the size distribution curve in the domain of fines. Sanchidrián et al. [5, 6] provides summary of many functions used for the description of the fragment size distribution. Djordjevic [7] in his work also considered the fines range that is obviously weak point in existing methodologies. It is very clear that Kuz-Ram model is the central point when it comes to the fragment size estimation. Also, it is noticeable that there is lack of the fundamental models that are based on the rock breakage theories. Although there are considerable researches in the area of the crack propaga-

* *University of Belgrade, Faculty of Mining and Geology, veljko.lapcevic@rgf.bg.ac.rs*

tion and development under the explosive load [8-10], there is no exact method to predict fragment size distribution. One part of the model that is presented and validated herein is based on the actual rock breakage theory and tends to provide exact fragment size and spatial distribution. Model considers idealistic situation where monolith rock specimens are blasted and analyzed for fragment size distribution. Validation of the model is done through the comparison between model and laboratory results.

2 ROCK BLASTING THEORY

According to a rock blasting theory [11] it is possible to calculate the radii of zones having a different density of radial cracks around the blasthole. This theory explains the fracturing mechanism of rock under explosive load. Here, the main part of that theory is presented, since it is the basis of this paper. Detonation of an explosive charge in rock results in dynamic loading of the walls of the blasthole and generation of a pressure wave that transmits energy through the surrounding medium.

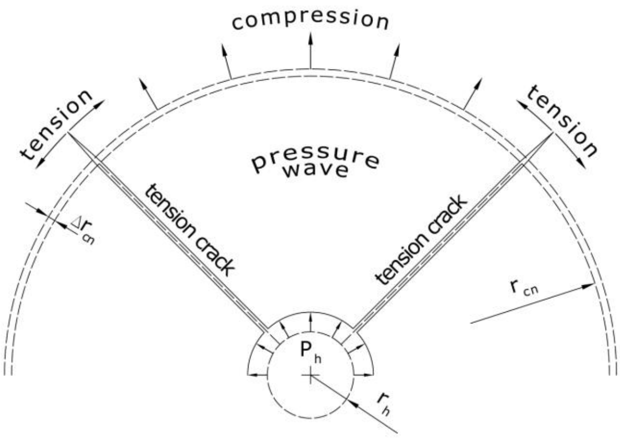

Figure 1 *Radial crack formation mechanism*

The pressure wave extends from borehole walls circularly around the borehole (Figure 1). At the distance r_{cn} from the borehole compressive stress of the rock in the radial direction is:

$$\sigma_{rc} = P_h \frac{r_h}{r_{cn}} \tag{1}$$

Where:

σ_{rc} - radial compressive stress

P_h - borehole pressure

r_h - borehole radius

r_{cn} - crack zone radius

On the other hand:

$$\sigma_{rc} = M \cdot e_r \tag{2}$$

Where:

$$M = E \cdot \frac{(1-\nu)}{(1+\nu)(1-2\nu)} \tag{3}$$

$$k = \frac{(1-\nu)}{(1+\nu)(1-2\nu)} \tag{4}$$

$$\sigma_{rc} = E \cdot k \cdot e_r \tag{5}$$

M - pressure wave modulus [12]

e_r - radial strain

E -Young`s modulus of rock
v - Poisson's ratio
Or:

$$e_r = \frac{\sigma_{rc}}{E \cdot k} \quad (6)$$

Therefore:

$$e_r = \frac{P_h \cdot r_h}{E \cdot k \cdot r_{cn}} \quad (7)$$

At the distance r_{cn}, before the pressure wave gets to it, the perimeter of the closed circular ring zone of rock mass is:

$$O_r = 2\pi r_{cn} \quad (8)$$

When the pressure wave reaches the closed circular ring zone of rock mass, it is moved to a new position with a radius $(r_{cn} + \Delta r_{cn})$, and with the perimeter:

$$O_{(r_{cn}+\Delta r_{cn})} = 2\pi(r_{cn} + \Delta r_{cn}) \quad (9)$$

Therefore:

$$O_{(r_{cn}+\Delta r_{cn})} = 2\pi(r_{cn} + e_r \cdot r_{cn}) \quad (10)$$

Once the closed circular ring zone of rock mass is subjected to tension with a lateral strain:

$$e_l = \frac{O_{(r_{cn}+\Delta r_{cn})} - O_{r_{cn}}}{O_{r_{cn}}} = e_r \quad (11)$$

For the formation of the radial tension cracks, tensile strain is required:

$$e_t = \frac{\sigma_t}{E} \quad (12)$$

Where:
e_t - tensile strain
σ_t - tensile strength
E - Young`s modulus

In addition, the number (n) of radial tensile cracks formed at a distance r_{cn} will be:

$$n = \frac{e_l}{e_t} \quad (13)$$

Therefore, it is:

$$n = \frac{P_h r_h}{k \sigma_t r_{cn}} \quad (14)$$

Therefore:

$$r_{cn} = \frac{P_h r_h}{k \sigma_t n} \quad (15)$$

Therefore, for the borehole radius r_h = 0.051m and the borehole pressure P_h= 1.6 GPa in limestone with tensile strength σ_t = 7MPa, Poisson's ratio v = 0.3 (Table 1) will be:

Table 1 *Zone radius with crack density*

n	2	4	8	16	32
r_{cn} (m)	4.32	2.16	1.08	0.54	0.27

In Figure 2, the schematic illustration of tension crack length and density around blast hole is shown. Practical application of this theory was demonstrated for the estimation of blasted rock fragmentation[13] and for the blast damage zone extent estimation [14].

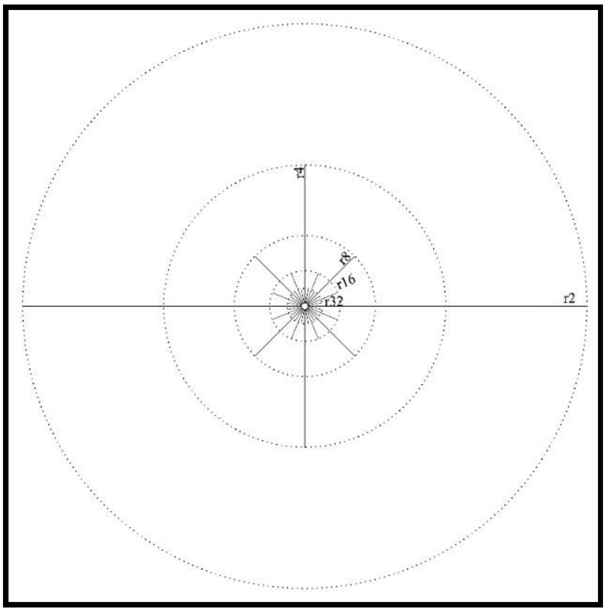

Figure 2 *Schematic illustration of tension crack length and density around blasthole*

3 FRAGMENTING OF THE CYLINDRICAL MONOLITH ROCK SPECIMENS UNDER THE EXPLOSIVE LOAD

Rock blasting theory [11] explains radial tension crack formation under explosive load. This theory was used for the constitution of the model for estimating blasted rock fragmentation [13]. Model describes application of the rock blasting theory for the fragment size estimation after the blasting in the rock mass. It should be mentioned that there is no theoretical explanation of complete rock fragmenting mechanism which may be used for fragment size estimation. Due to the lack of theoretical explanations of rock fragmenting, main assumption is that rock fragments have quasi-isometric dimensions while size distribution is fractal like. In this manner it is possible to estimate exact fragments size and their spatial distribution. Herein, fragmentation model is applied on the more idealistic situation where one cylindrical monolith rock specimen is blasted using one axially placed charge hole. In comparison with the blasting in the rock mass there are no pre-existing joints in the blasted medium. As it is well known pre-existing joints in the rock mass are limiting the propagation of the blast wave through the blasted medium. Therefore, radial tension cracks that are formed under the explosive load won't be limited in their length is blasted medium is monolith rock. Also, specimen has the "free surface" all around it, so it is assumed that tension cracks length will be equal in all directions. This makes it possible to compare model results with results of laboratory tests that were performed in same manner. One axial borehole in cylindrical rock specimen is filled with explosive and initiated. As a result of explosive detonation, radial tension cracks are formed. According to the rock blasting theory it is possible to calculate radii of zones with different density of radial cracks using Equation 15. Figure 3 and Figure 4 illustrates rock specimen with formed radial tension cracks and different cracking zones.

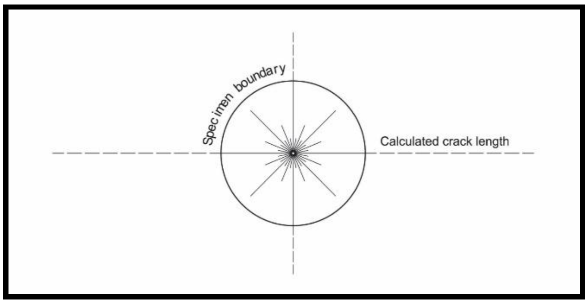

Figure 3 *Relation between specimen and calculated crack dimensions*

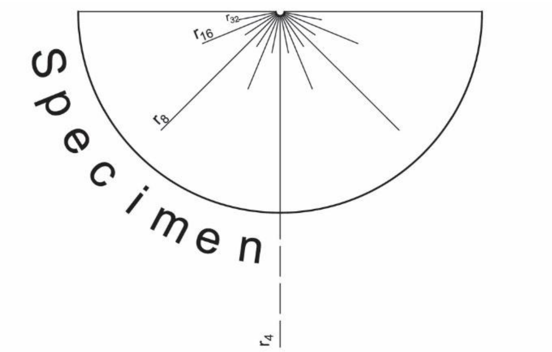

Figure 4 *Detail of rock specimen with tension cracks*

According to the previous assumptions, that fragments are quasi-isometric in their size with fractal like distribution, it is expected that fragmenting of the rock specimen is close to the idealistic fragmenting illustrated in Figure 5 and Figure 6.

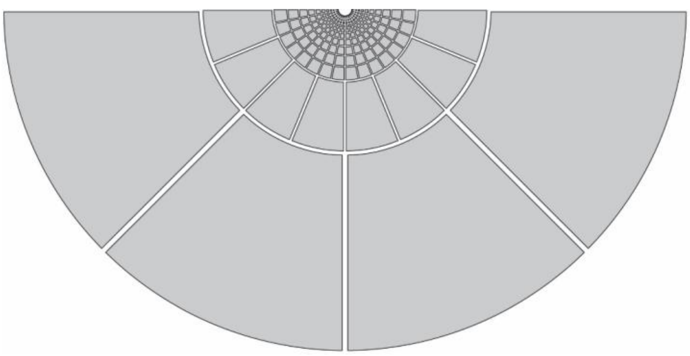

Figure 5 *Fragmenting of rock specimen*

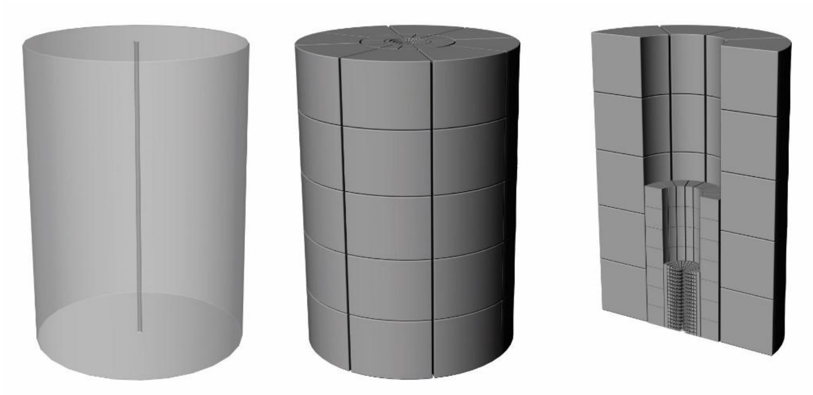

Figure 6 *3D illustration of specimen fragmenting*

As it can be seen fragment dimensions are determined by crack density and radii of each cracking zone. Fragment length depends on the length difference between two adjacent cracking zones, while fragment width is determined directly by crack density in each cracking zone. Height of each fragment is equal to the maximum dimension between length and width. Fragment dimensions are illustrated in Figure 7.

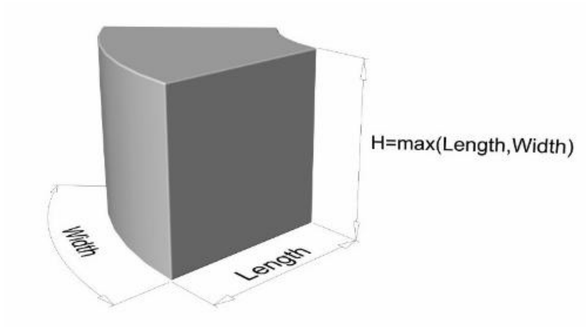

Figure 7 *Fragment dimensions*

4 COMPARISON BETWEEN MODEL AND LABORATORY RESULTS

Fragmentation model described in Section 3 was compared with lab-scale data [15]. Lab tests were conducted in the blast chamber in Leoben, Austria. Cylindrical rock specimens (Swedish Bårarp granite) were blasted using PETN explosive put into 5 mm diameter charge hole drilled axially through the sample; VOD was measured as control parameter. Figure 8 illustrates sample that is prepared for blasting. Tensile strength of rock is reported to be 13 MPa while no data regarding Poisson's ratio was available and therefore $\nu = 0.25$ is adopted for calculation. Table 2 presents sample data that was used in lab-scale tests. Results obtained from lab-scale tests are presented in Figure 9.

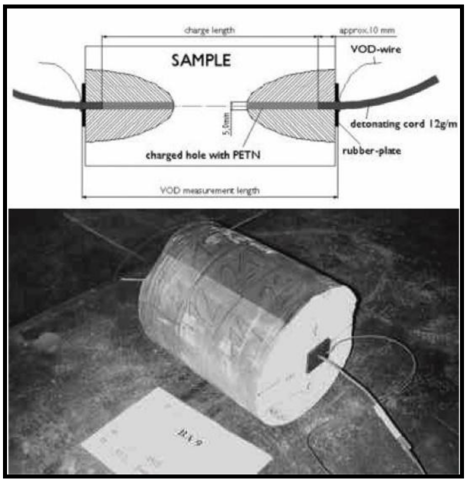

Figure 8 *Sample prepared for blasting [15]*

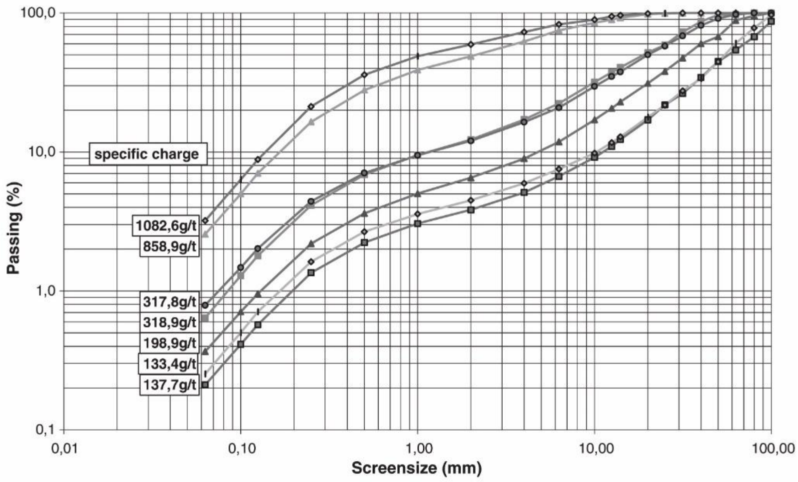

Figure 9 *Particle size distribution from lab-scale tests [15]*

Table 2 *Data for lab-scale tests [15]*

Sample	Diam. (mm)	Height (mm)	Mass (kg)	Charge (g)	Charge density (g/cm^3)	VOD (m/s)	Spec. charge (g/ton)
BA9	289	333	58.10	7.75	1.11	5743	133.4
BA2_2	192	310	24.04	7.64	1.18	5699	317.8
BA1_1	243	355	43.38	8.63	1.17	5879	198.9
BA10_2	103	218	4.84	5.24	1.19	5339	1082.6
BA10_1	103	217	4.82	4.14	0.94	5135	858.9
BA1_2	192	393	30.46	9.72	1.16	5842	319.0
BA2_1	290	367	61.21	8.43	1.07	5459	137.7

Calculation procedure for theoretical model is performed for each of the samples. Pressure inside the charge holes is calculated using measured VODs and reported charge densities. According to the Chapman-Jouguet detonation theory [16,17] pressure on the blast hole walls for explosives with density above 1 g/cm^3 can be calculated as:

$$P_d = \frac{\rho_e \cdot D^2}{8} \tag{16}$$

Where:

ρ_e – density of explosive (g/cm3)

D - detonation velocity of explosive (km/s)

For explosives with density below 1 g/cm^3 pressure on the blast hole walls is calculated by:

$$P_d = \frac{\rho_e \cdot D^2}{4.5} \tag{17}$$

Using calculated pressure and strength parameters of rock radii of each cracking zone is calculated at first. Then, fragments are drawn inside the sample boundaries and size of fragments in each of the zones is measured. Complete sample is divided into the fragments. Figure 10 a) presents reconstructed specimen after blasting [18] while b) presents fragmented rock specimen according to the presented model. Volume of the fragments is determined using CAD software and it is assumed that rock density is uniformly distributed in sample, so fragment volume and mass are proportional.

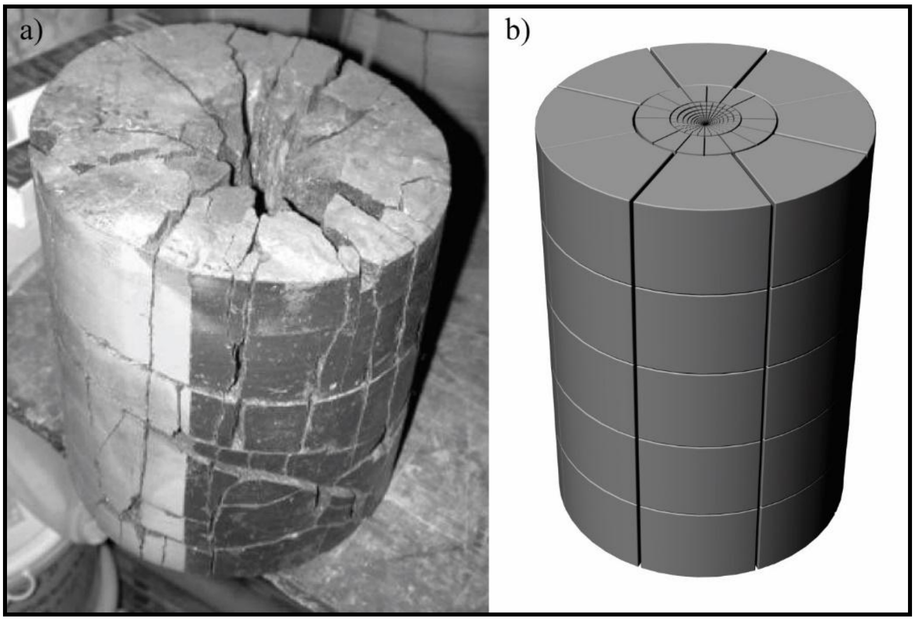

Figure 10 *a) reconstructed specimen after blasting [18] b) fragmented rock specimen according to the presented model*

Fragment sizes are classified in the same size classes as results from lab-scale tests and correlation coefficient is calculated for each sample. For each sample cumulative passing curves are plotted and shown in Figure 11 - Figure 18.

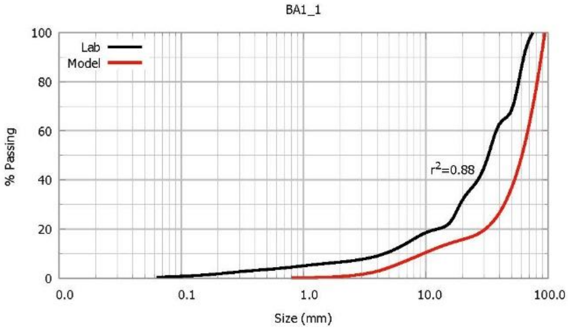

Figure 11 *Results comparison for sample BA1_1*

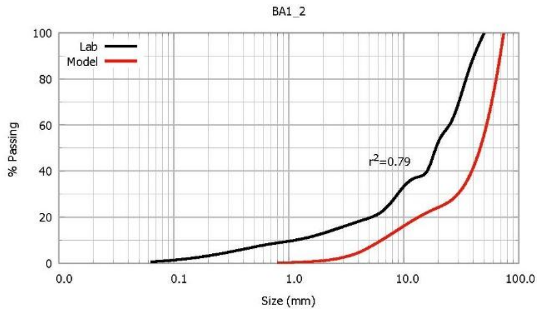

Figure 12 *Results comparison for sample BA1_2*

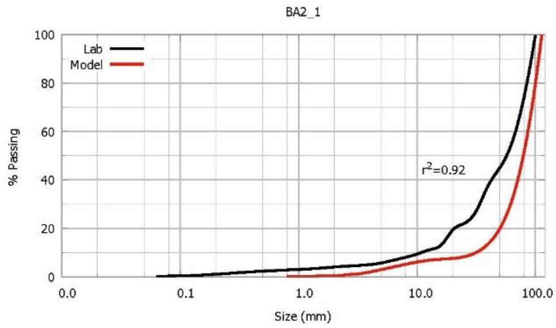

Figure 13 *Results comparison for sample BA2_1*

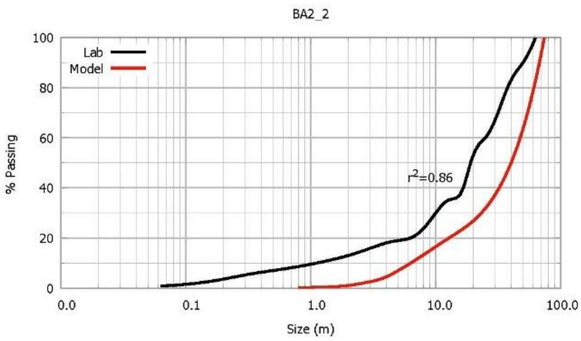

Figure 14 *Results comparison for sample BA2_2*

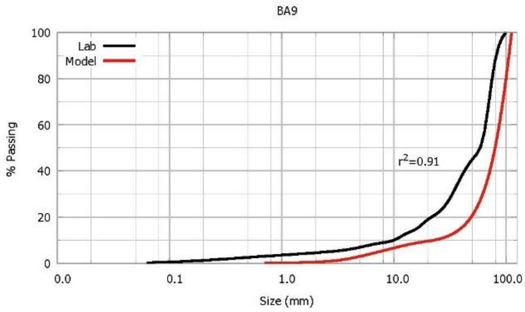

Figure 15 *Results comparison for sample BA9*

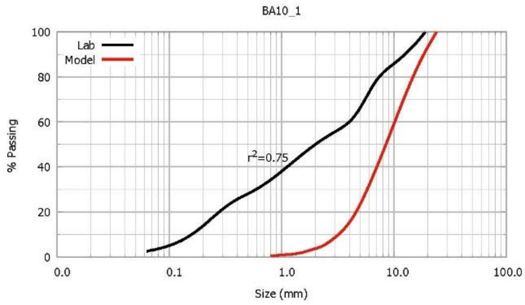

Figure 16 *Results comparison for sample BA10_1*

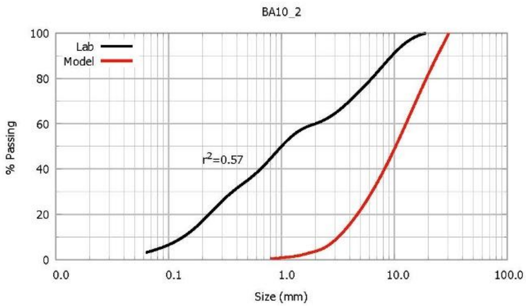

Figure 17 *Results comparison for sample BA10_2*

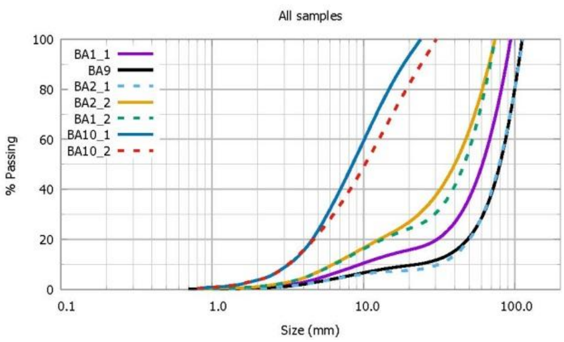

Figure 18 *Model results comparison for all samples*

As it can be seen there is strong agreement between model and laboratory test results for samples BA1_1. BA1_2, BA2_1, BA2_2, BA_9 where correlation coefficient spans between 0.79 and 0.92. In all of those cases model results show systematically larger fragment sizes in comparison to the laboratory ones. There are few possible reasons for this. As first, Poisson's ratio of rock is not known and value of 0.25 was adopted for calculation. Changing values from 0.2 and 0.3, for same borehole pressure, gives different sizes of cracking zones radii. This means that fragment size distribution may be slightly different depending on the real Poisson's ratio value. On the other side, model was calculated using measured VOD values in order to maintain comparability of results and to eliminate any inconsistency related to the input data. Using the theoretical VOD values that can be calculated according to the charge densities would give slightly higher VOD values, which implies higher borehole pressure and the longer radii of the cracking zones. In this case it is to be expected that fragment sizes are smaller, which would make laboratory and model results much closer.

On the other side, results for the samples BA10_1 and BA10_2 show lower agreement between laboratory and model results. In those cases correlation coefficients are 0.75 and 0.57, respectively. In both cases model results show higher fragment sizes. Reasons for this discrepancy of results may be different. In case of the sample BA10_1 charge density is below $1 g/cm^3$ which imply application of the different equation for the pressure calculation (see eqn. 17). Therefore, there is possibility that discrepancy of the results in this case comes from the incorrect pressure estimate. Other possibility may encompass the strength heterogeneity between samples. Due to the small specimen dimensions slight change of the input data would result in appreciable changes of the fragment sizes. In the case of the specimen BA10_2 there is even greater discrepancy of the results. However, it should be noted that in this case there is the highest charge density in comparison with all other samples while the VOD is significantly lower than it is expected. This means that borehole pressure may be incorrectly estimated which resulted in higher fragment sizes.

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Mladjan Maksimović[*], Snežana Urošević[**], Dragiša Stanujkić[*], Darjan Karabašević[*]

SELECTION A DEVELOPMENT STRATEGY OF MINING TOURISM BASED ON THE GREY RELATIONAL ANALYSIS

Abstract

Today, more attention is paid to development of mining tourism, which constitutes one of the important components of development of not only tourism, but also of integrated and sustainable development of the region as a whole. To select a strategy for positioning of mining tourism in the tourist map of the region, it is necessary to consider the economic importance of development the specific regions. Therefore, the aim of this paper is to propose the right approach to the choice of development strategy of mining tourism on Stara planina (Old Mountain), a tourist destination located in eastern Serbia, on its border with Bulgaria.The proposed approach for selection of strategies is based on the use the grey relational analysis. The usability and efficiency of the proposed approach is considered on the conducted numerical example.

Keywords: *mining tourism; development tourism strategy; MCDM; GRA*

INTRODUCTION

Many former mining areas have lost their industrial function and turn now to the tourism for the purpose of regional revitalization and economic development of communities, [6]. There is no doubt that the abandoned mining areas may become significant objects of tourism and recreation, and among other things, have scientific and educational significance. By including of these objects in the tourism program, a compromise has to be found between the interests of tourists, miners and environmental protection programs, which in some cases will not be easy [31].

Observed as a potential for development the new forms of tourism, mining tourism can be harmonized with already developed conventional tourism. Mining tourism is such a form of tourism that includes every type of the tourist activity in the area of an abandoned or active mine. A largenumber of abandoned mine sites overgrown with weeds have their own values, but what they have in common is the fact that they surely will perish. Instead, they can be transformed into an environmentally healthy environment or organized as an industrial cultural heritage which local and foreign tourists often visited and stayed in them.

Development of mining tourism can be an effective tool to mitigate the impact of economic crisis in mining, because it is focused on the applied mining research as well as the function of museum exhibitions representing a technology of the past used for exploitation of mineral resources [39, 27, 2]. This implies that tourism demand is increasingly striving towards avoiding usual tourist destinations, and requires return to the traditional and typical values and au-

[*] *Faculty of Management Zaječar, John Naisbitt University Belgrade, Park Šuma "Kraljevica" bb, 19000 Zaječar, Serbia. E-mail: mladjan21@yahoo.com*
[**] *Technical Faculty in Bor, University of Belgrade, Vojske Jugoslavije 12, 19210 Bor, Serbia*

thenticity. Tourist demand seeks various kinds of adventures and a deeper knowledge of where new tourist products appear with new environmental, ecological and social [30, 15] ones. In contemporary modern tourism, the aim of every tourist destination is to create a unique identity, or a difference in relation to competition, which will provide the basis for further growth and development in a competitive market [11].

According to Streimikiene and Bilan [34], tourism in general can also be seen as a natural process of change, because a proper understanding of this process enables a dynamic development of tourism and mining tourism, which also allows the identification of the main factors affecting changes in development of tourism, as well as the selection of appropriate strategies that have an impact on identification and evaluation the strategic directions of development the mining tourism.

To select a strategy for the purpose of positioning mining tourism in the tourist map of the region, it is necessary to research the economic impact of development the specific regions as well as the potential heritage of these regions. Cole [5], describes the existing social, environmental and economic perspective of sustainability of this form of tourism. Such an approach to development of mining tourism represents the basis for development the quality tourist offer, as well as its placement on the tourism market.

Multiple - criteria decision - making (MCDM) is one of the fastest - growing fields of operations research. The MCDM has found its application for solving the most diverse and complex decision-making problems. Over time, many MCDM methods have been developed; therefore, a good overview and comparisons of some prominent MCDM methods are given by Mardani *et al*. [21 ,22, 23], Kahraman *et al*. [13], Turskis and Zavadskas [35].

The MCDM is successfully used for the solving complex problems in tourism industry, such as the MCDM application to the sustainable urban development of the Naples port area [3], a hybrid delfi-anp-vikor approach for financing the tourism industry [1], the application of the fuzzy benchmarking approach for the strategic planning of tourism destination [19], creating an MCDM hybrid model for improving the tourism policy implementation [18], developing sustainable tourism using the multicriteria analysis [24], and so on.

The manuscript is organized as follows: Section 1 presents the primary objectives for development the mining tourism; Section 2 shows the potential for development the mining tourism on Stara planina (Old Mountain); Section 3 presents the potential strategies for development the mining tourism on Stara Planina; Section 4 shows the grey relational analysis, whereas in Section 5, a numerical example is presented, only to be followed by the final conclusions.

THE PRIMARY OBJECTIVES FOR DEVELOPMENT THE MINING TOURISM

The primary objective of development the mining tourism is the opening of new and providing support to the existing business entities and accelerating development of entrepreneurship, job creation, redundant workers, increasing the tax base, reconstruction and further use the existing facilities, improving the infrastructure and creation an image of community as the future center for innovation and entrepreneurship [28].

Mining tourism is certainly a real advantage of revitalization of many smaller and larger mining spatial units. The development of tourism in these areas can stop the emigration of young people through creation the basic conditions for general, significantly higher comfort. In such circumstances, young people can find not only economic, but also social and cultural themes in order to continue to live in this environment. Therefore, in countries with a developed tourism offer, there is an increasing focus on development the mining tourism, which today constitutes one of the important components of not only development of tourism, but also of integrated and sustainable development of the region as a whole [20].

The tourism and mining sectors are interacting, but the mining sector is often considered to be destroying the environment on which tourism is based, although that relationship does not necessarily have to be negative. The mining sector can actively participate in tourism by providing an access to industrial, mining attractions. Mining tourism can cause significant structural changes in the economy and society [4].

Revitalization and transformation of old mines into cultural tourism and museum centers have a tremendous support in Europe, where former mining areas are converted into the new tourist destinations [2]. A good example is the mine of "Idrija" in Slovenia, a former mercury mine, while an even better example is the former coal mine in Labin in Istra, Croatia, which has been transformed into a cultural center, with the-already-created project of decorating the "underground city" on 60,000 square meters of space in corridors and abandoned mining areas.

Another good example is the Wieliczka Salt Mine, Kraków, Poland the most unusual salt mine in the world that on the surface has a nothing special view; however, 200 meters below the surface, the place hides a startling secret. The salt mine has become a unique art gallery, with a cathedral and underground lake. Over a million visitors a year come to see this amazing salt mine. For the security reasons, less than one percent of the mine is open for visitors, but its almost 4 kilometers long corridors are more than enough for tourists to spend an hour or two on a tour of these amazing rooms.

The copper mine in Bor, Serbia is yet another good example. There, at the eleventh underground mining horizon, a large space was built, with the capacity of up to 60 people, where tourists are enabled to descend to a depth of 700 meters every 15 days, to take photos or talk with the miners, whereas on the surface, visitors have an opportunity to take a look at the oldest open pit mine, with the old open pit about 500 meters deep; a safari trail on the old flotation tailing dump with a length of 17 km and width of 10 meters is also under construction.

POTENTIAL OF STARA PLANINA MOUNTAIN POTENTIAL FOR DEVELOPMENT THE MINING TOURISM

According to Nikolić *et al.* [27], the tourist destinations of Stara planina, located in eastern Serbia, on the border with Bulgaria, are an attractive area with great prospects for entering the world tourism scene.

On the slopes of Stara planina, in the east of the village of Kalna, in the belt of beech forests, there is the mine field "Janja". As a part of this ore field, at the end of the nineteen-fifties in the last century, uranium was exploited from the three mines: "Mezdreja", "Gabrovnica" and "Srneći Do", that stopped working in 1966. In the mining field, along with the separation in Kalna, there were more than 800 miners working in three shifts. The miners were mostly locals from the area, who, during active operation, were materially well-situated and the mine field is hinted to be the rebirth of eastern Serbia.

The all mines were with underground exploitation. Within the mine "Mezdreja", there was a plant for ore processing, with the capacity of 60 tons/day, while in the mine "Gabrovnica", there was a plant for ore processing, with the capacity of 200 tons/day. Nikić *et al*. [25], state that the processing plants were the hydrometallurgical ore type, so there was no possibility of air pollution. Tailings from the mines were delayed near the mining pits. At the end of the nineteen-sixties, mining stopped and all of the three mines were closed. At that point, a certain conservation of the mines and the processing plants was executed. The mine field "Janja" covers an area of about

30 km^2, at an altitude ranging from 500 to 950 m above sea level, and now is covered with forest vegetation. In the wider area of the ore field “Janja”, next to the settlement of Kalna, there are large settlements of the rural type, namely Gabrovnica, Inova, Vrtovac, Janja, Balta Berilovac and Mezdreja, which, in the process of migration, have been largely left with no residents or with a small number of them, mostly elderly households [25].

This area then started to languish and the villages began to die. All of this has been attributed to industrialization because, at that time, the rural population went to the cities in masses, where they earned for their living better. However, a desire to preserve country life and open up the new opportunities for safe existence has caused many to return to the village and found their own businesses there. Thus, many households began to engage in tourism

This mining site has its own characteristics, although there are mining facilities falling into disrepair which can be converted into ethno- and eco-centers, an ecologically healthy environment which European tourists will be happy to visit and stay in, whereas the second part can be organized as an industrial cultural heritage that has to be put under the state protection.

THE STRATEGIES FOR DEVELOPMENT THE MINING TOURISM ON STARA PLANINA MOUNTAIN

Three strategies are proposed for development the mining tourism on Stara Planina Mountain, namely:

- Strategy A_1 - human resource development in the field of mining tourism;
- Strategy A_2 - the use of aggressive marketing, advertising and market approach of mining tourism;
- Strategy A_3 - creation of needed tourism and accompanying infrastructure that will help development of mining tourism.

On the basis of relevant literature and factual situation within the field, the ranking of the mentioned strategies will be carried out in order to select the best among them, with the aim of better positioning mining tourism on the tourist map of Serbia.

Strategy A1 - Human Resource Development in the Field of Mining Tourism

Parallel to development of tourism, the personnel also developed, they who gave their lives, as well as their work orientation, economic and existential interests tied to tourism as a social and economic activity [33]. Tourism is an economic activity largely depending on the human factor because in tourism, people and personnel are highly-integrated with consumers (tourists) [7].

The quality of tourism services largely depends on the quality of engagement, goodwill and training of human resources at all levels. That immediately implies questions about how the importance of human factor, as the bearer and executor of tourist activity in an area, is perceived. A positive attitude towards tourism, above all, shows the degree of social and cultural development the population of an area, which is the basic prerequisite for development of tourism.

Tourism is a labor-intensive industry, which means that, for that economic activity to be performed, it takes a lot of human potential. Modern technical aids mainly contribute to the accelerating certain work processes, technologies change work, but, as a rule, are less effective in reducing the number of employees, particularly in the hospitality industry. The human resources that have already worked or prepared to work in mining tourism are forced to constantly innovate and improve their professional knowledge due to the increasing competition of knowledge and ideas. Personnel must be trained specifically for the reason of being able to establish a direct contact with guests and provide them with comprehensive information [38].

Strategy A2 - The Use of Aggressive Marketing, Advertising and Market Approach of Mining Tourism

This strategy is aimed to establish an efficient marketing system in order for destinations to penetrate into the target markets and market niches, as well as to constantly identify the new sources of competitive advantage and monitor the capacity of loyalty or recommendation to visit the destination.

Tourist propaganda is one of the tourist policy instruments for achieving the certain goals, which means that actions of tourist propaganda must well thought out. Tourist propaganda has an influence on development tendency and a desire for tourist trips [41]. Tourist propaganda should be viewed integrally with the other instruments of tourist or business policies, such as the pricing policy, the policy of development of tourism and so forth [36], and it can be a stimulus for both the public and private sectors aiming at increasing the volume of visitors and rational use of energy and other resources [29].

Strategy A3 - Creation of Needed Tourism and Accompanying Infrastructure that will Help Development of Mining Tourism

The existing facilities caught after leaving the mines, are mostly in poor condition and need to be rebuilt in order to continue work on their development. It is necessary that such an infrastructure that would consist of extra accommodation, sports facilities, new access roads, training camps and places for excursions and active entertainment should be developed [27].

In accordance with the available resources of local community, the desired effect can be produced with minimum investment and minimum tourism demand, such as the opening of the mining museum and reuse the industrial tracks for tourist purposes, can be met.

THE GREY RELATIONAL ANALYSIS

The grey relational analysis (GRA) was developed by Deng [8], as a part of the grey system theory. Since then, it has been widely used to solve many uncertainty problems involving discrete data and incomplete information, optimization problems and multiple criteria decision-making (MCDM) problems, such as the application of the GRA method on the performance evaluation of airlines [9], the application of the GRA for evaluation the financial performance [17], the application of the GRA method for t corrosion failure of oil tubes [10], the supplier selection based on the use of the GRA method [40, 12], the application of the GRA method in the high-speed machining of aluminum alloy [16], etc.

The procedure of the GRA method can be shown as follows [32]: Let $A = \{A_1, A_2, ..., A_m\}$ be a discrete set of alternatives, $C = \{C_1, C_2, ..., C_n\}$ be a set of criteria and $w = \{w_1, w_2, ..., w_n\}$ the weighting vector, where $w_j = [0,1]$ and $\sum_{j=1}^{n} w_j = 1$. Then, the determination of the most acceptable alternative applying the GRA can be described through the following steps:

Step 1. Determine the ideal solution. The ideal solution (the reference point) is a solution that maximizes the benefit criteria and minimizes the cost criteria, and can be determined by using the following formula:

$$A^* = \{r_1^*, r_2^*, ..., r_n^*\} = \\ = \{(\max_i r_{ij} \mid j \in \Omega_{\max}), \\ (\min_i r_{ij} \mid j \in \Omega_{\min})\}, \qquad (1)$$

where A^* is the ideal solution, r_j^* is the j-th coordinate of the ideal solution, r_{ij} is the normalized rating of the i-th alternative to the j-th criterion, and $\Omega_{\max}$ and $\Omega_{\min}$ are sets of benefit and cost criteria, respectively.

Step 2. Calculate the grey relational coefficient of each alternative from the ideal solution using the following formula:

$$\xi_{ij} = \frac{\min\limits_i \min\limits_j | r_j^* - r_{ij} | + \varsigma \max\limits_i \max\limits_j | r_j^* - r_{ij} |}{| r_j^* - r_{ij} | + \varsigma \max\limits_i \max\limits_j | r_j^* - r_{ij} |}, \qquad (2)$$

where ξ_{ij} is the grey relational coefficient of the i-th alternative to the j-th criterion, ς is the distinguish coefficient, and $\varsigma \in [0,1]$.

Step 3. Calculating the grey relational grade of each alternative from the ideal solution using the following formula:

$$G_i = \frac{1}{n} \sum_{j=1}^{n} w_j \xi_{ij} \;, \qquad (3)$$

where G_i is the grey relational grade of the i-th alternative, and w_j is the weight of the j-th criterion

Step 4. Rank the considered alternatives and select the best one(s) in accordance with G_i. The alternatives with a higher G_i are better ranked, and the alternative with the highest G_i is the most appropriate / preferable one.

A NUMERICAL EXAMPLE

With the goal to briefly demonstrate the proposed approach and show the efficiency and usability of the GRA method, a numerical example will be conducted in this section. Suppose that a decision maker should evaluate the three strategies A_1, A_2 and A_3 in relation to the five evaluation criteria: C_1 – The implementation of strategy feasibility; C_2 – The speed of implementation; C_3 - Compliance with the strategy of development the tourism and local economic development; C_4 – An economic profit and C_5 –satisfaction of service users.

At the beginning of evaluation, the decision maker evaluates an alternative in relation to the selected criteria. The ratings of the considered alternatives are shown in Table 1.

Table 1 *Ratings of the considered strategies*

	C_1	C_2	C_3	C_4	C_5
A_1	4	4	3	4	3
A_2	4	3	3	4	2
A_3	2	2	3	4	3

After that, the ideal point is determined using Eq. (1). The Ideal point A^* and distances from alternatives to the Ideal Point are accounted for in Table 2.

Table 2 *Ideal Point and distances between the alternatives and ideal point*

	C_1	C_2	C_3	C_4	C_5
A^*	4	4	3	4	3
A_1	0	0	0	0	0
A_2	0	1	0	0	1
A_3	2	2	0	0	0

In the next step, using Eq. (2), the grey relational coefficient of each alternative in relation to the ideal point is calculated, as it is shown in Table 3. In this case, ς is set to 0.5.

Table 3 *The grey relational coefficient of each alternative to the Ideal Point*

	C_1	C_2	C_3	C_4	C_5
A_1	1.00	1.00	1.00	1.00	1.00
A_2	1.00	0.50	1.00	1.00	0.50
A_3	0.33	0.33	1.00	1.00	1.00

Finally, using Eq. (3), the grey relational grade of each alternative is calculated. The grey relational grades, and the rank order of alternatives, are shown in Table 4. In this case, the same weight w_j=0.2 was assigned to all the criteria.

Table 4 *The grey relational grades and the rank order*

	G_i	*Rank*
A_1	0.20	1
A_2	0.16	2
A_3	0.15	3

The data from Table 4 indicate that the strategy labeled as A_1 has the best ranked alternative. The strategy labeled as A_1 is based on development of human resources in mining tourism.

CONCLUSIONS

At the destination of Stara Planina Mountain, there is an untapped potential of the three former uranium mines, which, once appropriately adapted, may become an interesting tourist attraction for visitors, as well as a new type of tourism - i.e. mining tourism, which is already represented and developed in Europe and the world. This paper has proposed the three mining tourism development strategies, where, on the basis of relevant literature and factual situation on the field, the ranking and selection of development strategy is performed using the GRA method. The purpose of selection a strategy for development the mining tourism is to provide assistance to the employees in understanding the key elements necessary for efficient business management as well as the possibility of familiarizing themselves with the essential components used in development a business plan, as well as the assessment of profitability of business ventures in mining tourism. This study shows the three potential development strategies of mining tourism on Stara planina (Old Mountain) and also proposes one approach to make a selection of the best of them in accordance with the defined evaluation criteria. In accordance with the conducted numerical example, the development strategy designated as A1, which is based on development the human resources in mining tourism, ranks as the best one in terms of evaluation criteria. The study also shows that the abandoned mining areas on Stara planina could be the tourism development possibilities, just like similar examples in the world. The manuscript proposes one effective method that could be used for selection of development strategies.

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