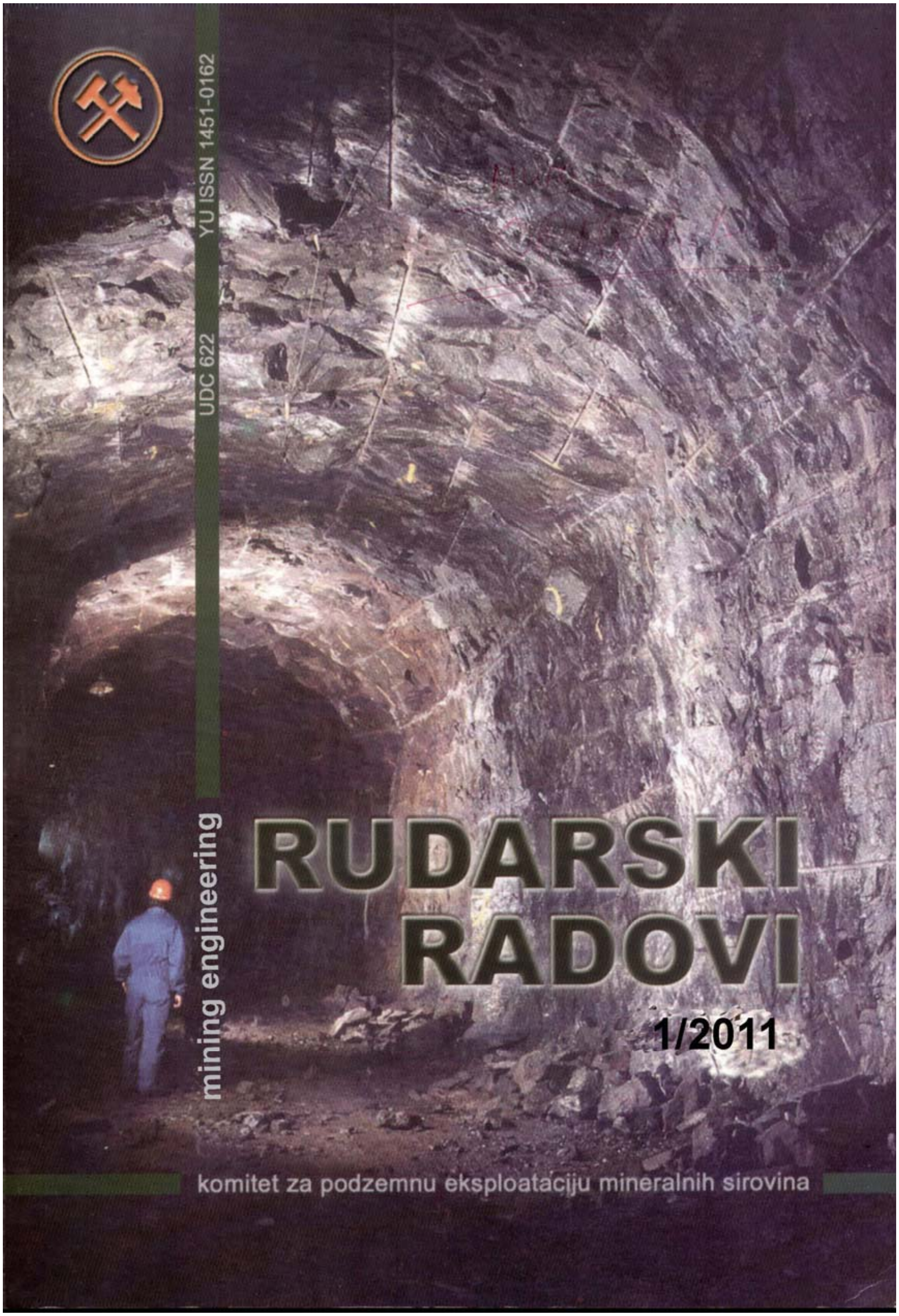

YU ISSN 1451-0162
UDC 622
mining engineering
RUDARSKI
RADOVI
1/2011
komitet za podzemnu eksploataciju mineralnih sirovina

INSTITUT ZA RUDARSTVO I METALURGIJU BOR
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA

RUDARSKI RADOVI je časopis baziran na bogatoj tradiciji stručnog i naučnog rada u oblasti rudarstva, podzemne i površinske eksploatacije, pripreme mineralnih sirovina, geologije, mineralogije, petrologije, geomehanike i povezanih srodnih oblasti. Izlazi dva puta godišnje od 2001. godine, a od 2011. godine četiri puta godišnje.

Glavni i odgovorni urednik

Dr Milenko Ljubojev naučni savetnik, dopisni član IAS
Institut za rudarstvo i metalurgiju Bor
E-mail: milenko.ljubojev@irmbor.co.rs
Tel. 030/454-110

Zamenik glavnog i odgovornog urednika

Dr Mirko Ivković, viši naučni saradnik
Komitet za podzemnu eksploataciju mineralnih sirovina Resavica
E-mail: mirko.ivkovic@jppeu.rs
Tel. 035/625-566

Urednik

Vesna Marjanović, dipl.inž.

Prevodilac

Nevenka Vukašinović, prof.

Tehnički urednik

Suzana Cvetković, teh.

Priprema za štampu

Ljiljana Mesarec, teh.

Štamparija: Grafomedtrade Bor

Tiraž: 100 primeraka

Internet adresa

www.mininginstitutebor.com

Izdavanje časopisa finansijski podržavaju

Ministarstvo za prosvetu i nauku Republike Srbije
Institut za rudarstvo i metalurgiju Bor
Komitet za podzemnu eksploataciju mineralnih sirovina Resavica

ISSN 1451-0162

Indeksiranje časopisa u SCIndeksu i u ISI.

Izdavač

Institut za rudarstvo i metalurgiju Bor
19210 Bor, Zeleni bulevar 35
E-mail: milenko.ljubojev@irmbor.co.rs
Tel. 030/454-254

VODEĆI ČASOPIS NACIONALNOG ZNAČAJA M51 ZA 2010.

MINING AND METALLURGY INSTITUTE BOR
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS

MINING ENGINEERING is a journal based on the rich tradition of expert and scientific work from the field of mining, underground and open-pit mining, mineral processing, geology, mineralogy, petrology, geomechanics, as well as related fields of science. Since 2001, published twice a year, and since 2011 four times year.

Editor-in-chief

Ph.D. Milenko Ljubojev, Principal Reasearch Fellow, Associate member of ESC
Mining and Metallurgy Institute Bor
E-mail: milenko.ljubojev@irmbor.co.rs
Phone: +38130/454-110

Co-Editor

Ph.D. Mirko Ivković, Senior Research Associate
Committee of Underground Exploitation of the Mineral Deposits Resavica
E-mail: mirko.ivkovic@jppeu.rs
Phone: +38135/625-566

Editor

Vesna Marjanović, B.Eng.

English Translation

Nevenka Vukašinović

Technical Editor

Suzana Cvetković

Preprinting

Ljiljana Mesarec

Printed in: Grafomedtrade Bor

Circulation: 100 copies

Web site

www.mininginstitutebor.com

MINING ENGINEERING is financially supported by

The Ministry of Education and Science of the Republic Serbia
Mining and Metallurgy Institute Bor
Committee of Underground Exploitation of the Mineral Deposits Resavica

ISSN 1451-0162

Journal indexing in SCIndex and ISI.

Published by

Mining and Metallurgy Institute Bor
19210 Bor, Zeleni bulevar 35
E-mail: milenko.ljubojev@irmbor.co.rs
Phone: +38130/454-254

LEADING NATIONAL JOURNAL CATEGORIZATION M51 FOR 2010.

SADRŽAJ
CONTENS

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:622.03:550.8.01:622.343(045)=861

Mile Bugarin, Vladan Marinković*, Vojka Gardić*, Gordana Slavković**

ISTORIJAT ISTRAŽIVANJA I GEOLOŠKA GRAĐA BORSKIH LEŽIŠTA BAKRA**

Izvod

Pronalaženje zlata na Deli Jovanu (1888. god., Glogovica) uticalo je da se pristupi obimnijim geološkim i rudarskim istraživanjima, što je dovelo do otkrića borskog rudišta 1902. god. Prvi istražni radovi u Boru počinju 1897. god. 1902. god. je pronađeno rudno telo „Čoka Dulkan", a 1904. god se počinje sa proizvodnjom. U periodu posle prvog svetskog rata aktivirana su istraživanja u Boru, a naročito se intenziviraju od 1927. do 1930. god. Godine 1948., počinje se sa sistematskim istraživanjem kako u Rudniku bakra Bor, tako i na području timočkog andezitskog masiva.

Stene od kojih je izgrađeno borsko ležište su: sveži i hidrotermalno izmenjeni andeziti, andezitski i dacitski piroklastiti, peliti sa tufovima i tufitima, konglomerati i peščari i kvarcni aluvijalni nanosi.

Ključne reči: *bakar, list Bor, rudno telo, hidrotermalno izmenjena zona, granični sadržaj.*

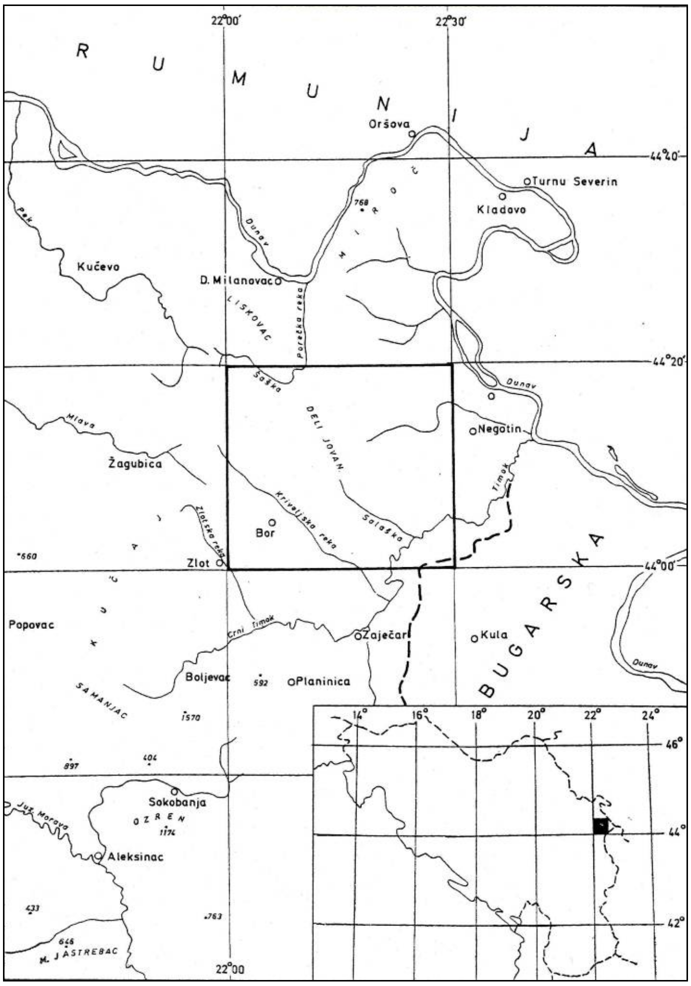
1. UVOD

Rudnik bakra Bor se nalazi u severoistočnom delu Srbije na oko 11 km. zapadno od Bugarske i na oko 70 km. južno od Rumunije (slika 1).

Područje rudnog polja Bor zauzima centralni deo timočkog eruptivnog masiva.

To je područje ograničeno sa zapada Crnim Vrhom (1.027 m), sa severa Malim Kršem (920m) i Velikim Kršem (1.148 m), a sa južne strane je teren mnogo niži te nema izrazitih visova.

* *Institut za rudarsvo i metalurgiju Bor*
** *Ovaj rad je proistekao iz Projekta broj 37001 koji je finansiran sredstvima Ministarstva za prosvetu i nauku Republike Srbije.*



Sl. 1. *Pregledna geografska karta – položaj lista Bor.*

2. ISTORIJAT ISTRAŽIVANJA

Tereni koje zahvata list Bor predstavljaju jedno od najinteresantnijih područja u istočnoj Srbiji kako zbog rudnog bogatstva, tako i zbog vrlo heterogenog geološkog sastava. Još od najstarijih vremena rudno bogatstvo ovih terena je bilo predmet rudarske aktivnosti, o čemu svedoče tragovi Rimskog rudarstva utvrđeni u okolini Bora (Bor, Brestovac, Pjatra Roš, Krivelj). Doseljavanjem Slovena na Balkansko Poluostrvo nastavljena je rudarska aktivnost koja je kasnije znatno pojačana dolaskom Sasa (Crnajka, Šaška reka). Moderno rudarstvo započelo je dolaskom A. Hedera (1835. god.) u Srbiju na poziv kneza Miloša, sa ciljem „da se rudna blaga učine poleznim za srpsko otačestvo".

Krajem 1848. god. počeli su istražni radovi sa ciljem pronalaženja gvožđa u Majdanpeku, Rudnoj Glavi i Crnajki.

Nalazak zlata na Deli Jovanu (1888. god., Glogovica) uticao je da se pristupi obimnijim geološkim i rudarskim istraživanjima, što je dovelo do otkrića borskog rudišta 1902. god.

Prvi istražni radovi u Boru počinju 1897. god.

Godine 1902. je pronađeno rudno telo „Čoka Dulkan", a 1904. god se počinje sa proizvodnjom.

U periodu do prvog svetskog rata Francusko društvo borskih radnika po preporuci inžinjera Šisteka izvodi istražne radove u Metovnici, Velikom i Malom Krivelju i tom prilikom je pronađeno malo ali bogato rudno telo u Kiridžijskom potoku kod Malog Krivelja. Ovo rudno telo francuzi su otkopali do 1912. god., a imalo je oko 6.000 tona bogate rude.

U ovom periodu, od strane francuskih geologa Blanšana i Šlumbegera primenjuju se i geofizičke metode, pored rudarskih istražnih radova.

U periodu posle prvog svetskog rata aktivirana su istraživanja u Boru, a naročito se intenziviraju od 1927. do 1930. god. U ovom periodu detaljno se istražuju rudna tela Tilva Mike, a delimično i Tilva Roš. Rudno telo Tilva Roš je privlačilo manju pažnju za istraživanje zbog niskog sadržaja bakra. Ovim istraživanjima su rudne rezerve bile znatno povećane, dok je proizvodnja blister bakra porasla na 40.000 tona.

U periodu do 1940. god., francuzi počinju sa detaljnim istraživanjem borske hidrotermalno izmenjene zone na nivou V horizonta i otkrivaju rudna tela „Tilva Ronton", „Kamenjar" i rudno telo „E".

U ovom periodu istraživanja su intenzivirana i u okolini Bora, tako da praktično nije bilo rudnog izdanka na kome se nije bar nešto radilo. Istražni radovi su bili u vidu kratkih podkopa i plitkog bušenja. Većina tih radova su ostali bez rezultata, bilo da je koncesionaru nestalo para za izvođenje skupih rudarskih radova ili je uspeo da koncesiju proda drugom vlasniku. Postignuti rezultati su čuvani kao tajna. Ako se ovome doda i činjenica da su radovi obavljani bez stručnog nadzora, dolazi se do zaključka da uspeha u okolini Bora nije ni bilo.

Prekretnica u istraživanu nastaje po završetku drugog svetskog rata, kada je čitavo rudno bogatstvo tadašnje FNRJ, a kasnije i SFRJ postalo vlasništvo naroda. Sa sistematskim istraživanjem se nije odpočelo odmah posle rata, obzirom da su sve snage bile angažovane na osposobljavanju rudnika i proizvodnju.

Godine 1948., počinje se sa sistematskim istraživanjem kako u Rudniku bakra Bor, tako i na području timočkog andezitskog masiva.

Istraživanja su se sastijala u izradi istražnih hodnika i dubinskog bušenja. Do 1962. god. profili istražnih hodnika su bili 4 m^2, dok se posle prelazi na 6 m^2, koji su locirani kao istražno pripremni.

Istraživanja u ovom periodu bila su ograničena na istraživanje poznatih rudnih tela, dok su istraživanja u cilju pronalaženja novih rudnih tela bila zapostavljena.

Godine 1961. se započelo sa istraživanjima u kriveljskoj hidrotermalno izmenjenoj zoni. Ova istraživanja su trajala sve do 1974. god. Doistraživanja ove zone su vršena u više navrata od 1982. do 1992. god. i od 1997. do 1998. god. Na osnovu rezultata ovih istraživanja otkriveno je ležište bakra „Veliki Krivelj" i izrađeno je nekoliko elaborata o rezervama rude bakra na ovom prostoru. Prvi elaborat je bio izrađen 1978. god., ovim elaboratom su overene rezerve od 64.351.000 tona rude bakra sa graničnim sadržajem od 0,30% bakra po toni rude. Drugi elaborat je bio izrađen 1992. god., ovim elaboratom su overene rezerve od 164.572.853 tona rude bakra sa graničnim sadržajem od 0,20% bakra po toni rude.

Treći elaborat je bio izrađen 2005. god., ovim elaboratom su overene rezerve od 465.150.392 tona rude bakra sa graničnim sadržajem od 0,20% bakra po toni rude. Poslednnji elaborat je bio izrađen 2010. god., ovim elaboratom su overene rezerve od 621.921.288 tona rude bakra sa graničnim sadržajem od 0,15% bakra po toni rude .

Na području Kraku Bugaresku koje se nalazi severno od Bora geološka istraživanja su vršena u više navrata, od 1965. do 1967. god. Detaljna geološka istraživanja su vršena u periodu od 1975. do 1978. god. Dok su doistraživanja vršena od 1981. do 1995. god. Ovim istraživanjima je otkriveno ležište bakra „Cerovo“. Prvim proračunom rezervi rude bakra je overeno 238.141.000 tona rude. Dok je elaboratom iz 2007. god. overeno 319.377.890 tona rude sa graničnim sadržajem od 0,20% bakra po toni rude.

Takođe na prostoru Kraku Bugaresku, nešto severnije u odnosu na ležište „Cerovo“ su vršena geološka istraživanja koja su započeta 1977. god. a završena 1986. god. Detaljna istraživanja ovog prostora su vršena u periodu od 1987. god i trajala do 1991. god., a doistraživanja su započeta 1999. god i trajala su do 2001. god. Ovim istraživanjima je potvrđeno postojanje još jednog ležišta bakra na ovom prostoru, sa rezervama rude bakra od 70.092.715 tona i graničnim sadržajem od 0,20% bakra po toni rude. Ovo ležište je nazvano „Kraku Bugaresku – Cementacija“.

U periodu od 1976. do 1999. god. vršena su geološka istraživanja u području Borske reke, pri čemu je otkriveno rudno telo „Borska reka“ sa ustanovljenim rezervama rude bakra od 15.496.154 tona.

3. GEOLOŠKA GRAĐA ŠIRE OKOLINE

Timočki andezitski masiv je nastavak subvulkanskog andezitskog masiva koji iz Rumunije prelazi u Srbiju. Pravac pružanja mu je sever – jug i proteže se u dužini od 70 km i širini od 15 km.

Magmatsko tektonska evolucija krajem krede i početkom tercijara je bila sledeća: narušavanjem izostatičke ravnoteže u srednjogorskoj geosinklinali od Majdanpeka preko Bora, pa sve do Bugarske na Crnom moru došlo je do vulkanske erupcije u donjem senonu.

Submarinski vulkanizam se odvijao u tri faze:

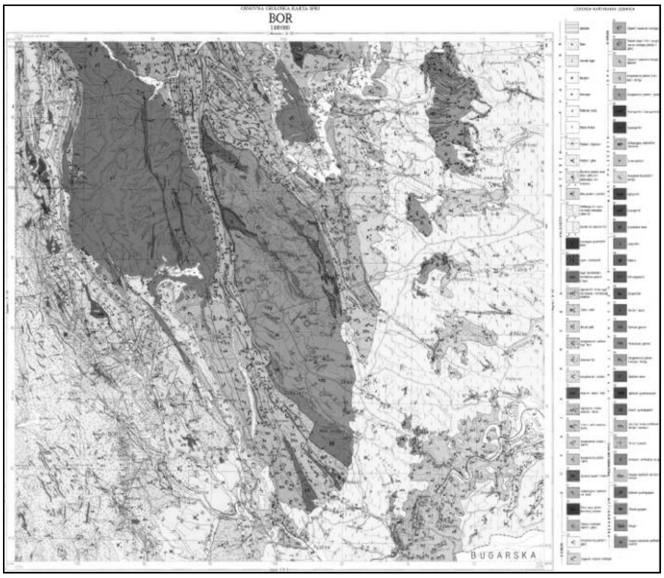
- U prvoj fazi je došlo do izlivanja, a zatim i do očvršćavanja hornblenda, hornblenda biotitskih andezita i dacita. Za ove vulkanite je karakteristična porfirska struktura sa krupnim fenokristalima plagioklasa, hornblende i biotita.
- U drugoj fazi su stvarani piroksenski, amfibolske i piroksensko – amfibolske andezite i piroklastite, sa sitnim fenomfistalima plagioklasa, hornblende i piroksena.
- Pirokseni treće faze nisu nađeni pa se smatra da su erodovani.

Laramijska orogeneza mobilisala je ognjišta dajući niz plutonita i njihovih hipoabisalnih ekvivalenata, odnosno monconita, diorita, kvarcdiorita i gabra.

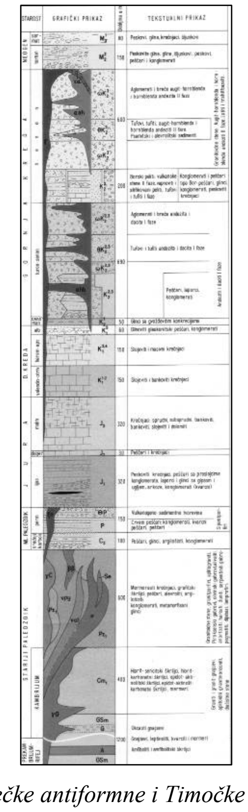
Sve pomenute magmatite su pratili hidrotermalni rastvori koji su vršili hidrotermalne izmene ranije stvorenih stena.

Hidrotermalno izmenjene zone javljaju se u vidu izduženih zona, dimenzija od više kilometara, a pravac pružanja im se poklapa sa laramijskim dislokacijama.

Sve napred opisane stenske jedinice su prikazane na osnovnoj geološkoj karti (List Bor) (slike 2. i 3.).



Sl. 2. *Osnovna geološka karta 1 : 100.000 (list Bor), umanjen prikaz.*



Sl. 3. *Geološki stub Porečke antiformne i Timočke sinformne strukture 1 : 15.000, umanjen prikaz*

4. GEOLOŠKA GRAĐA RUDNOG POLJA BOR

Borska hidrotermalno izmenjena zona se nalazi u povlati moćne serije konglomerata i peščara.

Neposrednu granicu predstavlja borski rased koji se proteže u dužini od 40 km. Uz borski rased leži i kriveljska hidrotermalno izmenjena zona. Obe zone imaju pravac severozapad – jugoistok i pad ka jugozapadu pod uglom od 70°.

Većina istraživača smatra de je Borski rased reversni rased duž koga je zapadni – povlatni blok kretan naviše. Na sveže otkrivenim delovima rasedne površi se to moglo i utvrditi.

Stene od kojih je izgrađeno borsko ležište su:

- sveži i hidrotermalno izmenjeni andeziti;
- andezitski i dacitski piroklastiti;
- peliti sa tufovima i tufitima;
- konglomerati i peščari;
- kvarcni aluvijalni nanosi.

Hidrotermalni rastvori su pratili laramijsku orogenezu, metasomatski su izmenili hornblenda – biotitske andezite i njihove piroklastične ekvivalente. Hidrotermalne promene nisu svuda iste, niti je njihov intenzitet isti. Do sada su konstatovane sledeće izmene: hloritizacija, kaolinizacija, alunitizacija, karbonatizacija, sericitizacija, silifikacija, piritizacija i epidotizacija.

Sve pomenute promene su hidrotermalnog porekla i vezane su za tektonske zone. Te zone predstavljale su put najmanjeg otpora nadolazećim hidrotermalnim rastvorima, čiji su se sadržaji i temperature lateralno menjali.

5. ZAKLJUČAK

Istorijat geoloških istraživanja a u vezi sa njima i rudarenja u Boru i okolini se proteže na period od preko 1000 godina, još iz doba Rimljana. A u novijoj istoriji na period od 1835. god. pa sve do danas. Pri čemu ovaj proces nikako nije završen, već se naprotiv sa porastom cene bakra na svetskom tržištu intenzivira sa ciljem doistraživanja već postojećih i pronalaženja novih ležišta, kako bi se postojeće rezerve bakra uvećale a samim tim i obezbedila budućnost rudarenja na ovim prostorima.

LITERATURA

[1] Miličić M., Grujčić B., i ostali: Elaborat o rudnim rezervama rudnika bakra Bor stanje, Institut za bakar Bor, Srbija, 1977.

[2] Grujčić B., Nikolić R., Mišković V. i ostali: Elaborat o rudnim rezervama rudnika bakra Bor stanje, Institut za bakar Bor, Srbija, 1982.

[3] Tumač za list Bor L34 – 141, 1976 god., Srbija.

[4] Brajović M. i ostali: Elaborat o rudnim rezervama rudnog tela „Borska reka" ležišta bakra Bor, Institut za bakar Bor, Srbija, 1999.

[5] Maksimović M., Pačkovski G. i ostali: Elaborat o rezervama ležišta bakra „Veliki Krivelj", Institut za rudarstvo i metalurgiju Bor, Srbija, 2010.

[6] Maksimović M, Nikolić K. i ostali: Elaborat o rezervama ležišta bakra „Cerovo", Institut za rudarstvo i metalurgiju Bor, Srbija, 2006.

[7] Maksimović M, Nikolić K. i ostali: Elaborat o rezervama ležišta bakra „Kraku Bugaresku-Cementacija", Institut za rudarstvo i metalurgiju Bor, Srbija, 2006.

MINING AND METALLURGY INSTITUTE BOR YU ISSN: 1451-0162
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS UDK: 622

UDK: 622.03:550.8.01:622.343(045)=20

Mile Bugarin, Vladan Marinković*, Vojka Gardić*, Gordana Slavković**

HISTORY OF INVESTIGATION AND GEOLOGICAL STRUCTURE OF THE BOR COPPER DEPOSITS**

Abstract

The finding of gold on Deli Jovan (1888, Glogovica) has affected to access with the extensive geological and mining investigations, what resulted into discovery the Bor ore deposit in 1902. The first prospecting works in Bor started in1897. In 1902, the ore body Čoka Dulkan was found and the production started in 1904. In the period after the First World War the investigations were activated in Bor, and they were especially intensified from the 1927 to 1930. In 1948, the systematic investigation has started both in the Copper Mine Bor and in the area of Timok andesite massif.

The Bor deposit is built of the following rocks: fresh and hydrothermally altered andesites, andesite and dacite pyroclastites, pelytes with tuffs and tuffites, conglomerates and sandstones and quartz alluvial deposits.

***Key words:** copper, the Bor leaf, ore body, hydrothermal altered zone, cut-off grade*

1. INTRODUCTION

Copper Mine Bor is located in the northeastern part of Serbia about 11 km on the west of Bulgaria and about 70 km on the south of Romania (Figure 1).

The area of the ore field Bor occupies the central part of the Timok volcanic massif. This area is bounded on the west with Crni Vrh (1,027 m), from the north with Mali Krš (920 m) and Veliki Krš (1,148m) and on the south side, the ground is much lower and there no pronounced peaks.

* *Mining and Metallurgy Institute, Bor*

** *This paper is produced from the project no. 37001 which is funded by means of the Ministry of Education and Science of the Republic of Serbia*

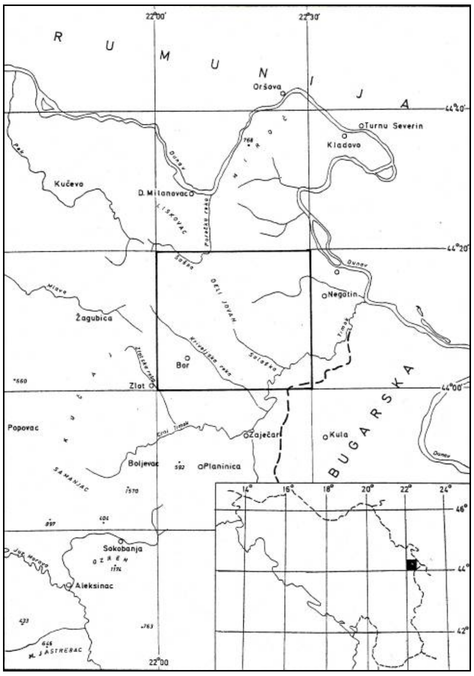


Figure 1. *Geographic map – position of the Bor leaf*

2. HISTORY OF INVESTIGATIONS

The fields affected by the Bor leaf represent one of the most interesting areas in the Eastern Serbia both for the mineral resources and very heterogeneous geological structure. Since the ancient times, the mineral resources of these fields was the subject of mining activity as evidenced by the traces of Roman mining identified in the vicinity of Bor (Bor, Brestovac, Piatra Roche, Krivelj). By settlement of the Slavs on the Balcan Peninsula, the mining activity continued that was later significantly increased with the arrival of the Sasa (Crnajka, Šaška River). Modern mining began with the arrival of A. Header (1835) into Serbia at the invitation of Duke Miloš, with the aim "to make useful the mineral resources for the Serbian homeland". At the end of 1848, the prospecting works began aimed at finding the iron in Majdanpek, Rudna Glava and Crnajka.

The finding of gold on Deli Jovan (1888, Glogovica) has affected to access with the extensive geological and mining investigations, what resulted into discovery the Bor ore deposit in 1902.

The first prospecting works in Bor started in1897. In 1902, the ore body Čoka Dulkan was found and the production started in 1904.

In the period before the First World War, the French Society of the Bor workers as recommended by engineers Sistek carried out the prospecting works in Metovnica, Veliki and Mali Krivelj and at that occasion a small but rich ore body was found in the Kiridžijski Creek near

Mali Krivelj. This ore body, the French was excavated up to 1912 and it had about 6,000 tons of rich ore.

In this period, the geophysical methods were applied by the French geologists Blanchan and Schlumbeger in addition to the mining exploration works.

In the period after the First World War, the investigations were activated in Bor, and especially intensified from 1927 to 1930. In this period, the ore bodies of Tilva Mika were investigated in detail, and partly Tilva Roche. The ore body Tilva Rosh attracted less attention for investigation due to the low copper content. By these investigations, the ore reserves were significantly increased, while the production of blister copper increased to 40,000 tons.

In the period to 1940, the French began with a detailed investigation of the Bor hydrothermally altered zone at the level of V horizon and discovered the ore bodies Tilva Ronton, Kamenjar and the ore body "E".

During this period, the investigations were intensified in and around Bor, so there was no practically the ore outcrop on which at least something was worked. The exploration works were in the form of short adits and shallow drilling. Most of these works were left with no results, whether the concessionaire missed the money for realization the expensive mining works or he was able to sell the concession to another owner. The achieved results were kept as a secret. If the fact is added to this that the works were undertaken without professional supervision, it can be concluded that the success in the region of Bor did not exist.

A milestone in the investigation occurred at the end of the Second World War, when the entire mineral resources of the former FPRY and later SFRY became the property of the people. A systematic study did not started immediately after the war, since all forces were engaged in the reconstruction of mine and production.

In 1948, the systematic investigation started both in the Bor copper mine, and in the area of the Timok andesite massif.

Investigations consisted of drifting and deep drilling. Until 1962, the profiles of drifts were 4 m^2, and later they were 6 m^2, located as the exploratory – preparation.

Investigations in this period were limited on exploration the known ore bodies, while the investigations, aimed to finding the new ore bodies, were neglected.

In 1961, the investigations began in the Krivelj hydrothermal altered zones. Those investigations lasted until 1974. Additional investigations of this zone were carried out on several occasions from 1982 to 1992 and from 1997 to 1998. Based on the results of these investigations, the copper deposit Veliki Krivelj was discovered and several elaborated were made on the reserves of copper ore in this area. The I Elaborate was made in 1978 that has confirmed the reserves of 64,351,000 tons of copper ore with a cut-off grade of 0.30% copper per ton of ore. The II Elaborate was made in 1992 that has confirmed the reserves of 164,572,853 tons of copper ore with a cut-off grade of 0.20% copper per ton of ore. The III Elaborate was made in 2005 that has confirmed the reserves of 465,150,392 tons of copper ore with a cut-off grade of 0.20% copper per ton of ore. The last Elaborate was made in 2010 that has confirmed the reserves of 621,921,288 tons of copper ore with a cut-off grade of 0.15% copper per ton of ore.

In the area Kraku Bugaresku, located north of Bor, the geological investigations were carried out on several occasions, from 1965 to 1967. Detailed geological investigations were carried out in the period from 1975to 1978; while the additional investigations were carried out from 1981 to 1995. These investigations have discovered the copper deposit Cerovo. The first calculation of the copper ore reserves has verified 238,141,000 tons of ore. While the elaborate from 2007 has verified 319,377,890 tons of ore with a cut-off grade of 0.20% copper per ton of ore.

Also in the area Kraku Bugaresku, slightly further north than the Cerovo, the geological investigations were carried out that began in 1977 and finished in 1986. Detailed investigations of this area were carried out in the period from 1987 and lasted until 1991, and the additional investigations began in 1999 and lasted until 2001. These investigations have confirmed the existence of another copper deposit in this area, with reserves of copper ore from 70,092,715 tons and a cut-off grade of 0.20% copper per ton of ore. This deposit was called the Kraku Bugaresku - Cementation.

In the period from 1976 to 1999, the geological investigations were carried out in the area of the Bor River, where the ore body Bor River was discovered with the established reserves of copper ore of 15,496,154 tons.

3. GEOLOGICAL STRUCTURE OF THE WIDER ENVIRONMENT

The Timok andesite massif is a continuation sub-volcanic andesite massif that moves from Romania to Serbia. The direction of it is north - south and extends to length of 70 km and width of 15 km.

Magmatic and tectonic evolution of the late Cretaceous and early Tertiary period was the following:

Violation of isostatic equilibrium, in the central mountain syncline from Majdanpek over Bor up to Bulgaria at the Black Sea, resulted into a volcanic eruption in the lower Shannon. Submarine volcanism took place in three phases.

The first phase resulted into spilling and then the induration of hornblende, hornblende biotite andesite and dacite. These vulcanites are characterized by porphyritic structure with large phenocrysts of plagioclase, hornblende and biotite.

Pyroxene, amphibole and pyroxene - amphibole andesites and pyroclastics were created in the second phase with fine phenophystale plagioclase, hornblende and pyroxene.

Pyroxenes of the third phase were not found and it is believed that they were eroded.

Laramian orogeny mobilized the homes, giving a series of plutonic rocks and their hypabyssal equivalents or monzonites, diorite, and quartzdiorite and gabbro.

All mentioned magmatic rocks were accompanied the hydrothermal solutions that performed the hydrothermal changes of previously generated rocks.

Hydrothermally altered zones occur in the form of elongated zones, the dimensions of several kilometers and their direction coincides with the Laramian dislocations.

All of the above described rock units are shown in the basic geological map (the Bor leaf) (Figures 2 and 3).

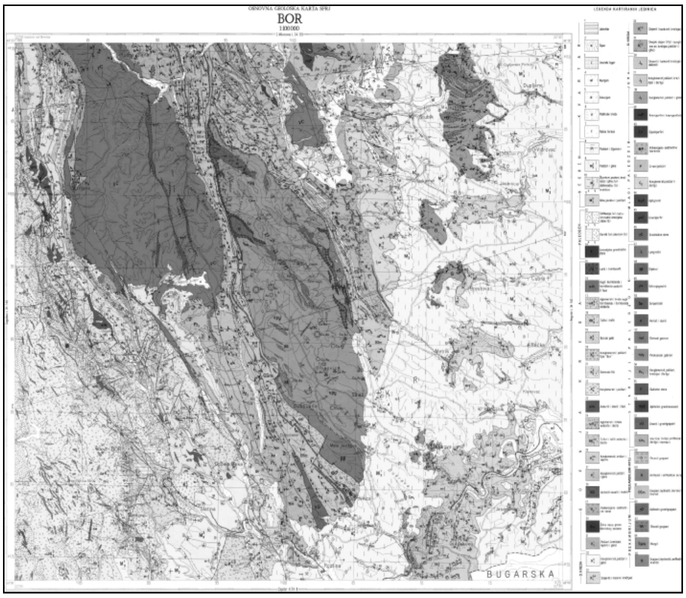


Figure 2. *Geographic map 1 : 100,000 (the Bor leaf)*

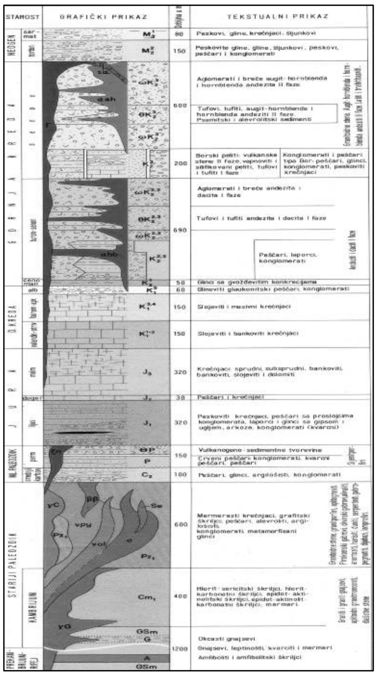


Figure 3. *Geological column of the Poreč antiform and Timok synform structure 1 : 15.000, thumbnail*

4. GEOLOGICAL STRUCTURE OF THE ORE FIELD BOR

The Bor hydrothermal altered zone is located in a fault block of powerful series of conglomerates and sandstones.

The immediate border is the Bor Fault that extends to a length of 40 km. The Krivelj hydrothermally altered zone lies next to the Bor Fault. Both zones have direction to the northwest – southeast and fall to the southwest at angle of 70°.

Most scientists believe the Bor Fault is a reverse fault along which the west - fault block moved upwards. It could be seen and determined on freshly uncovered parts of the fault surface.

The Bor deposit is built of the following rocks:

- fresh and hydrothermally altered andesites;
- andesite and dacite pyroclastics;
- pelites with tuffs and tuffites;
- conglomerates and sandstones;
- quartz alluvial deposits.

Hydrothermal solutions followed the Laramian orogeny and metasomaticaly altered the hornblende - biotite andesite and their pyroclastic equivalents. Hydrothermal alterations are not everywhere neither the same, nor their intensity is the same. Until now, the following alterations were stated: chloritization, kaolinization, alunitiyation, carbonation, sericitization, silicification, pyritization and epidotizacion.

All these changes are of hydrothermal origin and related to the tectonic zone. These zones represented the path of the lowest resistance of the coming hydrothermal solutions with their lateral changes of contents and temperatures.

5. CONCLUSION

The history of geological investigations and, in connection with them, the mining in Bor and its surroundings are stretches for over 1000 years, since the Roman times; and in the recent history for the periods from 1835 until nowadays. While this process is by no means complete, but rather with the increase of copper prices on the world market intensifies with the aim of additional investigations of already existing and finding the new deposits as the existing copper reserves would be increased and thus ensure the future of mining in this area.

REFERENCES

[1] Miličić M., Grujčić B., et al., 1977: Elaborate on the Ore Reserves of the Copper Mine Bor, condition 01. 01. 1977, Copper Institute Bor, Serbia (in Serbian)

[2] Grujčić B., Nikolić R., Mišković V., et al., 1982: Elaborate on the Ore Reserves of the Copper Mine Bor, condition 01. 01. 1982, Copper Institute Bor, Serbia (in Serbian)

[3] Legend for the Bor Leaf L34 – 141 1976, Serbia (in Serbian)

[4] Brajović M., et al., 1999: Elaborate on the Ore Reserves of the Ore Body Bor River the Copper Deposit Bor, Copper Institute Bor, Serbia (in Serbian)

[5] Maksimović M., Pačkovski G., et al., 2010: Elaborate on the Ore Reserves of the Copper Deposit Veliki Krivelj, Mining and Metallurgy Institute Bor, Serbia (in Serbian)

[6] Maksimović M, Nikolić K., et al., 2006: Elaborate on the Ore Reserves of the Copper Deposit Cerovo, Mining and Metallurgy Institute Bor, Serbia (in Serbian)

[7] Maksimović M, Nikolić K., et al., 2006: Elaborate on the Ore Reserves of the Copper Deposit Kraku Bugaresku-Cementacija, Mining and Metallurgy Institute Bor, Serbia (in Serbian)

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK: 711.455(045)=861

Gordana Milentijević, Blagoje Nedeljković**

HIDROGEOLOŠKE KARAKTERISTIKE TERMOMINERALNE VODE VUČA I NJEN UTICAJ NA ZDRAVLJE**

Izvod

U ovom radu prezentovani su rezultati novih saznanja o termomineralnoj vodi Vuča u smislu geneze, ocene potencijalnosti, fizičko-hemijskog sastava i lekovitosti. U tom pravcu izvedena su i neophodna terenska (geološka i hidrogeološka) i laboratorijska (mineraloško-petrografska analiza, fizičko-hemijska analiza i analiza radioaktivnosti) istraživanja. Rezultati istraživanja su prikazani u ovom radu. Termomineralna voda Vuča je ocenjene kao lekovita voda sa ograničenim mogućnostima upotrebe, što proizilazi iz činjenice da su pH vrednosti veoma visoke.

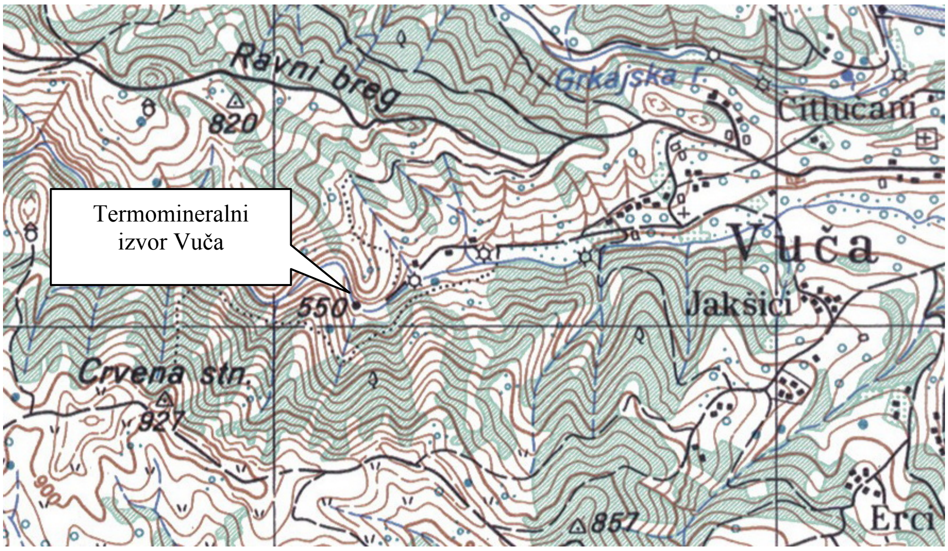
Ključne reči: *termomineralna voda, geneza, fizičko-hemijske karakteristike, balneološke karakteristike, lekovitost.*

1. UVOD

Analizirana termomineralna voda izvire u selu Vuča na južnim padinama planine Rogozna, na levoj dolinskoj strani Ibra, na 520 m nadmorske visine (slika 1.). Selo Vuča nalazi se na oko 17 km severozapadno od Kosovske Mitrovice. Pojava termomineralne vode Vuča je posledica brojnih ektonskih [3] i vulkanskih aktivnosti u prošlosti. Narodna medicina je prepoznala lekovitost termomineralne vode Vuča, tako da je ona, do sada, korišćena u tom pravcu. Cilj ovog rada je prezentovanje saznanja proistekla novim, geološkim, hidrogeološkim i laboratorijskim istraživanjima, u smislu ocene potencijalnosti, geneze, fizičko-hemijskih karakteristika i lekovitosti.

* *Univerzitet u Prištini, Fakultet tehničkih nauka Kosovska Mitrovica*

** *Ovaj rad je realizovan u okviru projekta „Istraživanje klimatskih promena na životnu sredinu: praćenje uticaja, adaptacija i ublažavanje“ (43007) koji finansira Ministarstvo za prosvetu i nauku Republike Srbije u okviru programa Integrisanih i interdisciplinarnih istraživanja za period 2011-2014. godine.*



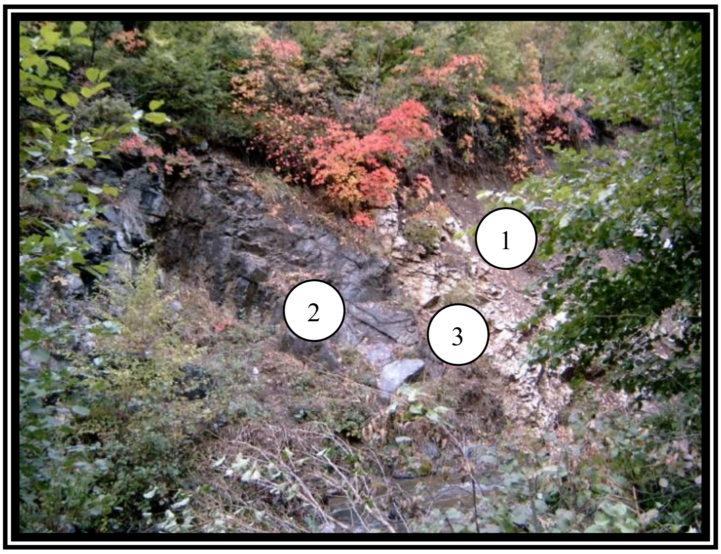
Slika 1. *Prostorni položaj mesta isticanja termomineralne vode Vuča*

2. GENEZA TERMOMINERALNE VODE VUČA

Geneza termomineralne vode dovodi se u vezu sa tektomagmatizmom Rogozne. Evolutivni razvoj planine karakterišu tektomagmatski procesi koji svojim produktima beleže pojedine faze razvića mezozojika i kenozojika. Magmatizam u trijasu i gornjoj kredi reprezentuju serpentinisani peridotiti, dijabazi sa efuzivnim ekvivalentima i gabro stene. U hronologiji tektomagmatizma najstarije i najrasprostranjenije magmatske stene su peridotiti. Po mineraloškom sastavu odgovaraju harcburgitskom tipu uz minimalno učešće ekvivalenata - lerzolita, dunita i dr. Karakteristika ovih stena je serpentinizacija-serpentinisani peridotiti i serpentiniti. Hidrotermalnim procesima tercijarnog magmatizma nastaju magnezitske žilice i žice. Procesi raspadanja su izraženi u serpentinitima (nontroniti, žični magnezit). Tercijarni magmatizam označava granitske intruzive praćene vulkanizmom u više sekvenci, kada se stvaraju vulkaniti (dacito-andenziti, kvarclatiti, piroksensko-amfibolitski andenziti, tufovi i konglomerati). U opisanim stenama, pojavljaje se termomineralna voda [7].

Rezervoar termomineralnih voda čini kompleks karbonatnih mezozojskih i paleozojskih stena. Najverovatnije se radi o trijaskim krečnjacima, s obzirom da su bili kratko izloženi eroziji tokom jure i donje krede. Vode u rezervoaru potiču iz perioda semiaridne klime (20.000 god.) i imaju temperaturu oko 120^0C [6].

Radi sticanja novih saznanja o genezi termomineralne vode Vuča urađene su mineraloško-petrografske analize iz proboja u zoni isticanja (slika 2.).



Slika 2. *Proboj za koji je vezano isticanje termomoneralne vode Vuča, a gde su uzeti uzorci za mineraloško-petrografsku analizu (foto G. Milentijević, 2008.g.)*

2.1. PRIKAZ MINERALOŠKO-PETROGRAFSKE ANALIZE UZORAKA IZ PROBOJA U ZONI ISTICANJA TERMOMINERALNE VODE VUČA

U sklopu istraživačkog projekta Hidrogeološka istraživanja mineralnih i termomineralnih voda severnog dela Kosova i Metohije, koji je finansiralo Ministarstvo za zaštitu životne sredine i prostornog planiranja, uzeti su uzorci stena iz proboja u zoni isticanja termomineralne vode i urađene su mineraloško-petrografske analize. Mineraloško-petrografske analize su urađene u laboratoriji za mineraloško-petrografska ispitivanja na Rudarsko-geološkom fakultetu u Beogradu.

Analizirani su uzorci iz ''krovine proboja'' (2), iz ''podinskog dela proboja'' (1) i iz ''centralnog dela proboja'' (3) [8].

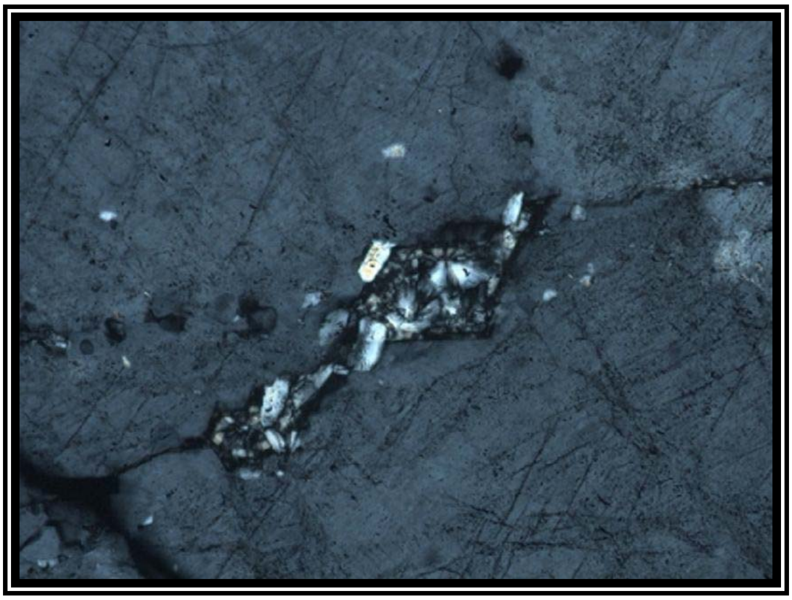
Analize dva uzoraka iz ''krovine proboja'' prema odlikama sklopa i mineralnog sastava ukazuju da su ispitivane stene: **kvarcit** sa kalcitom i epidotom i **serpentinit** nastao metamorfozom harcburgita.

Analize dva uzoraka iz podinskog dela proboja prema odlikama sklopa i mineralnog sastava ukazuju da su ispitivane stene **tonalit** i **hidrotermalno promenjeni tonalit.**

Analiza uzorka iz centralog dela proboja prema odlikama sklopa i mineralnog sastava ukazuje da je to **kvarcit**.

Makroskopski izgled stene: kvarcit je stena mlečnobele boje, granoblastične strukture i homogenog sastava. Po svojim makroskopskim karakteristikama odgovara uzorku br. 1. Oštrih je prelomnih ivica otporna na dejstvo hlorovodonične kiseline, para staklo. Stenska masa je intezivno ispucala, a pukotine imaju više pravaca pružanja. Prisutne su i limonitske skrame po pojedinim površinama i duž prslina.

Mikroskopski izgled stene: stena je granoblastične strukture. Mikroskopski se takođe zapaža ispucalost stenske mase. Izgrađena je od krupnozrnog kvarca, pri čemu je petografskim preparatom najverovatnije zahvaćeno jedno zrno. Manje od 1% vol. stene čine neprovidni minerali, kao i nešto filosilikata koji zapunjavaju tanke prsline. Karakteristično je da kvarc pokazuje dva pravca prslina koje su najverovatnije nastale tokom deformacija koje su odgovrne i za pojavu talasastog potamnjenja. Kretanjem po navedenim rupturama došlo je do otvaranja prslina i formiranja tzv. „pull-apart'' struktura u kojima su deponovani sitnozrni filosilikatni agregati (slika 3.).



Slika 3. *Tzv."pull-apart"struktura nastala kretanjima posle deformacija, xpl*

3. PRIKAZ REZULTATA MERENJA IZDAŠNOSTI I FIZIČKO-HEMIJSKOG SASTAVA TERMOMINERALNE VODE VUČA

Terenskim istraživanjima utvrđeno je da voda ističe po dnu napravljenih bazena, sa jasno vidljivim mehurićima gasa. Konstatovano je i da je u zoni isticanja vode urađena jedna bušotina dužine 70 m gde su postavljena dva tuša i da sada voda i tu ističe. Merenje izdašnosti je bilo moguće na ostavljenom prelivu iz prvog bazena i iz bušotine.

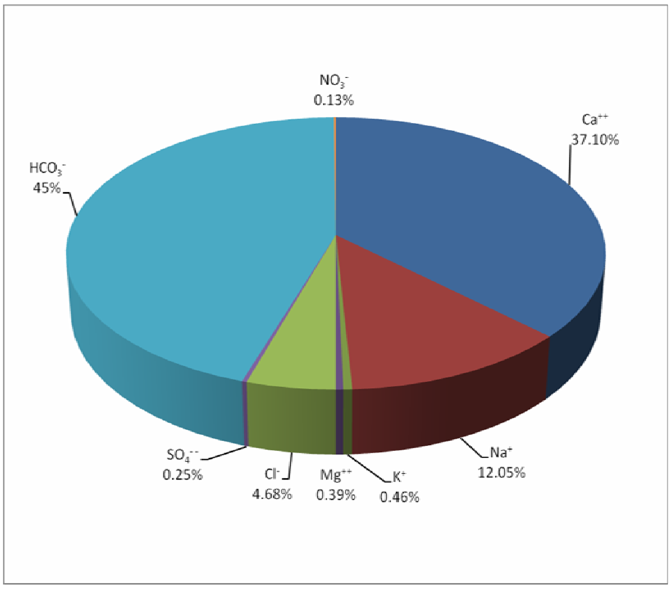
Temperatura je merena na mestima najjačeg prisustva mehurića gasa, odnosno na mestima pretpostavljenog najvećeg priliva voda kroz dno bazena i iz tuša. Režim izdašnosti i temperature praćen je u toku 2008. godine [8].

Može se zaključiti da je režim izdašnosti prilično stabilan, što ide u prilog pretpostavkama o dubokoj cirkulaciji termomineralne vode Vuča. Izdašnost se kreće u granicama od 0,8 – 1,2 l/s (preliv iz bazena) i od 0,9-1,3 l/s (isticanje iz bušotine) dok je temperatura vode oko 25°C (preliv iz bazena) i 32°C (isticanje iz bušotine).

Radi utvrđivanja kvaliteta termomineralne vode urađena je kompletna fizičko-hemijska analiza, bakteriološka analiza i radiološka analiza (tabela 2.). Pored toga analizirani su i publikovani podaci ranijih istraživanja [9]. Za potrebe izrade ovog rada prikazane su osnovne fizičko-hemijske veličine i makrokomponente (tabela 1, slika 4)

Tabela 1. *Fizičko-hemijski sastav termomineralne vode Vuča (Institut za javno zdravlje "dr Milan Jovanović-Batut", Beograd, 2008.godine)*

Redni boj	Osnovne fizičko-hemijske veličine	Sadržaj	Oznaka metode
1.	Temperatura (^{0}C)	32,0± 0,1	UP-501
2.	pH	11,5± 0,1	UP-503
3.	Boja (Pt-Co skale)	bezbojna	UP-536#
4.	Miris	Bez mirisa	UP-537#
5.	Elektroprovodljivost (μS/cm)	430±40	UP-507
6.	Ukupna tvrdoća (dH)	6,7	UP-510
7.	Utrošak $KMnO_4$(mg/l)	2,2	UP-506
8.	Suvi ostatak (mg/l)	168	UP-505
	Makrokomponente		
	Katjoni	**Sadržaj(mg/l)**	**Oznaka metode**
1.	Kalcijum (Ca^{++})	48±4	UP-516#
2.	Natrijum (Na^+)	15,6	UP-916#
3.	Kalijum (Ka^+)	0,6	UP-917#
4.	Magnezijum (Mg^{++})	<0,5	UP-517
	Anjoni		
1.	Hloridi (Cl^-)	17±1	UP-521
2.	Sulfati (SO_4^{--})	0,9±0,1	UP-521
3.	Hidrokarbonati (HCO_3^-)	164±10	UP-509



Slika 4. *Kružni dijagram hemijskog sastav*

Merenja radioaktivnosti u uzorku termomineralne vode obavljena su u Institutu za medicinu rada i radiološku zaštitu „dr Dragomir Karajović", u Beogradu čiji rezultati su dati u tabeli 2.

Tabela 2. *Tabelarni prikaz rezultata gamaspektrometrijske analize, (Institut za medicinu rada i radiološku zaštitu „dr Dragomir Karajović", Beogradu 2008. godine):*

Vrsta uzorka	^{137}Cs (Bq/l)	^{134}Cs (Bq/l)	^{40}K (Bq/l)	^{232}Th (Bq/l)	^{238}U (Bq/l)	^{226}Ra (Bq/l)
Termomineralna voda Vuča	< 0.006	< 0.002	0.10 ± 0.01	< 0.02	< 0.11	< 0.02

4. DISKUSIJA REZULTATA ISTRAŽIVANJA

Mineraloško-petrografska analiza ukazuje da užu zonu isticanja termomineralne vode Vuča izgrađuju serpentiniti koji su intezivno tektonizirani, odnosno izrasedani i ispucali s jedne strane, i pojava jedne markantne strukture kvarcita linijskog pravca pružanja duž toka Vučanske reke, s druge strane. Pretpostavlja se da su ove dve činjenice glavno hidrogeološko obeležje terena. Za njih je vezivana i geneza termomineralne vode. U okviru pukotinskog sistema u serpentinitima i kalcitima olakšana je cirkulacija podzemnih voda koje se duž rasednih struktura i mreže pukotina spuštaju do znatnih dubina u terenu, poprimajući karakteristični hemijski sastav i temperaturu. Formiranje i isticanje termomineralne vode vezano je za pukotinski tip izdani zastupljen u okviru serpentinita i sočiva kvarcita u njima. Zone prihranjivanja izdani sa termomineralnom vodom treba tražiti na znatno većim udaljenostima od isticanja duž regionalnih rasednih struktura i pukotinskih sistema s obzirom na temperature i mineralizaciju termomineralne vode.

Na osnovu rezultata fizičko-hemijske analize može se reći da od katjona dominira sadržaj kalcijuma i natrijuma. Od anjona, najviše ima hidrokarbonata, zatim hlorida, a ukupan sadržaj anjona oko tri puta je veći od sadržaja katjona. Na osnovu dosadašnjih saznanja i saznanja pristeklim navedenim istraživanjima može se reći da je termomineralna voda Vuče hidrokarbonatno – natrijumskog tipa. Analizirana voda se odlikuju visokom pH vrednošću koja se kreće do 11,5 [8].

Na osnovu pregleda osnovnih karakteristika mineralnih voda reona Šumadijsko-kopaoničko-kosovske oblasti, termomineralna voda Vuča ima sledeću formulu hemijskog sastava [9] :

$$M_{0.3} \frac{CO_{86}^{3} Cl_{12}}{Na \ + K_{98}} \ Q = 0.8$$

U svetu su retke pojave voda sa navedenim pH vrednostima. Registrovane su u Kaliforniji, Oregonu, Omanu, Novoj Kaledoniji [2] , u Kulašima u Bosni [4]. Na Zlatiboru su otkrivene kalcijum-hidroksidne vode sa pH vrednostima od 11,4-11,9 duž dva paralelna raseda i to: u reci Ribnici (Jovanova voda) i Crnom Rzavu (Lazarevo vrelo) i Kamišnoj reci u Mokroj Gori (Bela voda) [5]. Poreklo ovih voda u svežim i delimično serpentinisanim ultramafitima (lerzoliti, harcburgiti, dunit) objašnjava se savremenom serpentinizacijom primarnih anhidrovanih minerala (olivina, enstatita, diopsida) i stvaranjem hrizotil-lizarditskih serpentinskih stena [1].

Visoka alkalnost daje specijalne i vrlo ograničene balneološke karakteristike. Eventualno pijenje (hronično) ove vode može da izazove ozbiljne poremećaje u sekreciji, a takođe u varenju i apsorpciji nutrijenata i digestrivnom sistemu. Visoka alkalnost u unutrašnjosti tela bi izazvala ozbiljne poremećaje pre svega centralnog nervnog sistema i bubrega. Zbog svega ovoga ova voda može da se koristi u spoljašnoj aplikaciji (kupanje) u slučaju nekih nezapaljivih i neifektivnih kožnih bolesti, kao što je keratoza [5].

Rezultati gamaspektrometrijske analize vode (specifična aktivnost) ukazuje da su analizirane vode u skladu sa propisima za vode za piće (shodno propisima S.L.SRJ br. 9/1999) .

5. ZAKLJUČAK

Geneza, potencijalnost, kvalitet i lekovitost termomineralne vode Vuča čine lokalitet veoma interesantan za dalja proučavanja i sticanja novih saznanja.To se pre svega odnosi na dalja proučavanja u cilju dobijanja novih saznanja o genezi termomineralne vode i uslova pod kojima se formira karakterističan fizičko-hemijski sastav što se pre svega odnosi na veoma visoke pH vrednosti. Lekovitost termomineralne vode je, pre svega zbog veoma visoke pH vrednosti, i iste se mogu koristiti kao pomoćno sredstvo u lečenju različitih oboljenja kod čoveka uz lekarsku kontrolu.

LITERATURA

[1] Bames, I., O, Neil, J, (1969): The relationship between fluids in some fresh alpine-type ultramafics and possible modern serpentinization, Western United States.-Geol. Soc. Bull., 80, 1947-1960.

[2] Bames, I., O, Neil, J. And Trescasses, J.J. .(1978) Present day serpentization in New Caledonija, Oman and

Yugoslavia. Geoshim. Cosmochimica Acta, 42, 144-145.

[3] Dimitrijević, M. D.(1995). Geologija Jugoslavije. Beograd.: Geoinstitut, Beograd.: Barex

[4] Đerković, B. (1973) A new type of strongly hydrokside-sodium-calcium waterat Kulaši (Bosnia) Yugoslavia.- Bull. Sci. Acad. Sci. Arts Yugoslavia, Sect. A, 18, 134-135

[5] Maksimović Z., Ršumović M., Jovanović T, (1997): Vode iz ultramafita Zlatibora i njihov uticaj na zdravlje. Monografija: 100 godina hidrogeologije u Jugoslaviji, Beograd.: RGF

[6] Milivojević M, (2001). Elaborat o rezervama geotermalnih mineralnih voda Novopazarske banje Beograd: RGF

[7] Milentijević, G. (2005). Podzemne vode severnog dela Kosova i Metohije – iskorišćavanje i zaštita. Beograd.: Rudarsko-geološki fakultet, doktorska disertacija

[8] Milentijević, G., Nedeljković B. i dr. (2008). Elaborat o izvedenim hidrogeološkim istraživanjima po aneksu projekta “Hidrogeološka istraživanja mineralnih i termomineralnih voda severnog dela Kosova i Metohije”. Kosovska Mitrovica.: Univerzitet u Prištini, Fakultet tehničkih nauka

[9] Filipović, B., Krunić, O. i Lazić, M. (2005). Regionalna hidrogeologija Srbije. Beograd.: Rudarsko-geološki fakultet.

UDK:711.455(045)=20

Gordana Milentijevic[*], *Blagoje Nedeljkovic*[*]

HYDROGEOLOGY CHARACTERISTICS OF THE THERMO-MINERAL WATER VUČA AND ITS EFFECT ON HUMAN HEALTH[**]

Abstract

The results of new study the thermo mineral water Vuča, in a sense of genesis, potentiality evaluation, physical-chemical composition and healing properties are present in this paper. The necessary field investigation (geological and hydrogeolical) and laboratory investigations (mineralogical – petrographical analysis, physical-chemical analysis, microbiology analysis, radioactivity analysis and balneological analysis) were carried out. The investigation results are present in this paper. The thermo mineral water Vuča was evaluated as the healing water with limited usage as the pH values are very high.

Key words: *thermo mineral water, genesis, physical-chemical characteristics, balneological characteristics, healing property*

1. INTRODUCTION

The analyzed thermo mineral water sources in village Vuča, on the southern slopes of Rogozna mountain, on the left valley bank of the river Ibar, at altitude of 520 m (Figure 1). Village Vuča is situated at 17 km northwest of Kosovska Mitrovica. The occurrence of thermo mineral water Vuča is a consequence of numerous tectonic [3] and volcanic activities in the past. The folk medicine has recognized the healing properties of thermo mineral water Vuča, so it was used as medicine water. The aim of this work is to present the achieved knowledge as the result of a new geological, hydrogeological and laboratory investigations regarded to the evaluation the potentiality, genesis, physical-chemical characteristics and healing properties.

[*] *University of Priština, Faculty of Technical Sciences Kosovska Mitrovica*

** *This paper was realized as a part of the project "Studying climate change and its influence on the environment: impacts, adaptation and mitigation" (43007) financed by the Ministry of Education and Science of the Republic of Serbia within the framework of integrated and interdisciplinary research for the period 2011-2014.*

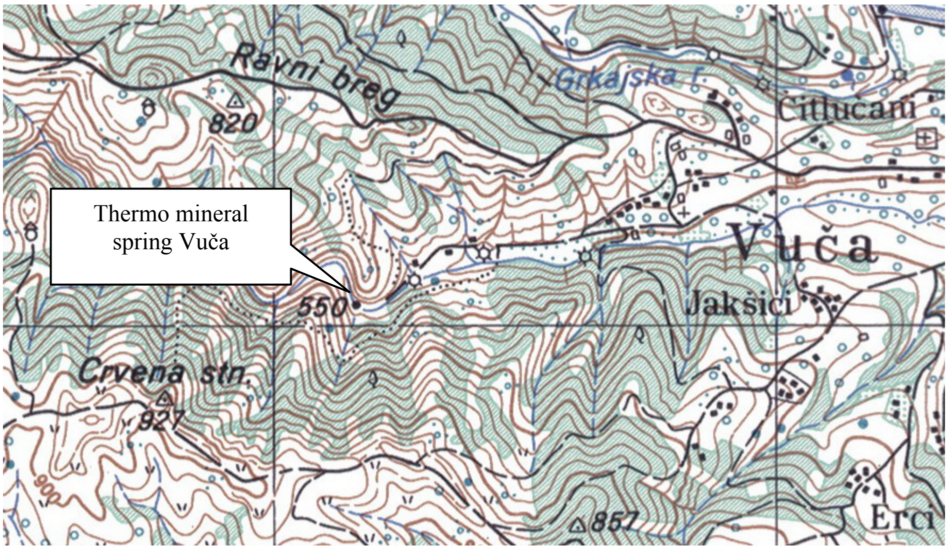


Figure 1. *Spatial location of discharge the Vuča thermo mineral water Vuča*

2. GENESIS OF THE THERMO MINERAL WATER VUČA

Genesis of thermo mineral water is related to the tecto-magmatism of Rogozna. The evolutionary development of mountain is characterized by tecto-magmatic processes, marking separate phases of development the Mesozoic and Cenozoic. The magmatism in Triassic and Upper Cretaceous is represented by the serpentinized peridotite, diabases with effusive equivalents and gabbro rocks. In the chronology of tecto-magmatism, periodites are the oldest and the most prevailing magmatic rocks. By their mineralogy compositions, they correspond to the hartzburgite type with minimum participation of equivalents-lherzolite, dunite etc. The characteristics of these rocks are serpentinization-serpentinized peridotites and serpentinites. The magnesite veins areformed by hydrothermal processes of tertiary magmatism. The degradation processes are present in serpentinites (nontronites vain magnesite).

Tertiary magmatism marks granite intrusive followed by the volcanism in several sequences, when the vulcanite are formed (dacite-andesite, quartzlatite, pyroxene-amphibolite andesite, tuffs and conglomerates). Thermo mineral water occurs in described rocks [7].

The reservoir of thermo mineral water includes a complex of carbonate Mesozoic and Paleozoic rocks. The most probably those are the Triassic limestone, as they were shortly exposed to the erosion during Upper Cretaceous. Water in the reservoir originated from the period of semi arid climate (20,000 year) with temperature of 120°C [6].

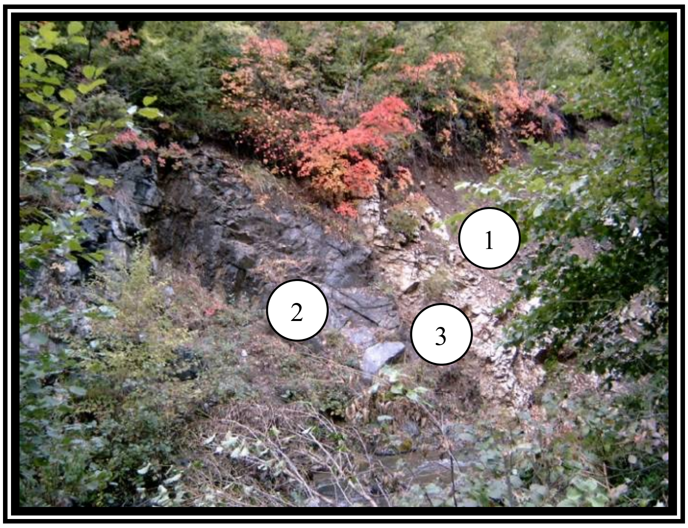


Figure 2. *Breaking related to the discharge of thermo mineral water Vuča, and sampling locations for mineralogical - petrographical analysis (Photo G. Milentijevic, 2008)*

2.1. REVIEW OF MINERALOGCAL PETRO GRAPHICAL ANALYSIS OF SAMPLES TAKEN FROM A BREAKING IN DISCHARGE ZONE OF THERMO MINERAL WATER VUČA

Within the investigation project " Hydrogeological Investigations the Mineral and Thermo Mineral Water of the Northern part of Kosovo and Metohija" financed by the Ministry of Environment and Spatial planning, the samples were taken from a penetration in thermo mineral water discharge zone and mineralogical and petrographical analyses were carried out. Mineralogical-petrographical analyses were carried out in a laboratory for mineralogy-petrography investigations at the Mining-Geology Faculty in Belgrade. The results are presented in the further text.

The analyzed samples were taken from the "roof breaking" (2), "floor breaking" (1) and ''central part of breaking'' (3) [8].

Analyze of two samples taken from the ''roof breaking'' showed that the investigated rocks are **quartzite** with calcite and epidote, and **serpentinite** formed by hartzburgite metamorphose, according to the block features and mineral compositions.

Analyze of two samples taken from the ''floor breaking'' showed that the investigated rocks are **tonalite** and **hydrothermally altered tonalite,** according to the block features and mineral compositions.

Analyze of a sample taken from the ''central breaking'' showed that the investigated rock is **quartzite**, according to the block features and mineral compositions.

Macroscopic view of rock: Quartzite is milky white rock of a granuloblastic structure, with homogenous composition.

By its macroscopic properties it matches to the sample no.1. The rock surface shows glassy glow, sharp cutting edges, ability to cut the glass, and resistant to the hydrochloric acid. The rock mass is a cracked- impregnated with fissure extending in several directions. At the rock surface and along the cracks, the limonite is observed.

Microscopic view of rock: The rock has a granuloblastic structure. The cracking of rock mass is observed by microscope. It is made of large grained quartz, where using the petrographic analysis, only one grain is observed. Less than 1% of volume is made of non-transparent minerals, as well as some phyllosilicates filling narrow fissures. It is characteristic that quartz shows two directions of fissures, most likely created during deformations responsible also for wave darkening. By moving along listed ruptures, there are opened fissures and formation of so called "pull-apart" structures with small grained phyllosilicates aggregates depo-sited (Figure 3).

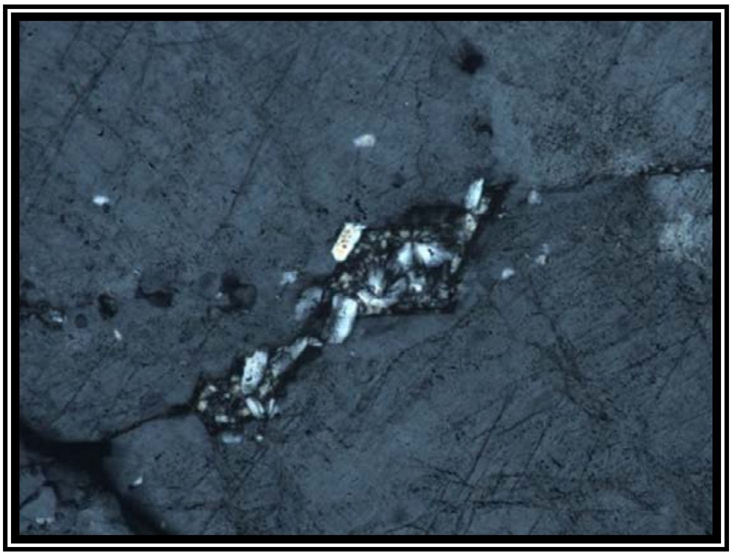


Figure 3. *So called "pull-apart" structure made by movements after deformations, xpl*

3. REVIEW OF MEASURING RESULTS THE YEILD AND PHYSICAL-CHEMICAL COMPOSITION OF THE THERMO MINERAL WATER VUČA

The field investigations have defined that water is discharged on the bottom of formed basins, with clearly visible gas bubbles. It is determined that in the zone of discharge one drill hole was made, 70 m length, and two showers were installed, and the water is discharged there. The yield measurement was possible on overflow from the first basin and drill hole. Temperature was measured on the sites of the largest presence of gas bubbles, or at the sites of assumed larges yield of water through basin bottom and from shower. The yield regime and temperatures were monitored during 2008 [8].

It can be concluded that the yield regime is a quite stable, supporting the assumptions on deep circulation of thermo mineral water Vuča. The yield is in the intervals of 0.8–1.2 l/s (overflow from the basin) and 0.9-1.3 l/s (outflow from the drill hole) and the water temperature is about 25°C (overflow from the basin) and 32 °C (outflow from the drill hole).

For determination the quality of thermo mineral water, a complete physical-chemical analysis, bacteria analysis and radiology analysis are were carried out (Table 2). In addition to this, the data published in previous studies were analzyed [9]. For the purpose of this work, the basic physical-chemical values and macro-components (Table 1 and Figure 4) were presented.

Table 1. *Physical-chemical composition of thermo mineral water Vuča (Institute for Public Health "Dr Milan Jovanovic-Batut", Belgrade, 2008)*

Order No.	Basic physical –chemical parameters	Value	Method mark
1.	Temperature (^{0}C)	32.0± 0,1	UP-501
2.	pH	11.5± 0,1	UP-503
3.	Color (Pt-Co scale)	colorless	UP-536#
4.	Odor	No odor	UP-537#
5.	Electro conductivity (μS/cm)	430±40	UP-507
6.	Total hardness (dH)	6.7	UP-510
7.	Consumption of $KMnO_4$(mg/l)	2.2	UP-506
8.	Dry residue (mg/l)	168	UP-505
	Macro components		
	Kat ions	Composition (mg/l)	Method applied
1.	Calcium (Ca^{++})	48±4	UP-516#
2.	Sodium (Na^{+})	15.6	UP-916#
3.	Potassium (Ka^{+})	0.6	UP-917#
4.	Magnesium (Mg^{++})	<0.5	UP-517
	Anions		
1.	Chlorides (Cl^{-})	17±1	UP-521
2.	Sulfates (SO_4^{--})	0.9±0.1	UP-521
3.	Hydro carbonates (HCO_3^{-})	164±10	UP-509

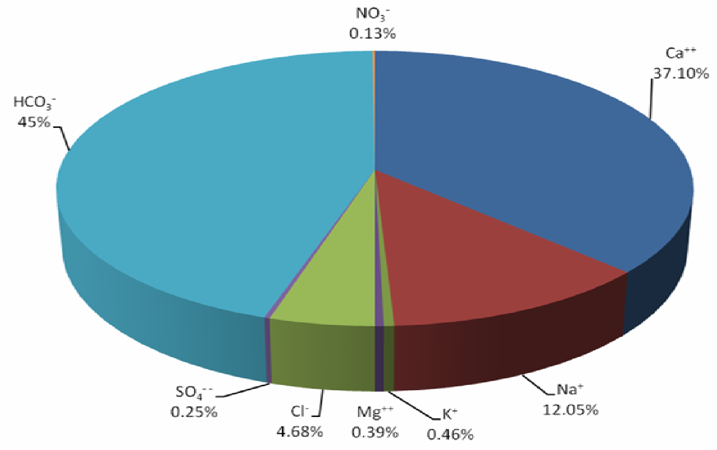


Figure 4. *Circular diagram of chemical composition*

Radioactivity measurements in a sample of thermo mineral water were carried out in the Institute for Professional Medicine and Radiological Protection "Dr Dragomir Karajovic" in Belgrade, and the results are presented in Table 2.

Table 2. *The results of gamma spectrometry analysis (Institute for Professional Medicine and Radiological Protection "Dr Dragomir Karajović" Belgrade, 2008)* .

Sample type	^{137}Cs (Bq/l)	^{134}Cs (Bq/l)	^{40}K (Bq/l)	^{232}Th (Bq/l)	^{238}U (Bq/l)	^{226}Ra (Bq/l)
Thermo mineral water Vuča	< 0.006	< 0.002	0.10 ± 0.01	< 0.02	< 0.11	< 0.02

4. DISCUSSION OF THE INVESTIGATION RESULTS

Mineralogical-petrographiccal analysis shows that the narrow discharge zone of thermo mineral water Vuča is made by serpentinites which are intensively tectonized or faulted and cracked on one side, and occurrence of an imposing structure of quartzite with direction along the river Vuča, on the other side. It is assumed that these two facts are the main hydrogeological feature of the terrain. The genesis of thermo mineral water is connected with these two facts. Within the cracking system in serpentinites and calcites, the circulation of ground water is released, discharged into the depth of terrain, getting the characteristic chemical composition and temperature along fault structures and crack network. Formation and discharge of thermo mineral water is related to the crack type spring with thermo mineral water within the serpentinite and quartzite grains in them. Recharge zone with thermo mineral water should be found in long distances from discharge zone along regional fault structures and fissure systems, considering the temperature and water mineralization.

Based on the results of physical-chemical analysis, it can be said that the calcium and sodium content is a predominant cation. For anions, the most presented are hydro carbonates, then chlorides, and total content of anions is three times higher than content of cations. Based on previous studies and present investigations, it can be said that thermo mineral water Vuča is hydrocarbonate-sodium type of water. The analyzed water has high pH value up to 11.5 [8].

Based on the review of basic characteristics the thermo mineral water in the Šumadija –Kopaonik - Kosovo region, thermo mineral water Vuča has the following formula of chemical composition [9]:

$$M_{0.3}\,\frac{CO_{86}^{3}Cl_{12}}{Na\ \ +K_{98}}\ Q = 0.8$$

There are just a few occurrences of water with such high pH values in the world. They are registered in California, Oregon, Oman, New Caledonia [2], Kulasi in Bosnia [4]. On the mountain Zlatibor , the calcium hydroxide types of water were discovered with pH values in the interval 11.4-11.9 along two parallel faults: in the river Ribnica (the Jovan water) and Crni Rzav (the Lazar spring) and the river Kamišna in Mokra Gora (Bela voda) [5]. The origin of these types of water in a fresh and partly serpentinized ultramuffites (lherzolite, hartzburgite, dunite) is explained by modern serpentinization of primary anhydrated minerals: (olivine, enstatite, diopside) and formation of chrysotile-lizardite serpentinite rocks [1].

High alcalinity gives special and very limited balneology characteristics. Possible permanent drinking of this water can cause severe disfunctions in secretion, and also digestion and absorption of nutritients in the digestive szstem. High alkalinity inside the body would cause severe disfunctions of central nervous system and kidneys. Due to these reasons, this water can be only used in the external application (bathing) in a case of some non-flamable and non-infective skin deseases, as keratosis [5].

The results of gamma spectrometrical analysis of water (specific activity) showed that the analyzed water is in accordance with the regulations for drinking water (in accordance with the regulations, "Official Gazette SRS No. 9/1999").

5. CONCLUSION

The genesis, potentiality, quality and healing properties of the thermo mineral water Vuča made this locality very interesting for further investigations and acquiring a new knowledge. This is primarily related to the further investigation for the aim of obtaining a new knowledge on genesis the thermo mineral water and conditions of formation the characteristic physical-chemical composition, first of all very high pH values. The healing properties of thermo mineral water are due to very high pH value, and these can be used as a supplementary medicine in treatment of various diseases in humans with medical control.

REFERENCES

[1] Bames I., O'Neil J., (1969), The Relationship between Fluids in some Fresh Alpine-type Ultramafics and Possible Modern Serpentinization, Western United States.-Geol.Soc. Bull., 80, 1947-1960;

[2] Bames I., O'Neil J., Trescasses, J.J., (1978), Present Day Serpentization in New Caledonia, Oman and Yugoslavia, Geoshim. Cosmochimica Acta, 42, 144-145;

[3] Dimitrijević, M.D., (1995), Geology of Yugoslavia, Belgrade, Geoinstitut Belgrade (in Serbian)

[4] Djerković, B., (1973), A New Type of Strongly Hydroxide-Sodium-Calcium Water at Kulaši (Bosnia) Yugoslavia, Bull. Sci. Acad. Sci. Arts Yugoslavia, Sect. A, 18, 134-135 (in Serbian)

[5] Maksimović Z., Ršumović M., Jovanović T., (1997), Water from Ultramafics of Zlatibor and their Impact on Health. Monograph: 100 Years of Hydrogeology in Yugoslavia, Belgrade, Faculty on Mining and Geology (in Serbian)

[6] Milivojević M., (2001), Elaborate on the Reserves of geothermal mineral water in Novopazarska Spa, Belgrade, Faculty on Mining and Geology (in Serbian)

[7] Milentijević, G., (2005), Ground Water of the North Part of Kosovo and Metohija – Utilization and Protection, Belgrade, Faculty on Mining and Geology, Doctoral Dissertation (in Serbian)

[8] Milentijević G., Nedeljković B., et all (2008), Elaborate on Realized Hydrogeological Investigations According to the Annex of Project “Hidrogeological Investigation the Mineral and Thermomineral Water of the North Part of Kosovo and Metohija”, Kosovska Mitrovica, University of Priština, Faculty of Technical Sciences (in Serbian)

[9] Filipović B., Krunić O., Lazić M., (2005), Regional Hydrogeology of Serbia, Belgrade, Faculty of Mining and Geology (in Serbian)

INSTITUT ZA RUDARSTVO I METALURGIJU BOR | YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA | UDK: 622

UDK:622.271:622.36(045)=861

Dragoslav Rakić, Laslo Čaki*, Slobodan Ćorić*, Milenko Ljubojev***

REZIDUALNI PARAMETARI ČVRSTOĆE SMICANJA VISOKOPLASTIČNIH GLINA I ALEVRITA PK "TAMNAVA –ZAPADNO POLJE"***

Izvod

Prilikom analiza stabilnosti završnih kosina na površinskim kopovima, posebna pažnja se posvećuje određivanju parametara čvrstoće smicanja. U radu je prikazan način određivanja rezidualnih parametara čvrstoće smicanja visokoplastičnih proslojaka glina i alevrita sa P.K. Tamnava –Zapadno Polje, pomoću aparata za kružno smicanje. Osim toga, dat je i osvrt na druge metode laboratorijskog određivanja parametara čvrstoće - posebno rezidualnog smicanja. Treba reći da su ova ispitivanja po prvi put izvedena u Srbiji aparatom sa kružnim smicanjem.

Ključne reči: *opit kružnog smicanja, rezidualna čvrstoća smicanja, ugao unutrašnjeg trenja.*

1. UVOD

Složeni geotehnički uslovi u severo-zapadnom delu površinskog kopa „Tamnava – Zapadno polje" često dovode do pojave nestabilnosti završnih kosina. Jedno veće klizanje masa, desilo se u blizini groblja Kalenić, gde je površina pokrenutog dela terena iznosila oko 3 ha, sa zapreminom koluvijuma od oko 180 000 m^3. Ovim klizanjem, istočna strana groblja bila je ozbiljno ugrožena, jer se našla na kritičnom rastojanju od oko 25 m od čeonog ožiljka klizišta [1]. Iz tih razloga vršena su laboratorijska ispitivanja za određivanje vršnih i rezidualnih parametara čvrstoće smicanja. Po pravilu se za projektovanje eventualnih sanacionih mera, koristi rezidualna čvrstoća smicanja, koja je na uzorcima sa pomenute lokacije, pored opita direktnog smicanja, po prvi put u Srbiji određivana i u aparatu za kružno smicanje, koristeći Rowe-vu konstrukciju.

2. ŠIRA GEOLOŠKA GRAĐA LOKACIJE

U široj građi prodručja "Tamnava Zapad" učestvuju paleozojski škriljci koji čine osnovu tercijarnog basena, dacito-andeziti premiocenske starosti, neogeni sedimenti i kvartarni sedimenti kao završni

* *Rudarsko-geološki fakultet, Beograd*
** *Institut za rudarsvo i metalurgiju Bor*
*** *Rad je proizašao iz projekta broj 36014 koji se finansira sredstvima Ministarstva za prosvetu i nauku Republike Srbije*

član sedimentacije u basenu. Pliocen, odnosno pontski kat, predstavlja najvažniji stratigrafski član, kako u ovom području tako i u celom Kolubarskom basenu i on je nosilac produktivnih horizonata u basenu. U okviru njega mogu se izdvojiti facije peskova, glina i alevrita. Ovaj kompleks ugljene serije u P.K. "Tamnava–Zapad" raslojava iz jednog jedinstvenog sloja P.K. "Tamnava-Istok" debljine 30 m u više tanjih slojeva. Ove nepovoljne promene su posledica paleoreljfa i različitih uslova sedimentacije. U građi takve, heterogene i anizotropne serije, učestvuju pre svega ugljevi, ugljevite gline, sivo-zelene gline, alevriti i peskovi. Ugljonosna serija, u celini posmatrano približno je horizontalna sa blago izraženim plikativnim formama u vidu sinformi i antiformi a generalni pad serije je od severoistoka prema jugozapadu pod blagim nagibom od oko 2^0. Na otvorenim kosinama zapaženo je prisustvo neotektonike disjuktivnog karaktera u vidu tenzionih i smičućih pukotina. Ugljonosnu seriju čine dva ugljena sloja razdvojena slojem peska. Ugalj je ksilitni i amorfni. Debljina glavnog-gornjeg ugljenog sloja se povećava od severa ka jugu i od istoka ka zapadu, pa u neraslojenom delu iznosi 10-20 m dok je u raslojenom znatno deblja i iznosi 20-60 m. Donji ugljeni sloj je takođe promenljive debljine. Najmanji je na severu 2 m, a najveći na jugu 26 m. Bitno je istaći da sa raslojavanjem opada kvantitet i kvalitet uglja. Visokoplastične slojeve predstavljaju alevriti i ugljevite i sivo-zelene gline koje prožimaju ugalj u zapadnom i jugozapadnom delu ležišta [1]. Debljine su promenljive, od nekoliko santimetara pa čak do 10 m. Međuslojni pesak nalazi se između gornjeg i donjeg ugljenog sloja i debljine je uglavnom oko 5 m, izuzetno u severozapadnom delu kopa gde iznosi i 30 m. Za potrebe ispitivanja čvrstoće smicanja, izvršen je izbor uzoraka visokoplastičnih slojeva iz kompleksa ugljene serije (ugljevite gline, sivo-zelene gline i alevriti).

3. OPŠTE O METODAMA LABORATORIJSKOG ODREĐIVANJA REZIDUALNIH PARAMETARA ČVRSTOĆE SMICANJA

Triaksijalni opit

Triaksijalni opit je najčešći način za određivnje vršnih parametara čvrstoće smicanja c', φ'. Međutim, nije pogodan za određivanje čvrstoće pri kritičnom stanju a posebno rezidualne čvrstoće, zato što u njemu nije moguće proizvesti velika pomeranja duž kliznih površina. Izvode se najčešće konsolidovani nedrenirani opiti (CU) na zasićenim uzorcima sa merenjem pornog pritiska, ili konsolidovani drenirani opiti (CD). Za praktične potrebe ovi opiti daju iste vrednosti efektivnih parametara čvrstoće smicanja ako se ispitivanja korektno izvode. Pojedinosti aparata i postupke ispitivanja detaljno je prikazao Head [2]. Pri izboru parametara čvrstoće smicanja, na osnovu više izvedenih triaksijalnih opita za istu sredinu, preporučuje se da se oni definišu iz s-t dijagrama a ne osrednjavanjem podataka dobijenih za pojedine opite ili pak crtanjem Morovih krugova svih ispitivanja na jedan dijagram.

Opit direktnog smicanja

Opit direktnog smicanja je najčešći postupak određivanja smičuće čvrstoće tla kako vršne tako i rezidualne čvrstoće oslabljenih zona (ravni) u tlu - npr. kliznih površina i pukotina u stenama. Opit direktnog smicanja se može koristi i za određivanje vršne čvrtoće sredina bez oslabnjenih zona. Rezidualna čvrstoća je najmanja čvrstoće smicanja ostvarena pri velikim pomeranjima duž klizne površine. Skempton [3] je dao tabelarni prikaz neophodnog pomeranja za određena stanja kod tla koja sadrže >30% glinenih frakcija (Tabela 1).

Tabela 1. *Neophodna pomeranja pri različitim stanjima smicanja kod tla sa >30% glinenih frakcija, Skempton (1985)*

Stanje(1)	Pomeranje mm	
	Prekonsolidovani	Normalno konsolido-vani
Vršna	0.5-3	3-6
Zapreminska promene dV=0(2)	4-10	
Pri φ'_R+1^0	30-200	
Rezidualno φ'_R	100-500	

(1) *za σ'_n <600 kPa*
(2) *potpuno omekšala čvrstoća (kritično stanje)*

Većina laboratorijskih aparata za direktno smicanje, omogućuju maksimalna pomeranja u rasponu od 6-10 mm, što je dovoljno samo za određivanje vršne čvrstoće, eventualno za delimično omekšalu čvrstoću. Zbog toga rezidualna čvrstoća smicanja, korišćenjem aparata za direktno smicanje može da se dobije samo višestrukim smicanjem. Neke od poteškoća, koje se javljaju tokom izvođenja ispitivanja u aparatu za direktno smicanje, pri određivanju rezidualne čvrstoće su:

-neophodnost ponavljanja smicanja u suprotnom (povratnom) smeru, remeti uređenost čestica u kliznoj ravni, čime se sprečava dostizanje rezidualne čvrstoće;

-vraćanje kutije aparata često dovodi do istiskivanja uzorka između kutija aparata;

-obezbeđivanje potpune zasićenosti uzorka je otežano;

-nasuprot triaksijalnom opitu, porni pritisak se ne može meriti;

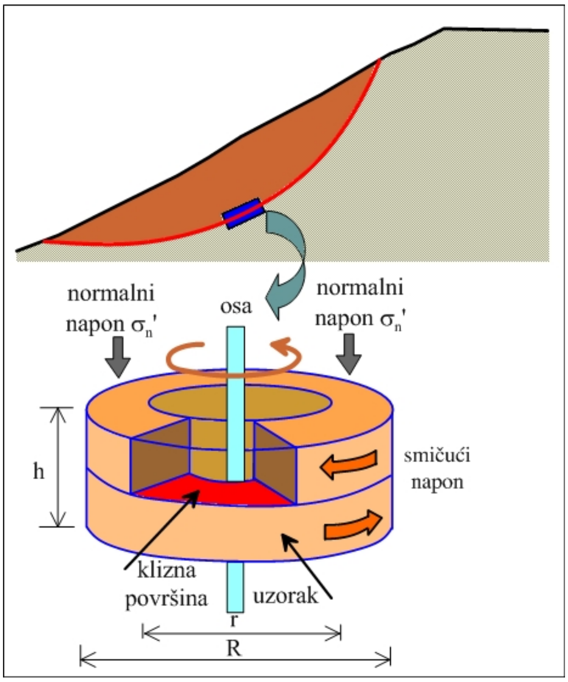
-ispitivanja u domenu “visokih vrednosti” normalnih napona dovodi do precenjivanja c' i podcenjivanja φ', kao posledica zakrivljenosti anvelope loma (ovo je karakteristično i za triaksijalni opit).

Iskustva su pokazala (Wernick, [4]) da je neophodno obezbediti paralelnost kutija tokom smicanja, posebno pri određivanju rezidualne čvrstoće. U suprotnom, za glinovito tlo, dobija se i do 4^0 veći ugao a kod peskova i do 6^0 manji efektivni ugao trenja. Ovaj zahtev, prilikom izvođenja opita, uključen je i u novije propise o ispitivanju Eurocod 7: Part 2 [5]. Za razliku od ovde izloženih praktičnih problema, postoje i teorijska ograničenja primene aparata za direktno smicanje.

Opit kružnog smicanja

Opit kružnog smicanja, kao posebna varijanta direktnog smicanja, dosta se retko izvodi ali u mnogim slučajevima omogućuje mnogo pouzdaniji način određivanje rezidualne čvrstoće smicanja. Aparat je konstrukciono složen i skup, i može se reći da spada u karegoriju istraživačke opreme. On je prvobitno bio dizajniran za ispitivanje rezidualne čvrstoće smicanja duž glatkih kliznih površina, s obzirom da omogućava neograničenu deformaciju uzorka. Opšti koncept konstrukcije aparata predložio je Hvorslev (1939), koji je kasnije iskorišćen i poboljšan od strane Bishopa [6], Bromheda [7], Savage and Sayed (1984), Sassa (1984, 1992), Hungr and Morgenstern (1984), Tika (1989), Garga i Sendano (2002) (Tabela 2), [8]. U svetu je široko prihvaćen postupak ispitivanja koji je razvijen od strane stručnjaka Imperial College of Science and

Technology (Bishop, [6]) i Norveškog geotehničkog instituta. Uopšteni princip aparata, prikazan je na Slici 1. U aparatu za kružno smicanje, ugrađuje se prstenasti uzorak koji se izlaže konstantnom normalnom opterećenju σ'$_n$ pri sprečenoj bočnoj deformaciji.



Sl. 1. *Opšti koncept aparata za kružno smicanje*

Uzorak se smiče sa konstantnom brzinom rotacije (donje u odnosu na gornju površinu uzorka). Kod Bishopovog aparata, dimenzije uzorka su: spoljni prečnik R = 150 mm, unutrašnji prečnik r = 100 mm a debljina uzorka iznosi h = 19 mm (Tabela 2) [8]. Obično se ispituje poremećeni ali moguće je ispitivanje i neporemećenog uzoraka. Bromhed [7] je opisao takođe jedan jednostavan aparat. Uzorak u Bromhedovom aparatu je nešto manjih dimenzija: spoljni prečnik je 100 mm, unutrašnji prečnik 70 mm a visina uzorka 5 mm. U ovom aparatu moguće je ispitivanje samo poremećenog uzorka - zbog male debljine uzorka. U oba ova aparata opit se obično izvodi na zasićenom uzorku u konsolidovanim dreniranim uslovima.

Tabela 2. *Osnovne karakteristike različitih aparata za kružno smicanje*

Autor	Bishop i sar. (1971)	Hungr i Morgenster (1984)	Tika (1989)	Garga i Sendano (2002)	Sassa DPRI-3 (1992)	Sassa DPRI-4 (1996)	Sassa DPRI-5 (1997)
Dimenzije							
unutr.preč. (cm)	10.16	22	10.16	9.2	21	21	12
spolj. preč. (cm)	15.24	30	15.24	13.3	31	29	18
visina uzor. (cm)	1.9	2	1.9	2.0	9	9.5	11.5
odnos vis/duž	0.75	0.5	0.75	0.98	1.8	2.38	3.83
površ. smicanja (cm^2)	101.34	326.73	101.34	72.45	408.4	314.16	141.37
max. nor. nap. (kPa)	980	200	980	660	500	3000	2000
max. brz. sm. (cm/s)	-	100	9.33	-	30	18	10
kont. obrt. momenta (max. frekven.)	Ne	Ne	Ne	Ne	da (0.5 Hz)	da (5 Hz)	da (5 Hz)
nedren opit i kontr. pornog pritiska	Ne	Ne	Ne	Ne	Da	Da	Da

Originalni aparat za kružno smicanje velike brzine (DPRI-1), sa kojim je bilo moguće obezbediti ciklične napone smicanja, razvijen je od strane prof. Sassa (1984), sa Kyoto Univerziteta [8]. Prvi dinamički aparat za kružno smicanje (DPRI-3), omogućio je da se pomoću sistema kontrole, modeliraju seizmički uticaji i izvodi nedrenirani opit sa merenjem pornih pritisaka. Aparat je kasnije nekoliko puta modifikovan (Tabela 2), tako da noviji DPRI aparati prate čitav proces loma uzorka počev od poznavanja inicijalnog statičkog i dinamičkog opterećenja, preko loma izazvanog smicanjem, velika pomeranja, promenu pornog pritiska, a u peskovima je na neki način moguće pratiti i pojavu likvefakcije.

Suština aparata za kružno smicanje je da omogući neograničenu veličinu pomeranja u jednom smeru, čime se prevazilazi nedostatak višestrukog smicanja u aparatu za direktno smicanje. Međutim, kao i kod triaksijalnog opita i opita direktnog smicanja, i kod izvođenja ovog opita, postoje određena ograničenja i poteškoće, i to:

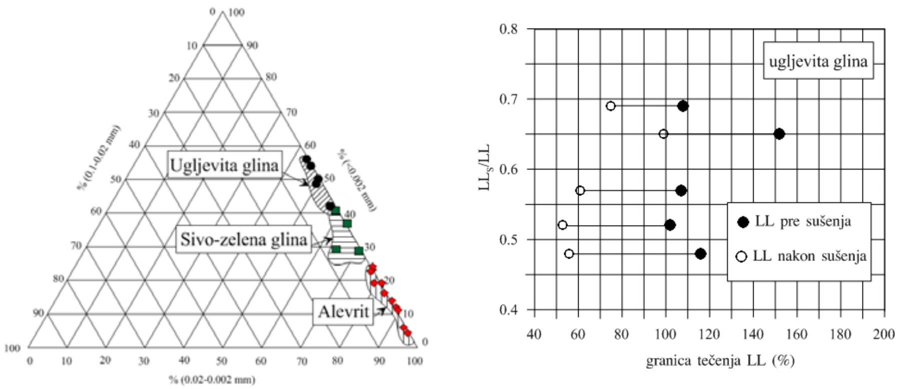
- najčešće se ispituje samo poremećeni uzorak (aparati sa malom visinom uzorka; ovo je prevaziđeno konstrukcijom DPRI aparata);
- može se dobiti samo rezidualna čvrstoća smicanja;
- uzorak teži da se istiskuje bočno između prstenova (odnosi se na starije aparate).

ANALIZA DOBIJENIH REZULTATA

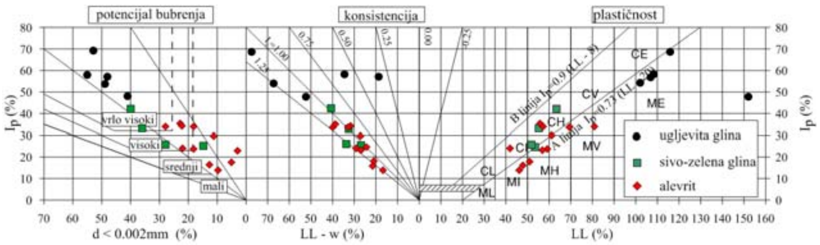
U sklopu ovog rada prikazana je analiza rezultata koji se odnose na rezidualnu čvrstoću smicanja, dobijenu na glinovitim visokoplastičnim uzorcima iz kompleksa ugljene serije. Na njima su pored opita direktnog smicanja i pratećih identifikaciono-klasifikacionih opita, izvršeni i opiti kružnog smicanja. Za razliku od ranijih ispitivanja, kada su se rezidualni parametri čvrstoće smicanja određivali na osnovu uobičajenih konvencionalnih opita (triaksijalnog i direktnog smicanja), ovog puta je za ispitivanje izabran i aparat za kružno smicanje. Sam aparat za kružno smicanje, koji je korišćen prilikom ispitivanja je konstrukcija P. W. Rowe (Manchester University) [9].

Dimenzije uzorka su iste kao i u Bishopovom aparatu a i konstrukcija je vrlo slična uz neke minimalne razlike. Svrha izvođenja opita kružnog smicanja nije bila provera dobijenih rezultata u aparatu za direktno smicanje, već je cilj bio da se rezidualni parametri čvrstoće smicanja odrede i pomoću jednog nekonvencionlnog aparata, na način kako su i predložili njegovi autori.

Identifikaciono-klasifikaciona ispitivanja, kao i opiti direktnog smicanja, izvedeni su u Laboratoriji za mehaniku tla Rudarsko-geološkog fakulteta, dok su opiti kružnog smicanja izvedeni u geomehaničkoj laboratoriji Instituta za puteve, s obzirom da je to jedina ustanova u Srbiji koja poseduje aparat za kružno smicanje. Iz pomenutog kompleksa ugljene serije, laboratorijska ispitivanja su obavljena na ukupno dvadeset uzoraka, čime su obuhvaćeni: alevriti (ukupno 11 uzoraka), sivo-zelena glina (ukupno 4 uzorka) i ugljevita glina (ukupno 5 uzoraka). Kako je klizna površina definisana u sloju alevrita, uzorci su odabrani i iz nekretanog dela terena, i iz zone klizanja. S obzirom na poreklo uzoraka, na uzorcima ugljevite gline, posebana pažnja posvećena je sprovođenju klasifikacionih ispitivanja, pre svega određivanju sadržaja organskih materija, tako da je oksidacija organskog materijala sprovedena pre analize granulometrijskog sastava (slika 2a). Pored toga, određivane su i plastične karakteristike uzoraka, stim da su ove analize sprovedena na uzorcima u prirodnom stanju vlažnosti tj. nije izvršeno sušenje (slika 2b). Na osnovu vrednosti indeksa konsistencije, moglo se zaključiti da je materijal iz nekretanog dela terena u polutvrdom stanju konzistencije (I_c=1.02-1.35), a materijal iz klizne površine u plastičnom stanju konzistencije (I_c=0.33-0.60). Kod ostalih uzoraka konstatovano je uglavnom polutvrdo ali i plastično stanje konzistencije (I_c=0.92-1.12). Rezultati ovih ispitivanja prikazani su na slici. 3.

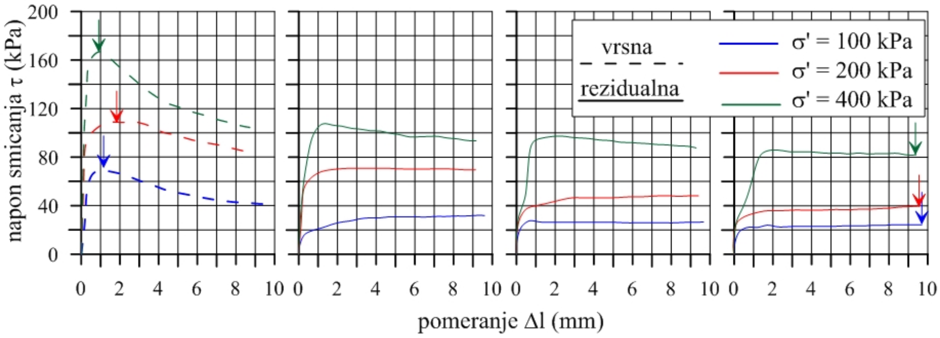


Sl. 2. *a) trougli dijagram granulometrijskog sastava; b) granice tečenja pre i nakon sušenja*

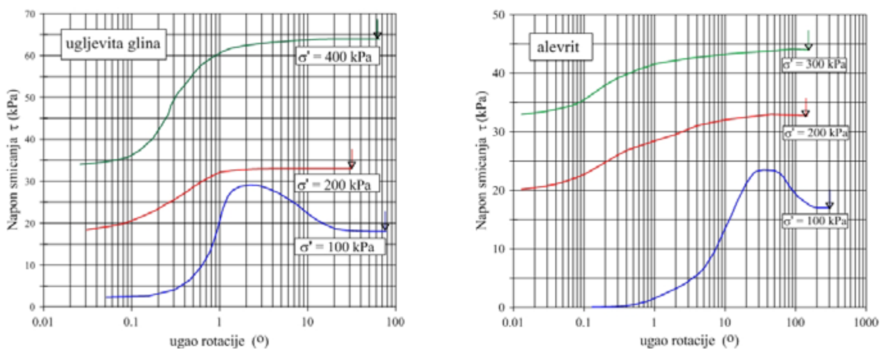


Sl. 3. *Identifikaciono-klasifikacioni pokazatelji ispitivanih uzoraka*

U aparatu za kružno smicanje ugrađivani su uzorci sa poremećenom strukturom, ali u stanju prirodne vlažnosti. Što se tiče uzorka alevrita iz klizne zone, za ispitivanje je korišćen materijal sa indeksom konsistencije koji je nešto veći od 0.70. Na jednom neporemećenom uzorku alevrita, smicanje je izvršeno i duž veštački formirane klizne površine. Osim alevrita, u aparatu za kružno smicanje ispitani su i uzorci ugljevite gline. Tok smicanja u aparatima za direktno i kružno smicanje za uzorke alevrita i ugljevite gline, predstavljen je na slikama 4 i 5.

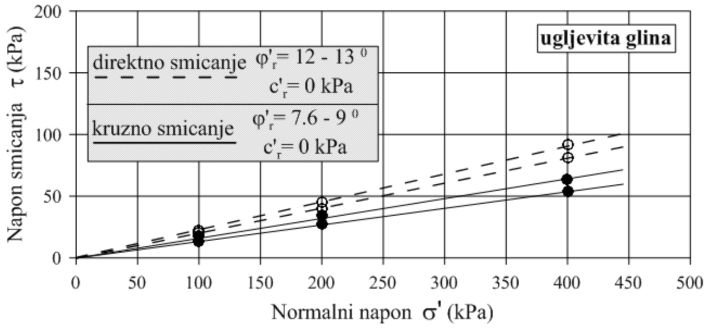


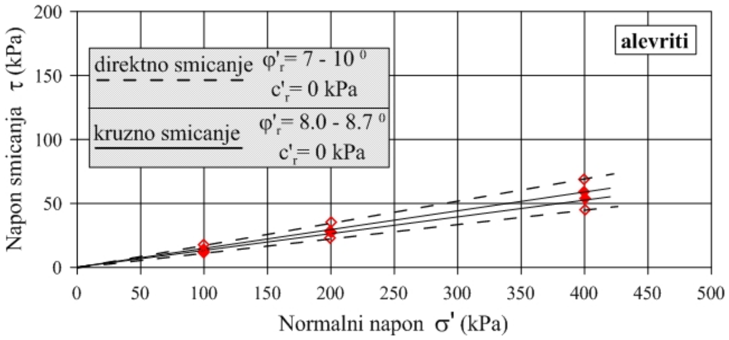
Sl. 4. *Tok smicanja u aparatu za direktno smicanje za uzorak ugljevite gline*



Sl. 5. *Tok smicanja u aparatu za kružno smicanje za uzorake ugljevite gline i alevrita*

Na osnovu izvršenih ispitivanja, dobijeni su rezidualni parametri čvrstoće smicanja koji su se za alevrite kretali u granicama od $\varphi`_r$ = 8.0 - 8.7$^0$, i $c`_r$ = 0 kPa, dok je rezidualni ugao unutrašnjeg trenja za ugljevitu glinu iznosio $\varphi`_r$= 7.6 - 9.0 ^{0} (slika 6).





Sl. 6. *Vrednosti rezidualnih parametara čvrstoće smicanja u zavisnosti od načina ispitivanja*

5. OSVRT NA IZBOR PARAMETARA ČVRSTOĆE SMICANJA

Koje parametre čvrstoće smicanja, vršne - pri kritičnom stanju, ili pak rezidualne, treba koristiti u analizi stabilnosti kosina, zavisi od prisustva odnosno od odsustva klizne površine i od stanja ispucalosti prirodne sredine [10]. U nastavku teksta daju se sledeće preporuke:

- Opšte je poznato da kada postoji klizna površina, tada se u analizama stabilnosti koriste isključivo rezidualni parametri čvrstoće smicanja. Međutim, kod umirenih klizišta, često kao posledica cirkulacije vode sa najrazličitijim jonima, dolazi do izmene mehaničkih osobina materijala duž klizne površine. Posledica toga je da se nova kretanja masa, ne odvijaju po već ranije formiranoj kliznoj površini, već dolazi do formiranja nove (Lokin i Ćorić) [11]. Prema tome u ovakvim slučajevima, prilikom analize stabilnosti, ne treba koristiti rezidualnu čvrstoću smicanja;
- Kllizne površine formirane duž međuslojnih ravni, imaju čvrstoću smicanja pribižnu rezidualnoj. U tom slučaju u praksi treba koristiti rezidualnu čvrstoću smicanja;
- Kod nasipa izgrađenih od zbijenog tla, na kojima se ne vide tragovi deformacija (pukotine), tj. nasipe

koji nisu zahvaćeni klizanjem, treba ih analizirati sa vršnom čvrstoćom smicanja;

- Ispucala tla imaju čvrstoću između vršne i rezidualne u zavisnosti od prirode ispucalosti, orijentacije pukotina, njihove kontinualnosti, širine-zeva, zapunjenosti pukotina i sl;

Pri izboru parametara čvrstoće smicanja za projektovanje, obično se povlači linija loma takva da iznad nje ostane 75 % a ispod 25% “rezultata” (linija donje kvartile), ali se može koristiti i metoda najmanjih kvadrata ili pak “donja granična” linija, zavisno od okolnosti. Međutim, bez obzira na sve gore navedeno, pri odabiru parametara čvrstoće smicanja, izuzetno je važno i “pravilno inženjersko rasuđivanje”. Razloga za to ima više a kao neki od najbitnijih su:

- slab kvalitet izvedenih ispitivanja;
- nelinearnost anvelope loma, i stim u vezi
- usvajanje niže efektivne kohezije koja pravilnije modelira ovakvo ponašanje.

Ima mnogo publikovanih radova u kojima se upoređuju čvrstoća smicanja dobijena povratnom analizom, i rezultati izvedenih opita direktnog smicanja, na materijalu iz klizne zone, kao i rezultati dobijeni višestrukim smicanjem u aparatu za direktno smicanje, odnosno, rezultati opita kružnog smicanja. Opšti zaključak za sva ova poređenja bio bi sledeći:

- opit direktnog smicanja izveden na uzorcima iz klizne površine ili duž slojevitosti, najpouzdaniji je indikator za terensku rezidualnu čvrstoću;
- opit kružnog smicanja, ili podcenjuje, ili daje približnu veličinu (-10 do +20) terenske rezidualne čvrstoće (Skempton) [3];
- višestruko direkno smicanje na glinama, precenjuje terensku rezidualnu čvrstoću za 10 do 40.

6. ZAKLJUČAK

U ovom radu, opisan je princip izvođenja opita kružnog smicanja, a prikazani su i konkretni rezultati dobijeni na uzorcima ugljevitih glina i alvrita iz tzv. uglenje serije PK. “Tamnava – Zapadno Polje”. Iako izvođenje opita iziskuje relativno složenu aparaturu za ispitivanje, na osnovu dobijenih rezultata može se zaključiti da su rezidualni parametri čvrstoće smicanja, približniji realnim vrednostima u odnosu na vrednosti koje su dobijene klasičnim opitom direktnog smicanja. Naime, povratnim analizama stabilnosti, koje su sprovedene za potrebe sanacije zapadne kosine kopa u blizini groblja Kalenić, utvrđen je rezidualni ugao unutrašnjeg trenja koji je za 1.5 - 2^0 ($\varphi'_m=6.5^0$) manji od rezidualnog ugla unutrašnjeg trenja dobijenog u aparatu za kružno smicanje. Međutim, klasičnim načinom određivanja rezidualnih parametara čvrstoće smicanja u aparatu za direktno smicanje, mobilisani ugao unutrašnjeg trenja je bio veći i za 4 - 6^0. Zato se u ovakvim slučajevima preporučuje izvođenje opita direktnog smicanja sa višestrukim smicanjem. Međutim, treba napomenuti da je zahtev EC 7 standarda da se rezidualna čvrstoća smicanja određuje kružnim smicanjem. Jedan od razloga za to je što za ispitivanje čvrstoće smicanja u aparatu za kružno smicanje, nije neophodan neporemećeni uzorak, koji se po pravilu teško obezbeđuje kada se radi o aktivnim klizištima.

LITERATURA

[1] L. Čaki, D. Rakić, V. Bogdanović, A. Pajić, Izbor parametara čvrstoće smicanja za analizu stabilnosti završne kosine "Tamnava-Zapadno polje", Treći simpozijum "Istraživanje i sanacija klizišta" Donji Milanovac, 2001., str. 493-500.

[2] K. H. Head, Manual of Soil Laboratory Testing, Vol. 1, 2 and 3. 1980, 1981, 1985, Pentech Press.

[3] A. W. Skempton, Residual strength of clays in landslides, Folded Strata and the Laboratory, Geotechnique 35, No. 1, 1985, pp. 3-18.

[4] E. A. Wernick, A "true direct shear apparatus" to measure soil parameters of shear bands, Proc. VII. ECSMFE, Brighton, Vol. 2, 1979, pp.175-182.

[5] Eurocode 7, EN 1997-1, 2007: "CEN (2007): Geotechnical design – Part 2": Ground investigation and testing

[6] A. W. Bishop, G. E. Green, V. K. Garga, A. Andersen and J. D. Brown, A new ring shear apparatus and its application to the measurement of residual strenght, Geotechnique 21, No. 4., 1971, pp. 273-328.

[7] E. N. Bromhed, A simple ring shear apparatus, Ground Engineering, Vol 12, No. 5., 1979, pp. 40-44.

[8] K. Sassa, H. Fukuoka, G. Wang, N. Ishikawa, Undrained dynamic-loading ring-shear apparatus and its application to landslide dynamics, Landslides, No. 1, 2004. pp. 7-19.

[9] P. W. Rowe, Instruction Manual for the ring shear Apparatus, 1969, pp, 1-12.

[10] D. Rakić, L. Čaki, Soil Strength Neogene Clayey Complex in the Belgrade Area, Journal of mining and geological sciences - Časopis za rudarske i geološke nauke, Volume 38. Belgrade, 2000, pp. 39-49.

[11] P. Lokin, S. Ćorić, Plenarni referat. Metodologija istraživanja klizišta. Drugi Simpozijum: Istraživanje i sanacija klizišta, Donji Milanovac, 1995.

UDK: 622.271:622.36(045)=20

Dragoslav Rakić, Laslo Čaki*, Slobodan Ćorić*, Milenko Ljubojev***

RESIDUAL PARAMETERS OF SHEAR STRENGTH THE HIGH PLASTICITY CLAY AND SILT FROM THE OPEN-PIT MINE "TAMNAVA – WEST FIELD"***

Abstract

When the final slope stability analysis for surface mining, special attention is paid to determining the shear strength parameters. This paper presents a method for determining the residual shear strength parameters from high plasticity layers of the clay and silt from the Open Pit Tamnava-West field, using a ring shear apparatus. In addition, it is reviewed to the other methods of laboratory determination of strength parameters - especially the residual shear. It should be noted that this tests were performed for the first time in Serbia device with a ring shear apparatus.

Key words: *ring shear test, residual shear strength, angle of internal friction.*

INTRODUCTION

Complex geotechnical conditions in the northwestern part of the open pit "Tamnava - West Field" often lead to the instability of the final slopes. One large mass sliding, occurred near the cemetery Kalenić, where the surface of moved part of the field was about 3 hectares, with a volume of colluvium of about 180 000 m^3. By this slide, the east side of the cemetery was severely endangered, as it was situated at a critical distance of about 25 m from the frontal scar of landslide [1]. For these reasons, the laboratory tests were performed to determine peak and residual shear strength parameters. According to the rule, for designing the possible remedial measures, the residual shear strength is used, which is on samples from the said site, in addition to the direct shear experiment for the first time in Serbia was determined using the apparatus for ring shear, using by the Rowe construction.

WIDER GEOLOGICAL STRUCTURE OF THE LOCATION

In wider geological structure of the area "Tamnava - West" , the Paleozoic shales participates that forms the base of the Tertiary basin, dacite-andesite Premiocene age, Neogene sediments and Quaternary sediments, as the final member of sedimentation in the basin. Pliocene, that is Pontian floor, is the most important

* *Faculty of Mining and Geology, Belgrade*
** *Mining and Metallurgy Institute, Bor*
*** *This paper is produced from the project no. 36014 which is funded by means of the Ministry of Education and Science of the Republic of Serbia*

stratigraphic member, both in this area and in the whole basin Kolubara and it is the holder of the productive horizons in the basin. Within it, it can be distinguished the facies sands, clays and silts. This complex of coal series at the Open Pit "Tamnava – West" is separated from a single layer of the Open Pit "Tamnava –East", width of 30m into multiple thin layers. These adverse changes are the result of paleorelief and different conditions of sedimentation. In a structure of such heterogeneous and anisotropic series, primarily are included: coal, carboniferous clay, gray-green clay, silts and sands. Coalbearing series, as a whole is approximately horizontal with a slightly pronounced compressive forms synforms forms as synforms and antiforms, and general decline of series is from the northeast to the southwest at a slight angle of about 2^0. On the open slopes, the presence of neotectonics of disjunctival character in the form of tension and shear cracks is observed. Coalbearing series include two coal layers separated by a layer of sand. Coal is xylinitic and amorphous. The thickness of the main-top coal layer increases from north to the south and from east to the west, and in a unstratified part is 10-20 m, while in the stratified is considerably thicker and 20-60 m. The lower coal layer also has a variable thickness. The smallest is 2 m in the north, and the largest in the south of 26 m. It is important to note that the stratification decreases the quantity and quality of coal. High plasticity layers are silts and carboniferous and gray-green clay that pervade the coal in the western and southwestern part of the deposit [1]. Thickness is variable, ranging from several centimeters or even up to 10 m. Interlayered sand, located between the upper and lower coal seam and the thickness, is generally around 5 m, exceptionally, in the northwestern part of the mine where it is and 30 m. For the purpose of shear strength tests, samples were selected from the complex layers of the high palsticity coal series (carboniferous clay, gray-green clay and silts).

GENERAL METHODS OF LABORATORY TESTING THE RESIDUAL SHEAR STRENGTH PARAMETERS

Triaxial experiment

Triaxial experiment is the most common method for determination of peak shear strength parameters c', φ'. However, it is not suitable for determination of strength in critical condition and, particular, the residual strength, because it is not possible to produce large displacements along the sliding surfaces. Consolidated undrained experiments (CU) are often carried out on saturated samples with pore pressure measurements, or consolidated drained experiments (CD). For practical purposes, these experiments give the same value of effective shear strength parameters if the tests are performed correctly. Details on apparatus and testing procedures in detail were shown by Head [2]. In the selection of shear strength parameters, based on several carried out triaxial experiments for the same environment, it is recommended that they are defined from s-t diagrams rather than averaging the data obtained for individual experiments or drawing the Moro circles of all tests on one diagram.

Direct shear experiment

Description of irect shear is the most common method of determining the shear strength of the soil as well as the peak and residual strength of weakened zone (planes) in the soil - e.g. sliding surfaces and cracks in the rocks. Direct shear experiment can be used for determination the peak strengths of areas without weak zones. The residual strength is the lowest shear strength achieved at high displacement along the sliding surface. Skempton [3] gave a table of necessary displacementsto the specific conditions in the soils containing >30% of clay fractions (Table 1).

Tabela 1. *Necessary movements at different states of shear in the soil with different conditions of shear with >30% of clay fractions, Skempton (1985)*

State(1)	**Displacement mm**	
	Preconsolidated	**Normally consolidated**
Peak	0.5-3	3-6
Volume changes dV=0(2)	4-10	
At φ'_R+1^0	30-200	
Residual φ'_R	100-500	

(1) *for σ'_n <600 kPa*

(2) *fully softened strength (critical state)*

Most laboratory apparatus for direct shear, allowing the maximum displacement in the range of 6-10 mm, that is enough to determine the peak strength, possibly for the partially softened strength. Therefore, the residual shear strength, using the direct shear apparatus can only be obtained by multiple shear. Some of the difficulties that arise during the experiments in the direct shear apparatus in determination the residual strength are:

- necessity of repeating the shear in the opposite (reverse) direction, alter the arrangement of particles in the sliding plane, which prevents the achievement of residual strength;
- restoring the box of apparatus often leads to displacement of the sample between the boxesof apparatus;
- ensuring a complete saturation of the sample is difficult;
- opposite to the triaxial experiment, the pore pressure cannot be measured;
- research in the domain of "high values" of normal stresses leads to overestimation of c' and underestimation of φ', as a consequence of the curvature failure envelope (this is also typical for the triaxial experiment).

Experiences have shown (Wernick) [4] that is necessary to provide parallel box during shearing, particularly in determining the residual strength. Otherwise, the clay soil, gets up to 40 higher slopes and in sand up to 60 smaller effective friction angle. This requirement, while performing the experiment, is involved in the recent regulations on examination of Eurocode 7: Part 2 [5]. Unlike here exposed to practical problems, there are theoretical limitations of use for direct shear apparatus.

Ring shear experiment

Ring shear experiment, as a special variant of the direct shear, is carried out very rarely but in many cases provides a more reliable way of determining the residual shear strength. Apparatus is constructive complex and expensive, and it can be said that it belongs into a category of research equipment. It was originally designed to investigate the residual shear strength along a smooth sliding surface, since it allows for unlimited deformation of the sample. The general concept of construction equipment proposed Hvorslev (1939), which was later used and improved by Bishop [6], Bromhead [7], Savage and Sayed (1984), Sassa (1984, 1992), Hungr and Morgenstern (1984), Tika (1989), Garga and Sendano (2002) (Table 2), [8]. The world has widely accepted the test procedure that was developed by the experts of the Imperial College of Science and Technology (Bishop, [6]) and the Norwegian Geotechnical Institute. General principle of apparatus is shown in Figure 1. In the apparatus, a ring sample is mounted that is exposed to a constant normal load σ'_n at the prevented lateral deformation.

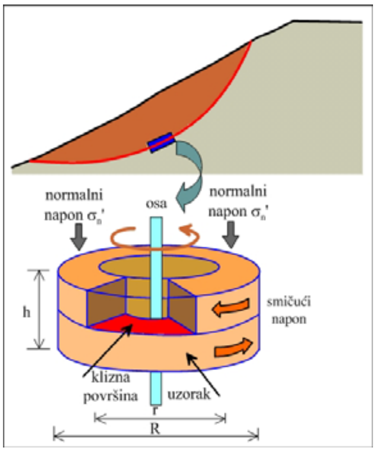


Figure 1. *General concept of the ring shear apparatus*

Sample is sheared with a constant speed of rotation (lower than the top surface of the sample). In the Bishop apparatus, the sample dimensions are: outer diameter R = 150 mm, inner diameter r = 100 mm and the sample thickness h = 19 mm (Table 2) [8]. Disturbed samples are usually tested and the undisturbed samples can be also tested. Bromhead [7] also described a simple apparatus. The sample in the Bromhead apparatus is with somewhat smaller dimensions: outer diameter 100 mm, inner diameter 70 mm and sample height 5 mm. In this apparatus, it is possible to test only a disturbed sample - due to the small thickness of the sample. In both of these apparatus, the experiment is usually performed on a saturated sample in the consolidated drained conditions.

Table 2. *Basic characteristics of different apparatus for ring shear*

Author	Bishop et al. (1971)	Hungr and Morgen-ster (1984)	Tika (1989)	Garga and Sendano (2002)	Sassa DPRI-3 (1992)	Sassa DPRI-4 (1996)	Sassa DPRI-5 (1997)
Dimensions							
Inner diameter (cm)	10.16	22	10.16	9.2	21	21	12
Outer diameter (cm)	15.24	30	15.24	13.3	31	29	18
Sample height (cm)	1.9	2	1.9	2.0	9	9.5	11.5
Ration height/length	0.75	0.5	0.75	0.98	1.8	2.38	3.83
Shear surface (cm^2)	101.34	326.73	101.34	72.45	408.4	314.16	141.37
max. normal stress (kPa)	980	200	980	660	500	3000	2000
max. shear rate (cm/s)	-	100	9.33	-	30	18	10
Continuous torque (max. frequency)	No	No	No	No	Yes (0.5 Hz)	Yes (5 Hz)	Yes (5 Hz)
Undrained experiment and control of pore pressure	No	No	No	No	Yes	Yes	Yes

Original apparatus for high-speed ring shear (DPRI-1), with which it was possible to provide cyclic shear stresses, was developed by Prof. Sassa (1984), the Kyoto University [8]. The first dynamic device for ring shear (DPRI-3), made it possible to use the control system, modeling of seismic actions and performing the undrained experiment with measurement of pore pressure. The appratus was later modified several times (Table 2), so that the newer devices DPRI monitor the whole process of sample fracture from the initial static and dynamic loading, over the fracture caused by shear, large displacements, change of pore pressure and, in sands, is somehow possible to follow also the occurrence of liquefaction.

The essence of the apparatus for ring shear is to allow the unlimited size of displacement in one direction, which overcomes the lack of multiple shear apparatus for direct shear. However, as with the triaxial experiment and the experiment of direct shear, in the execution of this experiment, there are certain limitations and difficulties, as follows:

- only the disturbed samples are usually examined (apparatus with low height of sample; this is overcome by the construction of apparatus DPRI)
- only residual shear strength can be obtained;
- sample tends to crowd out laterally between the rings (this is for the older machines).

ANALYSIS OF THE OBTAINED RESULTS

Within this paper, an analysis of the results concerning the residual shear strength, obtained on clay high plasticity samples from a complex series of coal. On them, in addition to the direct shear experiments and related identifiable-classification experiments, the experiments of ring shear were carried out. Unlike previous studies, when the residual shear strength parameters were determined on the basis of the usual conventional experiment (triaxial and direct shear), this one was chosen for testing the apparatus for ring shear. The ring shear apparatus, used in testing, was the construction of P. W. Rowe (Manchester University) [9]. Dimensions of samples are the same as the Bishop apparatus and the structure is very similar with some minor differences. The purpose of performing experiments was not checking the obtained results in apparatus for direct shear, but the aim was to determine the residual shear strength parameters using an unconvential apparatus in such a way as it was suggested by its authors.

Identification and classification testing and direct shear experiments, were performed in the Laboratory of Soil Mechanics, Faculty of Mining and Geology, and the experiments of ring shear were carried in the Geomechanical Laboratory of the Institute for Roads, since it is the only institution in Serbia, which has the ring shear apparatus. From that complex of carbon series, the laboratory tests were carried out on the total of twenty samples, which included: silts (total of 11 samples), gray-green clay (total of 4 samples) and carboniferous clay (total of 5 samples).

As the sliding surface was defined in the layer of silts, the samples were selected from non-displaced part of the field, and the slip zone. Regarding to the origin of samples, on samples of carboniferous clay, special attention was paid to the implementation of classification tests, primarily to determine the content of organic matter, so that the oxidation of organic material is carried out before the analysis of particle size distribution (Figure 2a). In addition, the plastic characteristics of sample were determined providing that the analysis was carried out on samples in natural moisture state, i.e. drying was not carried out (Figure 2b). Based on the consistency index value, it could be concluded that the material from non-displaced part of the field is in the semi-hard consistency condition (I_c=1.02-1.35), and the material from sliding surfaces is in the plastic state consistency (I_c=0.33-0.60).

In other samples, it was noted that mainly semi-hard but plastic consistency condition (I_c=0.92-1.12). The results of these tests are shown in Figure 3.

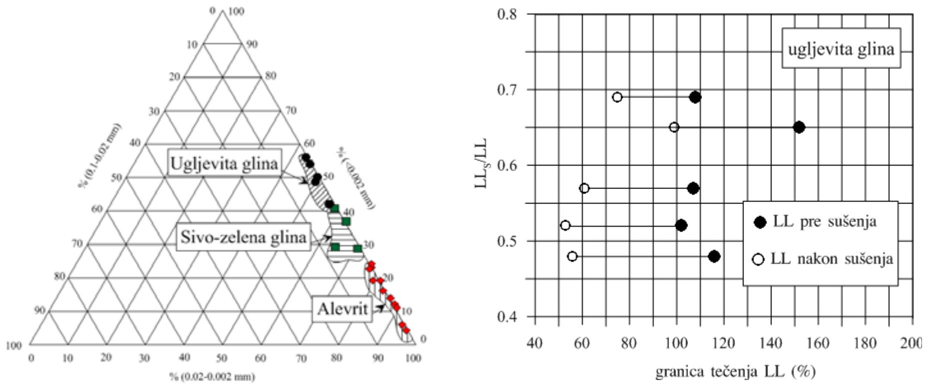


Figure 2. *a) triangle diagram of particle size distribution,* *b) yield strength before and after drying*

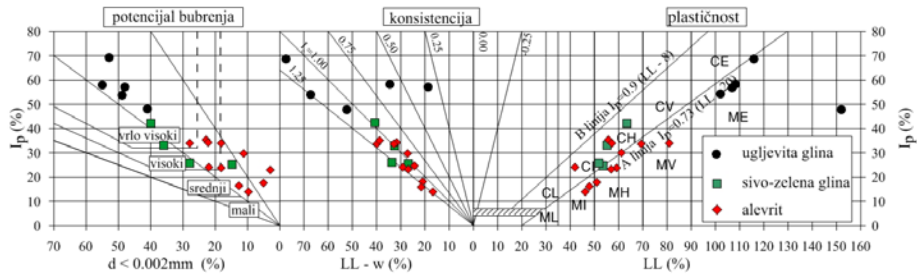


Figure 3. *Identification - classification parameters of tested samples*

Samples with disturbed structure were fitted into ring shear apparatus, but in a state of natural moisture. As for the sample of silts from sliding zone, the material with an index of consistency that is slightly higher than 0.70 was used for testing. At one undisturbed sample of silts, the shearing was done along artificially formed sliding surface. Besides silts, the samples of carboniferous clay were tested. Shear flow in apparatus for direct shear and circular shear for samples of silts and carboniferous clay, is presented in Figures 4 and 5.

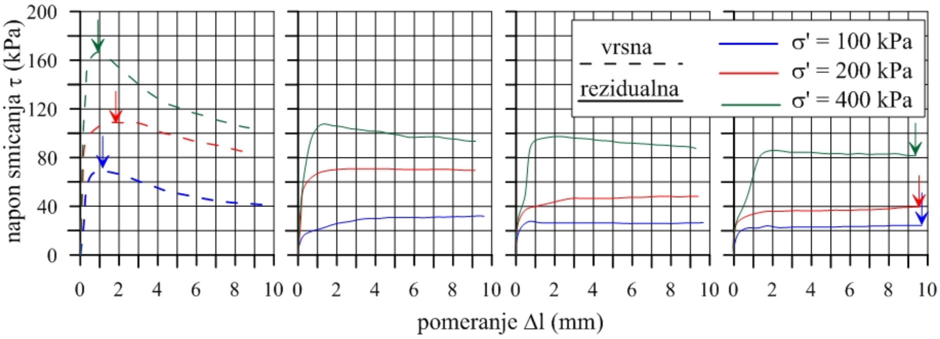


Figure 4. *Shear flow in the apparatus for direct shear for sample of carboniferous clay*

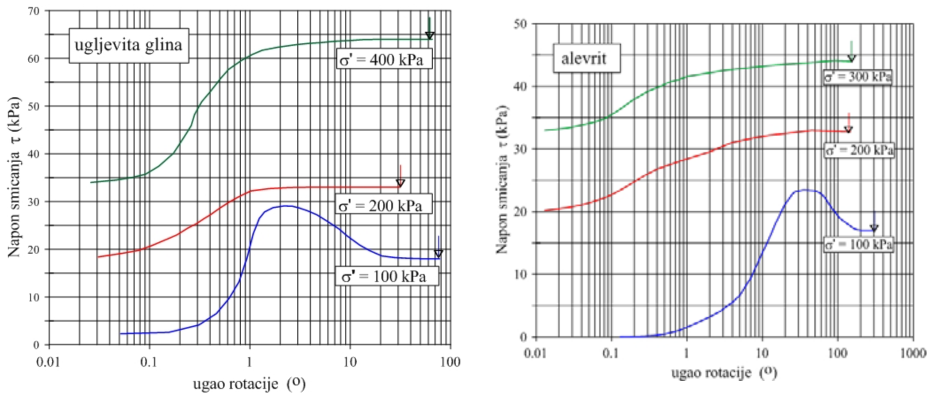


Figure 4. *Shear flow in the apparatus for ring shear for samples of carboniferous clay and silts*

Based on the realized testing, the residual shear strength parameters were obtained that were for silts in the limits of $\varphi_r^{'} = 8.0 - 8.7^0$, and c`$_r$ = 0 kPa, while the residual angle of internal friction for the carboniferous clay was $\varphi_r^{'} = 7.6 - 9.0^0$ (Figure 6).

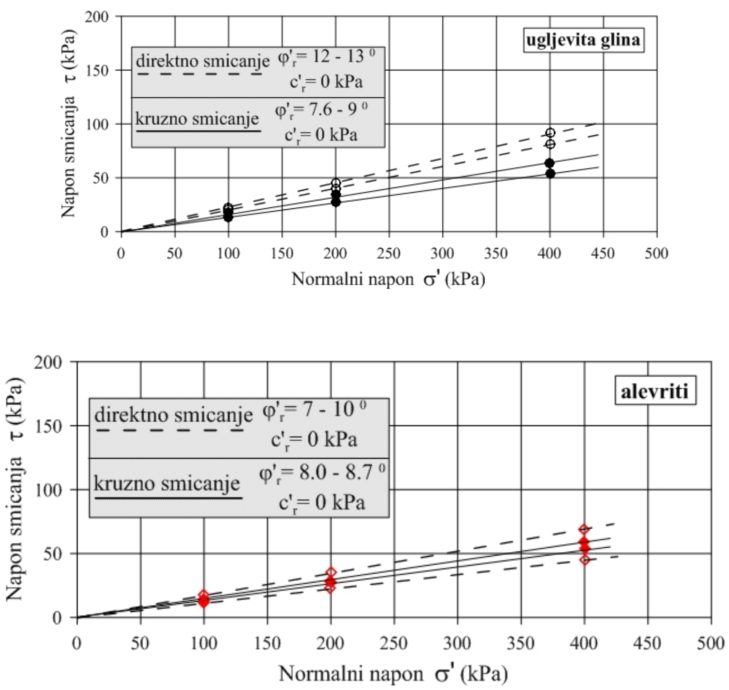


Figure 6. *Values of residual shear strength parameters depending on the method of testing*

REVIEW OF SELECTION THE SHEAR STRENGTH PARAMETERS

The parameters of shear strength, peak - in critical condition, or residual, that should be used in the analysis of slope stability, depend on the presence or absence of the sliding surfaces and state of cracks of the natural environment [10]. The following recommendations are given below:

- It is generally known that when there is a sliding surface, then the stability analysis using only the residual shear strength parameters. However, in the steady landslides, often as the result of water circulation with different ions, there are changes in mechanical properties of materials along the sliding surface. As the result, the new movements of the masses do not go along the previously formed sliding surface, but leads to the formation of a new one (Lokin and Ćorić) [11]. Therefore, in such cases, when analyzing the stability, the residual shear strength should not be used;
- Sliding surfaces formed along interlayer planes have shear strength approximate to the residual. In this case, the residual shear strength should be used in practice;
- The dams, constructed of compacted soil, on which there are no traces of deformstions (cracks), i.e. the dams that are not affected by sliding, should be analyzed from the peak shear strength;

- The cracked lands have strength between the peak and residual, depending on the nature of cracks, orientation of cracks, their continuity, width, filling of cracks, etc.;

In the selection of shear strength parameters for design, usually a line of fracture is drawn such as it remains above 75% and below 25% of the "results: (line of bottom quartile), but the method of least squares can be used, or "lower limit line", depending the circumstances. However, despite all above, in the selection of shear strength parameters, the proper "engineering reasoning" is extremely important. Reasons for this are many as some of the most important are:

- poor quality of realized testing;
- non-linearity of fracture envelope, and regarded to this
- adoption of lower effective cohesion that accurately models such behavior.

There are many published works in which the shear strength obtained by comparing the return analysis is compared with the results of experiments carried out on direct shear, the material from sliding zone, and the results obtained in the multi-shear apparatus for direct shear, i.e., the results of experiment on the ring shear. The general conclusion of all these comparisons would be as follows:

- direct shear experiment, performed on samples from the sliding surface or along the layers, the most reliable indicator of the residual field strength;
- experiment of ring shear, or underestimates, or gives the approximate size (-10 to +20) of the residual field strength (Skempton) [3];
- multiply directly shear on clay, overestimates the residual field strength by 1^0 to 4^0.

CONCLUSION

This paper describes the principle of performing the experiments of ring shear, and presents the concrete results obtained on samples of clay and carboniferous silts from the so-called coal series from the Open Pit "Tamnava – West Field". Although the performing of experiment requires a relatively complex apparatus for testing, based on the obtained results, it can be concluded that the residual shear strength parameters are closer to the real values in relation to values obtained by conventional direct shear experiment. Namely, the feedback stability analyses, carried out for the rehabilitation of the western slope pit near the cemetery Kalenić, have revealed a residual angle of internal friction, which is 1.5 - 2^0 ($\varphi_m^{'}$ =6.5^0) lower than the residual angle of internal friction obtained from in the apparatus for ring shear.

However, in the conventional method of determining the residual shear strength parameters in the apparatus for direct shear, the mobilized angle of internal friction was higher for 4 - 6^0. Therefore, the performing of direct shear experiment with multiple shear is recommended in these cases. However, it should be noted that the claim of EC 7 Standard is to determine the residual shear strength using the ring shear. One reason for this is that the shear strength test apparatus for ring shear does not require the undisturbed samples, which are normally difficult to provide when they come from the active landslides.

REFERENCES

[1] L. Čaki, D. Rakić, V. Bogdanović, A. Pajić, Selection of Parameters of Shear Strength for Analysis the Stability of Final Slope "Tamnava – West Field:, Third Symposium on Investigation and Remediation of Landslides, Donji Milanovac, 2001, pp.493-500 (in Serbian)

[2] K. H. Head, Manual of Soil Laboratory Testing, Vol. 1, 2 and 3. 1980, 1981, 1985, Pentech Press.

[3] A. W. Skempton, Residual strength of clays in landslides, Folded Strata and the Laboratory, Geotechnique 35, No. 1, 1985, pp. 3-18.
[4] E. A. Wernick, A “true direct shear apparatus” to measure soil parameters of shear bands, Proc. VII. ECSMFE, Brighton, Vol. 2, 1979, pp.175-182.
[5] Eurocode 7, EN 1997-1, 2007: “CEN (2007): Geotechnical design – Part 2”: Ground investigation and testing
[6] A. W. Bishop, G. E. Green, V. K. Garga, A. Andersen and J. D. Brown, A new ring shear apparatus and its application to the measurement of residual strenght, Geotechnique 21, No. 4., 1971, pp. 273-328.
[7] E. N. Bromhed, A simple ring shear apparatus, Ground Engineering, Vol 12, No. 5., 1979, pp. 40-44.
[8] K. Sassa, H. Fukuoka, G. Wang, N. Ishikawa, Undrained dynamic-loading ring-shear apparatus and its application to landslide dynamics, Landslides, No. 1, 2004. pp. 7-19.
[9] P. W. Rowe, Instruction Manual for the ring shear Apparatus, 1969, pp, 1-12.
[10] D. Rakić, L. Čaki, Soil Strength Neogene Clayey Complex in the Belgrade Area, Journal of Mining and Geological Sciences, Volume 38, Belgrade, 2000, pp. 39-49.
[11] P. Lokin, S. Ćorić, Plenary Paper, Research Methodology of Landslides, Second Symposium on Investigation and Remediation of Landslides, Donji Milanovac, 1995 (in Serbian)

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:622.23.05(045)=861

Ratomir Popović, Milenko Ljubojev*, Dragan Ignjatović**

SPECIFIČNOSTI RADNIH PROCESA I RADNIH OPTEREĆENJA ROTORA U PROCESU OTKOPAVANJA ROTORNIM BAGEROM

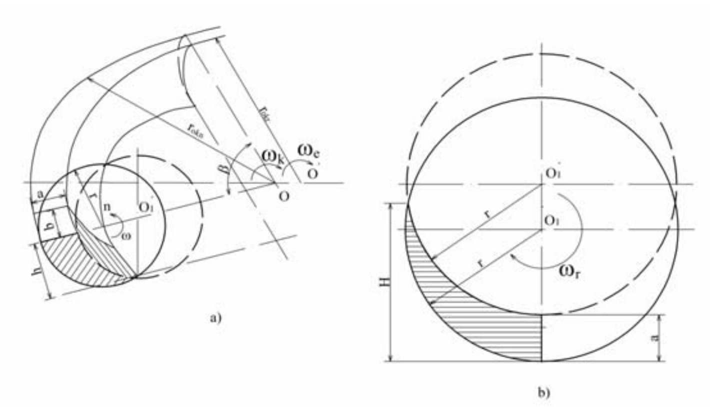
Izvod

U radu je analiziran rad rotornih bagera pri eksploataciji i proračun osnovnih parametara bagera ERG 1600 pri radu.

Ključne reči: *rotorni bager, ukupni otpor kopanju, vedrica rotora, tangencijalna komponenta otpora stenske mase, normalna komponenta otpora stenske mase, kapacitet, momenti na vratilu rotora*

UVOD

Proces otkopavanja i transport otkopane mase kod rotornih bagera je neprekidan. Rotorni bageri otkopavaju naizmeničnim rezovima koji se skidaju prilikom okretanja rotora u vertikalnoj ravni i zaokretom u horizontalnoj ravni platforme sa katarkom i rotorom koji se na njoj nalaze.



Sl. 1. *Šema otkopavanja rotornim bagerom*
- a, vertikalnim zahvatom
- b, horizontalnim zahvatom

* *Institut za rudarstvo i metalurgiju, Bor*

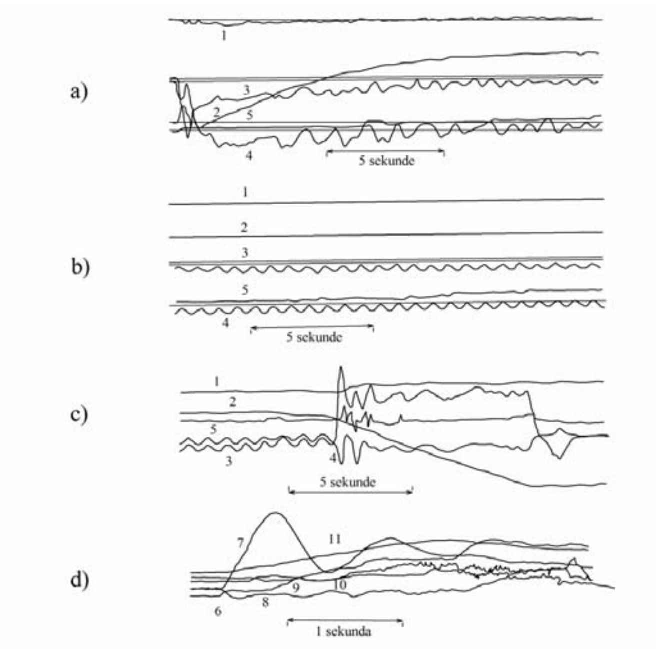
Otkopavanje se obavlja obično po vertikalnim ili horizontalnim rezovima sa nepromenljivim poluprečnikom otkopavanja r (ako se zanemare oscilacije radnog organa z) i pri određenim brzinama okretanja rotora w i katarke sa obrtnom platformom w_k.

Pri radu u vertikalnim rezovima posle svakog zaokreta platforme sa katarkom za ugao (β), čija je veličina uslovljena veličinom otkopa, bager sa katarkom ili samo katarka (ako je njena dužina promenljiva), pomera se za veličinu (a) jednaku maksimalnoj dubini reza.

Pri radu sa horizontalnim rezom posle svakog zaokreta platforme sa katarkom za ugao (β), katarka sa rotorom se spušta za veličinu (a).

Usled periodičnog ulaska i izlaska vedrica iz zahvata otkopavanja, i spoljašnje opterećenje na rotoru u procesu otkopavanja ima periodični karakter. Promena vanjskog opterećenja uvećava se dopunskim promenama sila rezanja koje potiču od neistovetnosti mehaničkih svojstava masiva koji se otkopava. Promene spoljašnjeg opterećenja su periodične i izazivanju oscilacije i do-punska dinamička opterećenja na katarci i rotoru, zatim na elementima konstrukcije gornjeg stroja i konstrukcije osnove rotornog bagera. Ta dopunska dinamička opterećenja mogu biti vrlo opasna za elemente konstru-kcije u trenutku pojave rezonantnih oscilacija.

Na sl. 2. prikazano je niz oscilograma koji karakterišu oscilacije i dinamička opterećenja koja se javljaju pri radu transportera odlagača.



Sl. 2. *Oscilogrami koji karakterišu dinamička opterećenja pri radu transportera odlagača OŠ 4500/180*

a) puštanje transportera u rad bez opterećenja, b) uspostavljeno kretanje transportera bez opterećenja, c) kočenje transportera bez opterećenja, d) opterećenje na različitim mestima trake po dužini pri puštanju opterećenog transportera (krive 6, 7, 8, 9, 10 i 11)

1. naponi u gornjoj sekciji odložne konzole, 2. brzina obrtanja motora glavnog pogonskog bubnja, 3 i 4. obrtni moment na vratilu pogonskog bubnja, 5. hod kolica zateznog mehanizma

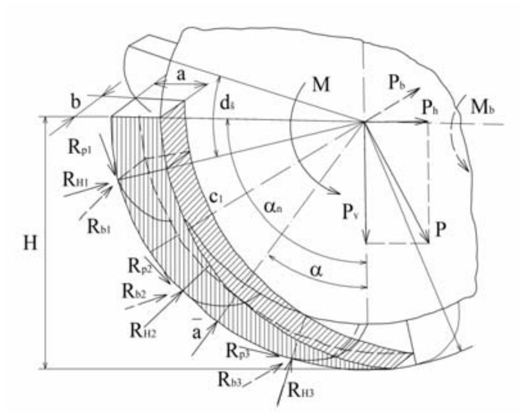
1. SPOLJAŠNJE OPTEREĆENJE

Od svih oblika spoljašnjih opterećenja i otpora koji se savladavaju pogonom najveću specifičnost za razmatrane mašine predstavljaju:

- otpor stenske mase u procesu otkopa-vanja,
- opterećenja koja se javljaju pri radu transportera.

1.1. Određivanje spoljašnjeg opterećenja pri otkopavanju bez oscilacija rotora i promena sile rezanja

Ukupni otpor otkopavanju stenske mase rotornim bagerom je zbir otpora koji se javljaju na pojedinim vedricama koje su u zahvatu stenske mase, sl. 3. Ukupan otpor kopanju na bilo kojoj vedrici sastoji se od tri komponente.



Sl. 3. *Sile koje deluju po obodu rotora*

$$R_i = \sqrt{R_{Ti}^2 + R_{Ni}^2 + R_{Bi}^2} \quad (1)$$

gde je:

- RTi, tangencijalna komponenta otpora stenske mase u vertikalnoj ravni
- RNi, normalna komponenta otpora stenske mase u vertikalnoj ravni
- RBi, bočna komponenta otpora stenske mase

Osnovna komponenta koja karakteriše otpor stenske mase kopanju je R_{Ti}, i obično se naziva otporom kopanju u procesu rezanja i može se odrediti za bilo koji položaj vedrice u zahvatu.

$$R_{Ti} = L_{sr.} \cdot K_L \quad (2)$$

$$R_{Ti} = F_{sr.} \cdot K_F \quad (3)$$

gde je:

- L, dužina dela režuće konture vedrice koja se nalazi u zahvatu i jednaka je

polovini obima preseka reza,

- F, površina preseka reza,
- K_L, koeficijent otpora kopanju [KN/m'],
- K_F, koeficijent otpora kopanju [KN/m^2],

U datom slučaju koeficijentima K_L i K_F obuhvaćeni su ne samo otpori kopanju, nego i otpori rezanju, pomeranje otkopane mase u vedrici i otpori na obodu radnog točka. Praktično, za određivanje srednjih veličina komponenti R_{Ni} i R_{Bi} polazi se od veličina tangencijalne komponente R_{Ti}.

$$R_{Ni} = \psi_n \cdot R_{Ti} \quad (4)$$

$$R_{Bi} = \psi_b \cdot R_{Ti} \quad (5)$$

gde su:

- ψ_n i ψ_b, koeficijenti dobijeni eksperimentom čije vrednosti uglavnom zavise od fizičko-mehaničkih svojstava otkopavane stenske mase i konstruktivnih karakteristika radnog organa, odnosno parametara rezanja a i b i bočne i obodne brzine V_b i V_o, sl. 3

Pri otkopavanju stenskih masa I i II kategorije $\psi_n = 0{,}4 \div 0{,}5; \psi_b = 0{,}25 \div 0{,}35$, a pri otkopavanju stenske mase IV i V kategorije njihove se vrednosti kreću:

$$\psi_n = 0{,}5 \div 0{,}8; \psi_b = 0{,}35 \div 0{,}50$$

Zavisnost R_{Ti} od ugla rezanja α, sl. 3, ili od vremena t pri konstantnoj brzini obrtanja rotora, svrsishodno je koristiti izraze (2) i (3) pošto eksperimentalna istraživanja pokazuju da prosečna vrednost K_L ostaje praktično konstantna na čitavoj dužini rezanja. Pošto je koeficijent K_L za mašine različite klase različit, a srednja vrednost K_F zavisi od fizičko-mehaničkih svojstava stenske mase koja se otkopava i konstrukcije radnog organa. Prema tome, celishodno je za korišćenje K_L pri funkciji $R_{Ti} = f(a)$ naći njegovu vezu sa K_F.

Polazeći od sl. 3 šrafiranog reza i jednakosti utrošenog rada moguće je napisati:

$$K_L \cdot r\int_0^{\alpha_n} L \cdot d\alpha = K_F \cdot r\int_0^{-\alpha_n} F \cdot d\alpha \quad (6)$$

gde je:

- α_n, polazni ugao rezanja.

Izražavajući tekuću vrednost dubine reza a preko $a_{\max} = a \cdot \sin\alpha$, moguće je napisati tekuće vrednosti F i L u sledećim oblicima

$$L = \frac{a \cdot \sin\alpha}{\sin\gamma} + b \approx a\sin\alpha + b \quad (7)$$

- γ, ugao nagiba bočne rezne ivice → 90°

$$F = a \cdot b \cdot \sin\alpha \quad (8)$$

$$\begin{aligned} K_L \cdot r\int_0^{-\alpha_n} (a \cdot \sin\alpha + b)d\alpha = \\ = K_F \cdot r\int_0^{-\alpha_n} a \cdot b \cdot \sin\alpha \cdot d\alpha \end{aligned} \quad (9)$$

gde je:

- a i b, maksimalna dubina i širina reza.

$$K_L \cdot r|a \cdot \cos\alpha + ba|_0^{\alpha_n} =$$
$$= K_F \cdot r|ab \cdot \cos\alpha|_0^{\alpha_n}$$

$$K_L \cdot r[-a + a\cos\alpha + b\alpha n] =$$
$$= K_F \cdot r(-ab + ab\cos\alpha n)$$

nakon sređivanja dobije se:

$$K_L = K_F \frac{ab(1-\cos\alpha n)}{\alpha n \cdot b + a(1-\cos\alpha n)} \quad (10)$$

Polazeći od pretpostavke da je kapacitet bagera funkcija parametara K_L i K_F i osnovnih parametara rotora, to ćemo koeficijente otpora kopanju K_L i K_F izraziti preko kapaciteta i osnovnih parametara rotora:

$$\frac{b}{a} = \nu = const. \text{ koeficijent rotora.} \quad (11)$$

Satni kapacitet će biti:

$$Q = 60 \cdot q \cdot nz\left[m^3/h\right] \quad (12)$$

gde je:

- q [m^3], zapremina vedrice
- z, broj vedrica na rotoru
- n, broj obrtaja rotora u minuti

Uzimajući u obzir i koeficijent rastresitosti k_r iz uslova promene materijala u vedrici moguće je napisati:

$$q = a \cdot b \cdot h = \frac{Q}{60 \cdot n \cdot z \cdot k_r} \quad (13)$$

gde je:

- h, ukupna visina rotora, i izražena preko poluprečnika rotora r je:

$$h = r(1-\cos\alpha n) \quad (14)$$

Uvrštavajući izraz (14) u (13) dobije se:

$$ab = \frac{Q}{60 \cdot n \cdot z \cdot r(1 - \cos \alpha n) \cdot k_r} \quad (15)$$

$$b = \sqrt{\frac{Q}{60 \cdot n \cdot z \cdot k_r \cdot h}}$$

$$a = \frac{1}{\nu} \sqrt{\frac{Q}{60 \cdot n \cdot z \cdot k_r \cdot h}}$$

Uvrštavajući izraze za a i b i (14) u izraz (10), dobije se:

$$K_L = \frac{K_F}{\alpha n + \frac{1}{\nu}(1 - \cos \alpha n)} \cdot \sqrt{\frac{Q(1 - \cos \alpha n)}{60\nu \cdot n \cdot z \cdot r \cdot k_r}} \quad (16)$$

Za $\alpha n = \frac{\pi}{2}$, što se najčešće usvaja u proračunima:

$$K_L = \frac{2K_F}{\pi + \frac{2}{\nu}} \cdot \sqrt{\frac{Q}{60 \frac{2}{\pi} \cdot n \cdot z \cdot r \cdot k_r}} \quad (17)$$

Polazeći od uslova minimalne specifične potrošnje $\nu = \frac{2}{\pi}$, tada će biti:

$$K_L = K_F \sqrt{\frac{Q}{120\pi \cdot n \cdot z \cdot r \cdot k_r}} \quad (18)$$

Ne uzimajući u obzir oscilacije rotora, veličinu srednjeg momenta na vratilu rotora od otpora kopanju možemo izraziti na sledeći način:

$$M_{sr.} = P_{sr.} \cdot r \quad (19)$$

gde je:

- $P_{sr.}$, srednji tangencijalni otpor kopanju stenske mase

Polazeći od jednakosti utrošenog rada može se napisati:

$$R_{sr.} = \int_0^{-\alpha n} d\alpha = K_L \int_0^{-\alpha n} (a \cdot \sin \alpha + b) d\alpha \quad (20)$$

gde je:

- α_n, ugao između vedrica
- R_{sr}, srednja vrednost tangencijalne komponente otpora kopanju pri zahvatu jedne vedrice,

$$R_{sr.} = \frac{K_L[a(1 - \cos \alpha_n + b \cdot \alpha_n)]}{\alpha_n}$$

Polazeći od prethodnog izraza (10) dobije se:

$$R_{sr.} = \frac{K_F \cdot a \cdot b(1 - \cos \alpha_n)}{\alpha_n}$$

ovo se odnosi na jednu vedricu. Ukupna sila će biti jednaka:

$P_{sr.} = R_{sr.} \cdot \frac{\alpha_n}{n} = R_{sr.} \cdot \frac{\alpha_n \cdot z}{2\pi}$, te će ukupni srednji tangencijalni otpor kopanju biti:

$$P_{sr.} = \frac{K_F \cdot a \cdot b(1 - \cos \alpha_n) \cdot z}{2\pi} \quad (21)$$

Uvrštavajući izraze *ab* iz (15) u (21)

$$ab = \frac{Q}{60 \cdot nzrk_r(1 - \cos \alpha_n)}$$

$$P_{sr} = K_F \frac{Q}{120\pi \cdot n \cdot r \cdot k_r} \quad (22)$$

$$M_{sr} = \frac{K_F \cdot Q}{120\pi \cdot n \cdot k_r} \quad (23)$$

Srednja vrednost snage na vratilu rotora koja se angažuje pri kopanju stenske mase je:

$$N_{sr.} = \frac{QK_F}{360 \cdot k_r}[kW] \quad (24)$$

2. DEFINISANJE RAČUNSKIH OPTEREĆENJA NA OSOVINI ROTORA UZROKOVANA OTPOROM U PROCESU KOPANJA

Polazeći od sl. 3 spoljašnji otpor kopanju moguće je zameniti zajedničkim opterećenjima koja deluju na osi rotora, momentom *M* i silom *P*, koji deluju u ravni rotora zajedničkom silom P_B od bočnih opterećenja pri kopanju i momentom M_B, koji uvrće katarku.

Svrsishodno je silu P razložiti na vertikalnu i horizontalnu komponentu (P_V i P_H), pošto se P_V pojavljuje kao osnovna sila koja pobuđuje konstrukciju u vertikalnoj ravni. Da bi se objasnili dinamički efekti na elemente konstrukcije, neophodno je objasniti prirodu promene momenta M na vratilu rotora u zavisnosti od otpora u procesu kopanja u funkciji ugla zaokreta rotora α i vremena zaokreta t.

U skladu sa izrazima (2), (3) i (7), moment od tangencijalnog otpora kopanju na bilo kojoj vedrici R_{Ti} u odnosu na osu rotora O, sl. 3, može se napisati

$$M_i = r \cdot K_L (a \cdot \sin\alpha + b) \qquad (25)$$

gde je:

- r, radijus rotora do režućeg ruba vedra.

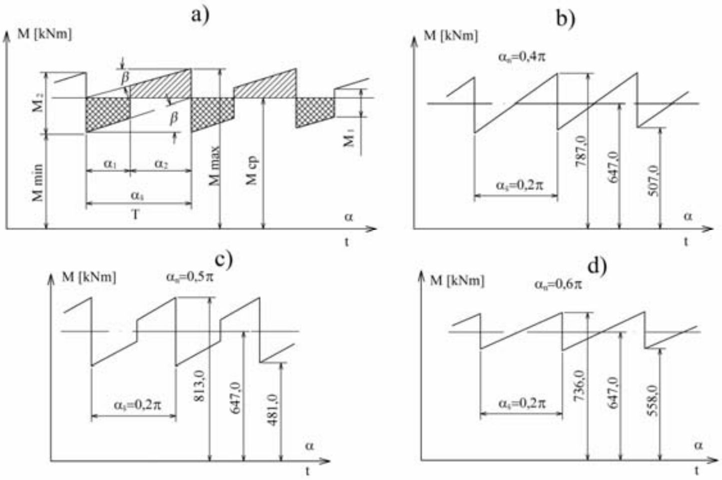
Polazeći od realne pretpostavke, da pri ulasku vedrice u proces rada vertikalnim rezom, moment na vratilu rotora brzo raste za veličinu:

$$M_1 = r \cdot K_L \cdot b \qquad (26)$$

a izlazak vedrice iz procesa rada praćen je smanjenjem ukupnog momenta na vratilu rotora, a koji potiče od otpora kopanju za vrednost:

$$M_2 = r \cdot K_L \cdot (a \cdot \sin\alpha_n + b) \qquad (27)$$

Ne uzimajući u obzir promenu otpora kopanju na račun odlamanja, promenu fizičko-mehaničkih svojstava po dužini reza i drugih faktora, koji izazivaju promenu sila otpora kopanju, i ako se pretpostavi da u periodu vremena od ulaska vedrice u zahvat do njenog izlaska menja se moment otpora na vratilu M_i, tada će dijagram promene ukupnog momenta na vratilu rotora $M_{(\alpha)}$ od otpora kopanju na svim vedricama moći da se izrazi redom, sl. 4.



Sl. 4. *Promena računskog momenta otpora kopanju na vratilu rotora*

a) opšti slučaj

b) rotorni bager ERG-1600 pri odgovarajućem $\alpha_n = 0{,}4\pi$

c) $\alpha_n = 0{,}5\pi$

d) $\alpha_n = 0{,}6\pi$

Prema izrazu (23) određuje se srednja vrednost momenta $M_{sr.}$ otpora kopanju. Veličine skokova M_1 i M_2 potrebno je odrediti za uglove α_1 i α_2 koji odgovaraju radu rotora sa manjim i većim brojem vedrica u radu. Veličinu uglova α_1 i α_2 određujemo sa sl. 3.

$$\alpha_1 = (m+1)\alpha_{š} - \alpha_n \quad (28)$$

$$\alpha_2 = \alpha_n m \alpha_{š} \quad (29)$$

gde je:

- $a_{š}$, uglovni korak vedrice
- $m = \dfrac{\alpha_n}{\alpha_{š}}$

$\dfrac{\alpha_1}{\alpha_{š}} = (m+1) - j$ i $\dfrac{\alpha_2}{\alpha š} = j - m$, tj. višak u ($j$) od celog broja ($m$) biće deo α_2 u $\alpha_{š}$, a nedostajući deo do sledećeg celog broja (m+1) iznosiće deo α_1 u $\alpha_{š}$.

Kod stvarnih konstrukcija rotora sa z = 6 do 14 vedrica i uglovima α_n=(0,4 do 0,6) π, (j) se nalazi u granicama 1,2 do 4,2, odgovarajući broj koji se istovremeno nalazi u sadejstvu sa stenskom masom će iznositi od 1-2 i od 4-5. Kod toga će većina vedrica (m+1) učestvovati u kopanju za vreme trajanja ugla α_2, a manji broj vedrica (m) menjaće se u toku ugla α_1.

Veličina ugla β, sl. 4a, potrebna za konstrukciju $M_{(\alpha)}$, određuje se kao $tg\beta = \dfrac{M_2 - M_1}{\alpha š}$. Veličine M_{max} i M_{min} određuju se u opštem obliku polazeći od uslova jednakosti šrafiranih površina koje se nalaze počev odozgo prema dole od $M_{sr.}$

$$M_{\max} = M_{sr.} + \frac{M_2}{2} - M_1[j - (m+0,5)] \quad (30)$$

$$M_{\min} = M_{sr.} - \frac{M_2}{2} - M_1[j - (m+0,5)] \quad (31)$$

Pri celom broju vedrica koje se istovremeno nalaze u sadejstvu sa stenskom masom, tj. kada je $j = \dfrac{\alpha_n}{\alpha_{š}} = m$ biće

$$M_{\max} = M_{sr.} + \frac{M_2 - M_1}{2} \quad (32)$$

$$M_{\min} = M_{sr.} - \frac{M_2 - M_1}{2} \quad (33)$$

Naglašavamo, da pri konstantnoj brzini obrtanja rotora, konstruisani dijagram $M = f(\alpha)$ predstavlja istovremeno i dijagram promene momenta u funkciji vremena $M = f(t)$.

Primera radi, razmotrićemo rad bagera ERG 1600 pri kapacitetu Q = 3750 [m^3/h] i broju obrtaja rotora n = 3,7 [min^{-1}] u stenskoj masi sa K_F = 0,3 [MPa] i sa koeficijentom rastresitosti k_r = 1,25.

Pri navedenim uslovima, tabelarnoi dijagramski smo prikazali $M = f(\alpha)$ i $M = f(t)$.

Koristeći izraze za a i b i izraze (16) i izraz (23), izračunate su veličine K_L, a, b i $M_{sr.}$, a zatim veličine skokova momenata M_1 i M_2 za uglove $\alpha_n = 0,4\pi$, $\alpha_n = 0,5\pi$ i $\alpha_n = 0,6\pi$, tj. kada se u sadejstvu sa stenskom masom nalazi srednji broj vedrica, a koji je jednak 2; 2,5 i 3.

Dobijeni podaci su prikazani u tabeli 1 i na dijagramima, sl. 4.

Tabela 1.

α_n **[rad.]**	$M_{sr.}$ **[kNm]**	***a*** **[m]**	***b*** **[m]**	K_L **[kN/m]**	M_1 **[kNm]**	M_2 **[kNm]**
0,4π	647,0	0,796	0,436	65,5	161,0	444,0
0,5π	647,0	0,614	0,389	58,5	129,0	332,5
0,6π	647,0	0,518	0,356	53,5	107,4	256,0

LITERATURA

[1] R. Popović sa saradnicima, Elaborat o terenskim istraživanjima uticajnih parametara na režim rada bagera schRS 315/1,5-12,5 u uslovima kopanja sloja uglja na P.K. „Gračanica“ Gacko, Institut za rudarska istraživanja, Tuzla, 1983.

[2] R. Popović, D. Đukić, Utvrđivanje uticaja radnih parametara rotornog bagera na njegov efektivni kapacitet u uglju, V Jugoslovenski simpozijum o površinskoj eksploataciji mineralnih sirovina, Skoplje, 1983.

[3] M. Ljubojev, R. Popović, Osnove rušenja stena primenjenom mehanizacijom pri eksploataciji čvrstih mineralnih sirovina (knjiga u štampi) Institut za rudarstvo i metalurgiju, Bor

[4] M. Ljubojev, R. Popović, Problematika eksploatacije stenskog materijala, Međunarodni naučno-stručni simpozijum u povodu 120 godišnjice Kreke, Tuzla, 2005.

[5] R. Popović, M. Ljubojev, M. Ivković, Mehanizacija i produktivnost preduslov promenama rudnika sa podzemnom eksploatacijom, Međunarodni naučno-stručni simpozijum u povodu 120 godišnjice Kreke, Tuzla, 2005.

[6] R. Popović, M. Ljubojev, Osnove rušenja stena primenjenom mehanizacijom pri eksploataciji čvrstih mineralnih sirovina, Monografija, Bor, 2011.

MINING AND METALLURGY INSTITUTE BOR YU ISSN: 1451-0162
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS UDK: 622

UDK:622.23.05(045)=20

Ratomir Popović, *Milenko Ljubojev*, *Dragan Ignjatović**

SPECIFICITY OF WORK PROCESSES AND WORK LOADS OF ROTOR IN THE EXCAVATION PROCESS USING THE BUCKET WHEEL EXCAVATOR

Abstract

This work gives an analysis of operation the bucket wheel excavator in the exploitation and calculation the basic parameters if excavator ERG 1600 in operation.

Key words: *bucket wheel excavator, total excavation resistance, rotor bucket, tangential component of rock mass resistance, normal component of rock mass resistance, capacity, torque of rotor shaft*

INTRODUCTION

The process of excavation and transport of excavated mass using the bucket wheel excavators is continuous. Bucket wheel excavators excavate by alternate cuts that are removed during the rotation of rotor in the vertical plane and by turn in the horizontal plane of platform with arm and rotor that are situated on it.

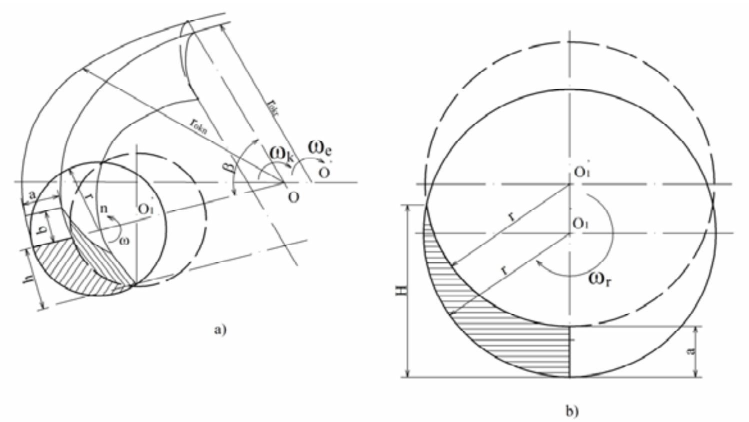


Figure 1. *Scheme of excavation using the bucket wheel excavator*
- a, vertical web
- b, horizontal web

---

* *Mining and Metallurgy Institute Bor*

Excavation is usually done by vertical or horizontal cuts with the fixed radius of excavation r (not considering the oscillations of the working body z) and at certain speeds the rotor turning w and arm with a turning platform w_k.

When working in vertical cuts after each shift of platform with arm for angle (β), whose size is determined by the size of excavation, the excavator with arm or just arm (if its length is variable) shifts for the size (a) equal to the maximum depth of cut.

When working with a horizontal cut, after each turn of the platform with arm for the angle (β), the arm with rotor is lowered for the size of (a).

Due to periodic entry and exit of buckets from the web of excavation, also the external load on the rotor in the process of excavation has a recurring character. Changing the external load is increased by additional changes in cutting forces that arise from non-similar mechanical properties of the massif which is excavated. Changes of external load are periodic and causing oscillations and additional dynamic loads on the arm and rotor, then the structural elements of the superstructure and base construction of rotor excavator. These additional dynamic loads can be very dangerous for the structural elements at the time of occurrence the resonant oscillations.

Figure 2 shows a series of oscillograms characterized by oscillations and dynamic loads that arise when working the conveyor stacker.

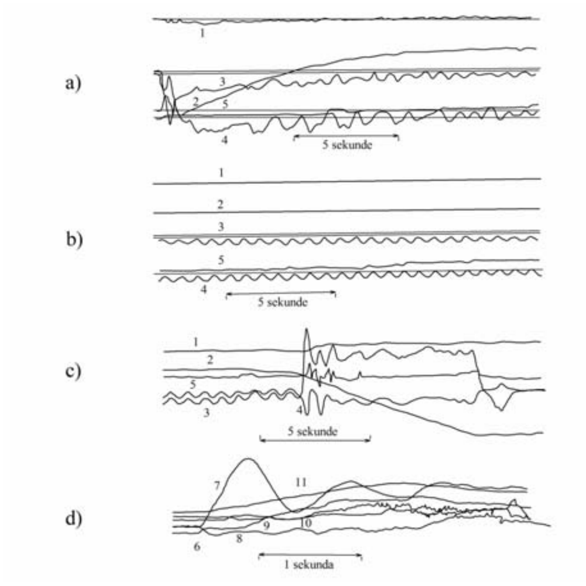


Figure 2. *Oscillograms characterized by dynamic loads at work of conveyor stacker OŠ 4500/180*

a) starting the conveyor into operation without load
b) setting the movement of conveyhor without load
c) no-load conveyor braking
d) load at different locations along the length f belt during starting the loaded conveyor (curves 6,7,8,9,10 and 11)
1 - stresses in the upper section of the shelf jib
2 - rotation speed of main engine of driving drum
3 and 4 - torque on the drive drum shaft
5 - walking stroller tension mechanism

1. EXTERNAL LOAD

The following types of external loads and overcoming resistance that are the most specific for machines are:

- resistance of rock mass in the excavation process
- loads that arise during working of conveyor.

1.1. Determination of external load during excavation without oscillations of rotor and change of cutting force

The total resistance of the rock mass excavation using the bucket wheel excavator is the sum of resistances which occur in some buckets that are in the web of rock mass, Figure 3. The total resistance to excavation on any bucket consists of three components.

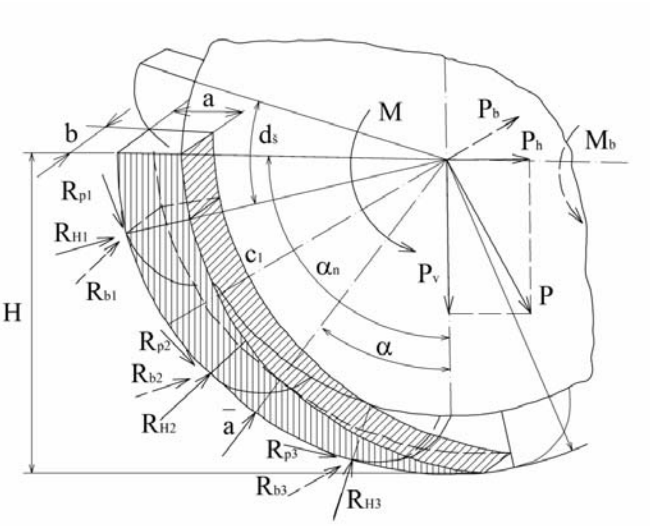


Figure 3. *The forces acting on the edge of the rotor*

$$R_i = \sqrt{R_{Ti}^2 + R_{Ni}^2 + R_{Bi}^2} \qquad (1)$$

where:

- R_{Ti}, tangential component of the rock mass resistance in the vertical plane
- R_{Ni}, normal component of the rock mass resistance in the vertical plane
- R_{Bi}, side component of the rock mass resistance

The basic component that characterizes the rock mass resistance of excavation is R_{Ti}, and is usually called the excavation resistance in the process of cutting and it can be determined for any position of the bucket in web.

$$R_{Ti} = L_{sr.} \cdot K_L \qquad (2)$$

$$R_{Ti} = F_{sr.} \cdot K_F \qquad (3)$$

where:

- L, length of cutting contour of bucket located in the web and it is equal to a volume half of intersection
- F, cross sectional cut
- K_L, resistance coefficient to excavation [KN/m’]

- K_F, resistance coefficient to excavation [KN/m^2]

In this case, the coefficients K_L and K_F are included not only to the excavation resistance, but the resistances to cutting, moving the excavated mass into bucket and resistance around the perimeter of the working wheel. Practically, the determination of medium size components R_{Ni} and R_{Bi} starts with the size of tangential component R_{Ti}.

$$R_{Ni} = \psi_n \cdot R_{Ti} \tag{4}$$

$$R_{Bi} = \psi_b \cdot R_{Ti} \tag{5}$$

where:

- ψ_n and ψ_b, coefficients obtained by experiment whose values generally depend on the physical-mechanical properties of excavated rock mass and structural characteristics of the working body, that is the cutting parameters a and b and lateral and circumferential velocities V_b and V_o, Figure 3.

In excavation of rock masses of the I and II category:

$\psi_n = 0{,}4 \div 0{,}5; \psi_b = 0{,}25 \div 0{,}35$, and in rock mass excavation of the IV and V category, their values are in the range:

$$\psi_n = 0{,}5 \div 0{,}8; \psi_b = 0{,}35 \div 0{,}50$$

Dependence R_{Ti} on the cutting angle α, Figure 3, or from time t at constant speed rotor, it is the best to use expressions (2) and (3) as experimental studies show that the average value of K_L remains practically constant across the entire length of the cutting. Since the coefficient K_L for different classes of different machines is different, and the mean value of K_F depends on the physical - mechanical properties of rock mass that is excavated and construction of working body. Therefore, the appropriate is use K_L in a function $R_{Ti} = f(a)$ to find its connection with the K_F.

Starting from Figure 3 and hatched cut and equality of consumption, it can be written:

$$K_L \cdot r\int_0^{\alpha_n} L \cdot d\alpha = K_F \cdot r\int_0^{-\alpha_n} F \cdot d\alpha \tag{6}$$

where:

- α_n, starting cutting angle

Expressing the current value of the depth of cut a through $a_{\max} = a \cdot \sin\alpha$, it is possible to write the current values F and L in the following forms

$$L = \frac{a \cdot \sin\alpha}{\sin\gamma} + b \approx a\sin\alpha + b \tag{7}$$

- γ, angle of the side cutting edges→ 90º

$$F = a \cdot b \cdot \sin\alpha \tag{8}$$

$$\begin{aligned} K_L \cdot r\int_0^{-\alpha_n} (a \cdot \sin\alpha + b)d\alpha = \\ = K_F \cdot r\int_0^{-\alpha_n} a \cdot b \cdot \sin\alpha \cdot d\alpha \end{aligned} \tag{9}$$

where:

- a and b, maximum cut depth and width

$$\begin{aligned} K_L \cdot r|a \cdot \cos\alpha + ba|_0^{\alpha_n} = \\ = K_F \cdot r|ab \cdot \cos\alpha|_0^{\alpha_n} \end{aligned}$$

$$\begin{aligned} K_L \cdot r[-a + a\cos\alpha + b\alpha n] = \\ = K_F \cdot r(-ab + ab\cos\alpha n) \end{aligned}$$

After arranging, the following is got:

$$K_L = K_F \frac{ab(1-\cos an)}{an \cdot b + a(1-\cos\alpha n)} \tag{10}$$

Assuming that the capacity of the excavator is the function parameters K_L and K_F and basic parameters of the rotor, the resistance coefficients to excavation K_L and K_F will be expressed in terms of capacity and basic parameters of the rotor

$$\frac{b}{a} = \nu = const. \text{ rotor coefficient} \tag{11}$$

Hourly capacity will be:

$$Q = 60 \cdot q \cdot nz \left[m^3/h\right] \tag{12}$$

where:

- q [m^3], volume of bucket,
- z, number of buckets on the rotor,
- n, number of rotor rotations per minute.

Taking also into account the coefficient of loosening k_r from the conditions of material change in the bucket, it is possible to write:

$$q = a \cdot b \cdot h = \frac{Q}{60 \cdot n \cdot z \cdot k_r} \quad (13)$$

where:

- h, total rotor height, and expressed over rotor radius r is:

$$h = r(1 - \cos \alpha n) \quad (14)$$

Including the expression (14) into (13), it is obtained:

$$ab = \frac{Q}{60 \cdot n \cdot z \cdot r(1 - \cos \alpha n) \cdot k_r} \quad (15)$$

$$b = \sqrt{\frac{Q}{60 \cdot n \cdot z \cdot k_r \cdot h}}$$

$$a = \frac{1}{\nu}\sqrt{\frac{Q}{60 \cdot n \cdot z \cdot k_r \cdot h}}$$

Including the expressions for a and b and (14) into expression (10), it is obtained:

$$K_L = \frac{K_F}{\alpha n + \frac{1}{\nu}(1 - \cos \alpha n)} \cdot \sqrt{\frac{Q(1 - \cos \alpha n)}{60\nu \cdot n \cdot z \cdot r \cdot k_r}} \quad (16)$$

For $\alpha n = \frac{\pi}{2}$, that is usually adopted in calculations:

$$K_L = \frac{2K_F}{\pi + \frac{2}{\nu}} \cdot \sqrt{\frac{Q}{60 \frac{2}{\pi} \cdot n \cdot z \cdot r \cdot k_r}} \quad (17)$$

Starting from the conditions of minimum specific consumption $\nu = \frac{2}{\pi}$, then it will be:

$$K_L = K_F \sqrt{\frac{Q}{120\pi \cdot n \cdot z \cdot r \cdot k_r}} \quad (18)$$

Without taking into account the oscillations of the rotor, the size of secondary torque on the rotor shaft of excavation resistance can be expressed as follows:

$$M_{sr.} = P_{sr.} \cdot r \quad (19)$$

where:

- $P_{sr.}$, middle tangential resistance of rock excavation

Based on the equality of consumption, it can be written:

$$R_{sr.} = \int_0^{-\alpha n} d\alpha = K_L \int_0^{-\alpha n} (a \cdot \sin \alpha + b) d\alpha \quad (20)$$

where:

- α_n, angle between buckets
- R_{sr}, mean value of tangential component of excavation resistance in the grip of a bucket

$$R_{sr.} = \frac{K_L[a(1 - \cos \alpha_n + b \cdot \alpha_n)]}{\alpha_n}$$

Based on the previous expression (10) it is obtained:

$$R_{sr.} = \frac{K_F \cdot a \cdot b(1 - \cos \alpha_n)}{\alpha_n}$$

this refers to a bucket. The total force will be equal to:

$P_{sr.} = R_{sr.} \cdot \frac{\alpha_n}{n} = R_{sr.} \cdot \frac{\alpha_n \cdot z}{2\pi}$, and the total average tangential excavation resistance will be:

$$P_{sr.} = \frac{K_F \cdot a \cdot b(1 - \cos \alpha_n) \cdot z}{2\pi} \quad (21)$$

Including the expressions ab from (15) into (21)

$$ab = \frac{Q}{60 \cdot nzrk_r (1 - \cos\alpha_n)}$$

$$P_{sr} = K_F \frac{Q}{120\pi \cdot n \cdot r \cdot k_r} \qquad (22)$$

$$M_{sr} = \frac{K_F \cdot Q}{120\pi \cdot n \cdot k_r} \qquad (23)$$

The mean value of power on the rotor shaft, engaged in rock mass excavation, is:

$$N_{sr.} = \frac{QK_F}{360 \cdot k_r}[kW] \qquad (24)$$

2. DEFINING THE CALCULATED LOADS ON THE ROTOR SHAFT CAUSED BY RESISTANCE IN THE EXCAVATION PROCESS

Starting from Figure 3, the external excavation resistance can be replaced by the joint loads acting on the rotor axis by the moment M and force P, acting on a joint force of the rotor plane P_B of the lateral loads in excavation and moment M_B,, which twists the arm.

It is useful to divide the force P on vertical and horizontal component (P_V and P_H), as P_V appears as a fundamental force that stimulates the structure in the vertical plane. In order to explain the dynamic effects on the structure elements, it is necessary to explain the nature of change the torque M on the rotor shaft, depending on the resistance in the process of excavation in a function of rotor turning angle α and turning time t.

According to the expressions (2), (3) and (7), a moment from tangential resistance to excavation at any bucket R_{Ti} regarding to the rotor axis O, Figure 3, could be written as

$$M_i = r \cdot K_L (a \cdot \sin\alpha + b) \qquad (25)$$

where:

- r, rotor radius from cutting edge of a bucket

Starting from the real assumption that in the bucket entry into the working process by vertical cut, the moment on the rotor shaft increases fast for the value:

$$M_1 = r \cdot K_L \cdot b \qquad (26)$$

and a bucket exit from the working process is followed by the reduction of the total torque on the rotor shaft, which is derived from the excavation resistance value:

$$M_2 = r \cdot K_L \cdot (a \cdot \sin\alpha_n + b) \qquad (27)$$

Without taking into account the change in excavation resistance at the expense of breaking off, change the physical-mechanical properties along the length of cut and other factors causing the change of resistance excavation force, assuming that during the period of time from entering the bucket in operation until its release, the torque on the rotor shaft M_i is changed, then the diagram of total torque on the rotor shaft $M_{(\alpha)}$ of resistance to excavation in all buckets can be expressed respectively, Figure 4.

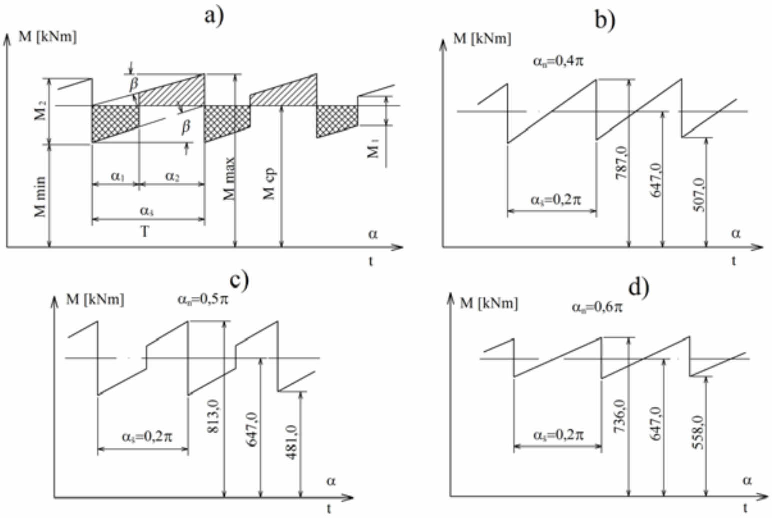


Figure 4. *Change the calculation of torque resistance to excavation on the rotor shaft*
a) general case
b) bucket wheel excavator ERG-1600 at suitable $\alpha_n = 0{,}4\pi$
c) $\alpha_n = 0{,}5\pi$
d) $\alpha_n = 0{,}6\pi$

According to the expression (23), the mean value is determined of the moment M_{sr} of the excavation resistance. It is necessary to determine the values of jumps M_1 and M_2 for the angles α_1 and α_2 corresponding to the rotor work with small and large number of buckets in the paper. Size of the angles α_1 and α_2 are determined from Figure 3.

$$\alpha_1 = (m+1)\alpha_š - \alpha_n \quad (28)$$

$$\alpha_2 = \alpha_n m \alpha_š \quad (29)$$

where:

- $\alpha_š$, angular step of a bucket
- $m = \dfrac{\alpha_n}{\alpha_š}$
- $\dfrac{\alpha_1}{\alpha_š} = (m+1) - j$ and $\dfrac{\alpha_2}{\alpha š} = j - m$, i.e.

excess in (j) of an integer (m) will be part α_2 in $\alpha_š$, and a missing part to the next whole number (m+1) will be a part $\alpha_1$in $\alpha_š$.

In the actual structures of the rotor with z = 6 to 14 buckets and corners α_n=(0,4 do 0,6) π, (j) is located within the 1.2 to 4.2, the corresponding number, at the same time in a conjunction with the rock mass, will be from 1-2 and 4-5. In addition, the majority of buckets (m+1) will participate in the excavation for the duration of the angle α_2, and smaller number of buckets (m) will be changed during the angle α_1.

Size of angle β, Figure 4a, required for the structure $M_{(\alpha)}$, is determined as $tg\beta = \dfrac{M_2 - M_1}{\alpha š}$. Sizes M_{max} and M_{min} are determined in general form starting from the conditions of equality of hatched surfaces from the top to the bottom of $M_{sr.}$

$$M_{\max} = M_{sr.} + \frac{M_2}{2} - M_1[j-(m+0{,}5)] \quad (30)$$

$$M_{\min} = M_{sr.} - \frac{M_2}{2} - M_1[j-(m+0{,}5)] \quad (31)$$

At the whole number of buckets which are also found in a conjunction with the rock mass, i.e. when $j = \frac{\alpha_n}{\alpha_š} = m$, will be:

$$M_{\max} = M_{sr.} + \frac{M_2 - M_1}{2} \qquad (32)$$

$$M_{\min} = M_{sr.} - \frac{M_2 - M_1}{2} \qquad (33)$$

It is emphasized that at a constant speed rotor, constructed diagram $M = f(\alpha)$ also presents a simultaneous diagram of moment changes in a function of time $M = f(t)$.

For example, we will consider the work of excavator ERG 1600 with capacity Q = 3750 [m^3/h] and rpm of rotor n = 3.7 [min^{-1}] in the rock mass, K_F = 0.3 [MPa] and with the coefficient of loosening k_r = 1.25.

At given conditions, $M = f(\alpha)$ and $M = f(t)$ were presented in tables and diagrams.

Using the expressions for a and b and expressions (16) and (23), the sizes K_L, a, b and $M_{sr.}$ were calculated, and then the sizes of torque jumps M_1 and M_2 for $\alpha_n = 0{,}4\pi$, $\alpha_n = 0{,}5\pi$ and $\alpha_n = 0{,}6\pi$, i.e. when the mean number of buckets is in a conjunction with the rock mass, and that is equal to 2; 2.5 and 3.

The obtained data are present in Table 1 and on diagrams, Figure 4.

Table 1.

α_n [rad.]	$M_{sr.}$ [kNm]	a [m]	b [m]	K_L [kN/m]	M_1 [kNm]	M_2 [kNm]
0.4π	647.0	0.796	0.436	65.5	161.0	444.0
0.5π	647.0	0.614	0.389	58.5	129.0	332.5
0.6π	647.0	0.518	0.356	53.5	107.4	256.0

REFERENCES

[1] R. Popović et al. Elaborate on the Field Investigations the Influential Parameters on the Operation Mode of Excavators schRS 315/1.5-12.5 in the Conditions of Excavation of Coal Layer at the Open pit “Gračanica“ Gacko, Institute of Mining Research, Tuzla, 1983.

[2] R. Popović, D. Djukić, Determination the Effects of Operational Parmeters of the Bucket Wheel Excavator at its Effective Capacity in Coal V Yugoslav Symposium on Open Pit Mining of Minerals, Skoplje, 1983.

[3] M. Ljubojev, R. Popović, Fundamentals of Rock Destruction by the Used Mechanization in the Exploitation of Solid Mineral Resources (books in print) Mining and Metallurgy Institute, Bor

[4] M. Ljubojev, R. Popović, Problems in Exploitation the Rock Material International Scientific Symposium on the Occasion of 120 Anniversary of Kreka, Tuzla, 2005.

[5] R. Popović, M. Ljubojev, M. Ivković. Mechanisation and Productivity Precondition to the Changes of Underground Mines, International Scientific Symposium on the Occasion of 120 Anniversary of Kreka, Tuzla, 2005.

[6] R. Popović, M. Ljubojev, Basis of rock destruction by aplied mechanization in exploitation of solid mineral raw materials Monograph, Bor, 2011.

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:551.49:622.26(045)=861

Dragan Ignjatović, Milenko Ljubojev*, Lidija Đurđevac Ignjatović*, Jelena Petrović**

KLASIFIKACIJA STENSKOG MASIVA PRE IZGRADNJE TUNELA (PO WICKHAM-U I BIENAWSKOM)**

Izvod

Pre bilo kakve aktivnosti na izradi tunela, neophodno je definisati stanje stene, kroz koju će biti konstruisan novi tunel.

Šema klasifikacije stenskih masiva je razvijena pre više od 100 godina, od kako je Riter (1879) pokušao da formalizuje empirijski pristup projektovanju tunela, naročito za definisanje podgrade. Dok su klasifikacione šeme prikladne za originalnu primenu, treba obratiti pažnju na njihovu primenu u klasifikaciji stenskog masiva za druge inženjerske problem.

Ključne reči: *klasifikacija stenskog masiva, izgradnja tunela*

1. UVOD

Tokom faze projekta o izvodljivosti i idejnog rešenja, kada je vrlo malo detaljnih informacija dostupno za stenske mase o njihovom stanju napona i hidrološkim karakteristikama, korišćenje šema klasifikacije stenske mase mogu biti od velike koristi. Najjednostavnije rečeno, one se mogu koristiti kao liste za proveru kako bi se osiguralo da su sve relevantne informacije uzete u obzir. Sa druge strane, jedna ili više klasifikacionih šema mogu da se koriste da se izgradi slika o sastavu i karakteristikama stenske mase da bi se obezbedile početne procene za podgradu, i da se obezbedi procena čvrstoće i deformacijskih svojstava stenske mase.

Veoma je bitno da se razumeju ograničenja šema za klasifikaciju stenske mase (Palmstrom i Broch, 2006) i da njihova primena ne treba (i ne može) da zameni neke procedure projektovanja. Ipak, korišćenje ovih procedura za projektovanje zahtevaju relativno detaljne informacije o naponu in situ, karakteristike stenske mase i planirani način otkopavanja, a sve ove informacije nisu dostupne u ranoj fazi projektovanja. Čim ova informacija bude dostupna, šeme klasifikacija treba dopuniti i koristiti u sprezi sa specifičnim terenskim istraživanjima.

* *Institut za rudarsvo i metalurgiju*
** *Ovaj rad je proistekao iz Projekta br. 33021 koji se finansira sredstvima Ministarstva za prosvetu i nauku Republike Srbije*

2. OCENA STENSKE STRUKTURE (RSR)

Vikam i saradnici (1972) su opisali kvantitativnu metodu za opisivanje kvaliteta stense mase i za odabir odgovarajuće podgrade na osnovu njihove klasifikacije o proceni stenske strukture (RSR). Većina istorijskih slučajeva, koji su korišćeni za razvoj ovog sistema, su bili za relativno male podgrade tunela čeličnim setovima, mada istorijski, ovaj sistem je bio prethodnica betonskim podgradama. Uprkos ovim ograničenjima, vredno je ispitati RSR sistem u nekim detaljima obzirom da ukazuje na logiku uključenu u razvoj kvazi-kvantitativne klasifikacije stenskog masiva.

Značaj RSR sistema, u kontekstu diskusije, je da ona predstavlja koncept za ocenu svake od komponenti koje su navedene naniže, da bi se dobila vrednosti *RSR = A + B + C*.

1. *Parametar A, Geologija:* Generalna ocena geološke strukture na osnovu:
   a. Porekla stene (vulkanske, metamorfne i sedimentne).
   b. Čvrstoća stene (čvrste, srednje i raspadnute).
   c. Geološke strukture (masivna, blago poremećene/naborane, umereno poremećene/naborane, intenzivno poremećene/naborane).

Tabela 1. *Ocena geološke strukture: Parametar A: Generalna oblast geologije*

	Osnovni tip stene				Geološka struktura			
	Čvrsta	**Srednja**	**Meka**	**Raspadnuta**	Masivna	Blago poremećene ili nabrane	Umereno poremećene ili nabrane	Intenzivno poremećene ili nabrane
Vulkanska	1	2	3	4				
Metamorfna	1	2	3	4				
Sedimentna	2	3	4	4				
Tip 1					30	22	15	9
Tip 2					27	20	13	8
Tip 3					24	18	12	7
Tip 4					19	15	10	6

2. *Parametar B, Geometrija*: Uticaj diskon tinuiteta u odnosu na pravac pružanja tunela na osnovu:
   a. Razmaka spoja.
   b. Orijentacija spoja (pravac i pad).
   c. Pravac linije tunela.
3. *Parametar C*: Uticaj priliva podzemnih voda i uslova spoja na osnovu:
   a. Sveukupni kvalitet na osnovu kombinacije parametara A i B.
   b. Uslovi spoja (dobri, srednji i loši).
   c. Količina priliva vode (u galonima po minuti po 1000 stopa tunela).

Tabela 2. *Ocena stenske strukture: Parametar B: Raspored spojeva, pružanje i pravac*

Prosečno rastojanje između spojeva	Pravac ⊥ na osu					Pravac ‖ na osu		
	Pravac pružanja					Pravac pružanja		
	Oba	U pravcu pružanja		Suprotno na pravac pružanja		U bilo kom pravcu		
	Pad istaknutih spojeva					Pad istaknutih spojeva		
	horizon-talno	po padu	verti-kalno	po padu	verti-kalno	horizontalno	po padu	verti-kalno
1. Veoma bliski spoj, <2 in	9	11	13	10	12	9	9	7
2. Bliski spoj, 2-6 in	13	16	19	15	17	14	14	11
3. Umereni spoj, 6-12 in	23	24	28	19	22	23	23	19
4. Umereno do kockasto, 1-2 ft	30	32	36	25	28	30	28	24
5. Kockasto do masivno, 2-4 ft	36	38	40	33	35	36	24	28
6. Masivno, >4 ft	40	43	45	37	40	40	38	34

Tabela 3. *Ocena stenske strukture: Parametar C: Podzemna voda, uslovi spoja*

Očekivani priliv vode gpm/1000 ft tunela	Zbir parametara A+B					
	13-44			45-75		
	Uslovi spoja					
	Dobri	Solidni	Loši	Dobri	Solidni	Loši
Bez priliva	22	18	12	25	22	18
Slabi priliv, <200 gpm	19	15	9	23	19	14
Umereni priliv, 200-1000 gpm	15	22	7	21	16	12
Veliki priliv, >1000 gpm	10	8	6	18	14	10

Na primer, čvrsta metamorfna stena koja je blago poremećena ili naborana ima ocenu A=22 (iz tabele 1). Stenska masa je umereno sastavljena, sa spojevima koji su upravni na osu tunela koji se pruža pravcem istok-zapad, i padom između 20° i 50°.

Tabela 2 daje ocenu za B=24 za pravac sa padom (definisan u nastavku). Vrednost za A+B=46 i to znači da, za spojeve u regularnim uslovima (blago degradirani i izmenjeni) i umerenim prilivom vode između 200 i 1000 galona u minuti, tabela 3, daje ocenu za C=16. Otuda je konačna vrednost za ocenu structure stene RSR=A+B+C=62.

3. GEOMEHANIČKA KLASIFIKACIJA

Bienawski (1976) je objavio detalje o klasifikaciji stenske mase, nazvana Geomehanička klasifikacija sistema ocene stenske mase (*RMR*). Tokom godina, ovaj sistem je sukcesivno obnavljan sa većim brojem rezultata ispitivanja i Bienawski je napravio značajne promene u oceni dodeljene različitim parametrima. Sledećih šest parametara s koriste za klasifikaciju stenske mase korišćenjem sistema *RMR*:

1. Jednoosna pritisna čvrstoća stena.
2. Oznaka kvaliteta stene (*RQD*).
3. Razmak između diskontinuiteta.
4. Stanje diskontinuiteta.
5. Stanje podzemnih voda.
6. Orientacija diskontinuiteta.

U primeni ovog sistema klasifikacije, stenska masa je podeljena na strukturne oblast i svaka oblast se klasifikuje posebno. Granice strukturnih oblasti su obično podudarne sa svojstvom, kao što je rased ili promena vrste stena. U nekim slučajevima, značajne promene u diskontinuitetu ili karakteristikama, unutar iste stene, mogu da zahtevaju podelu stenskog masiva na veći broj manjih strukturnih oblasti.

Sistem klasifikacije stenske mase je dat u tabeli 4, i on daje ocene za svaki od šest parametara koji su navedeni. Ove ocene su sabrane da bi dale vrednost RMR. Sledeći primer ilustruje korišćenje ovih tabela da bi se došlo do vrednosti za RMR.

Tunel bi trebalo da prođe kroz blago oštećeni granit sa dominantnim spojem čiji je pad pod uglom od 60° u odnosu na pravac pružanja tunela. Indeks testiranja i jezgrovanje dijamantskom bušilicom daje vrednosti indeksa za Point-load test od 8 MPa i prosečnu RQD vrednost od 70%.

Neznatno grubi i degradirani spojevi sa razmakom < 1 mm, su razmaknuti na 300 mm. Uslovi pri izgradnji tunela su predviđeni kao vlažni.

Tabela 4. *Sistem za ocenu stenskog masiva (po Bieniawskom 1989)*

<table>
<tr><td colspan="10">A. Klasifikacioni parametri i njihova ocena</td></tr>
<tr><td colspan="3">Parametar</td><td colspan="7">Opseg vrednosti</td></tr>
<tr><td rowspan="3">1.</td><td rowspan="2">Čvrstoća neporemećenog stenskog materijala</td><td>Indeks čvrstoće Point-load testa</td><td>>10 MPa</td><td>4-10 MPa</td><td>2-4 MPa</td><td>1-2 MPa</td><td colspan="3">Za ovaj mali opseg, poželjno je uraditi test čvrstoće na pritisak</td></tr>
<tr><td>Čvrstoća na pritisak</td><td>>250 MPa</td><td>100-250 MPa</td><td>50-100 MPa</td><td>25-50 MPa</td><td>5-25 MPa</td><td>1-5 MPa</td><td><1MPa</td></tr>
<tr><td colspan="2">Ocena</td><td>15</td><td>12</td><td>7</td><td>4</td><td>2</td><td>1</td><td>0</td></tr>
<tr><td rowspan="2">2.</td><td colspan="2">Kvalitet izbušenog jezgra RQD</td><td>90%-100%</td><td>75%-90%</td><td>50%-75%</td><td>25%-50%</td><td colspan="3"><25%</td></tr>
<tr><td colspan="2">Ocena</td><td>20</td><td>17</td><td>13</td><td>8</td><td colspan="3">3</td></tr>
<tr><td rowspan="2">3.</td><td colspan="2">Razmak diskontinuiteta</td><td>>2 m</td><td>0.6-2 m</td><td>200-600 mm</td><td>60-200 mm</td><td colspan="3"><60 mm</td></tr>
<tr><td colspan="2">Ocena</td><td>20</td><td>15</td><td>10</td><td>8</td><td colspan="3">5</td></tr>
<tr><td rowspan="2">4.</td><td colspan="2">Stanje diskontinuiteta</td><td>Veoma gruba površina Nenastavljeno Bez odvajanja Zidovi stena na koje nisu uticale atmosferilije</td><td>Blago gruba površina Odvojenost <1mm Zidovi stena na koje su blago uticale atmosferilije</td><td>Blago gruba površina Odvojenost <1mm Zidovi stena na koje su značajno uticale atmosferilije</td><td>Klizna površina ili glina čija je debljina <5mm ili kontinualno odvajanje od 1-5mm</td><td colspan="3">Meka glina debljine >5mm ili kontinualno odvajanje >5mm</td></tr>
<tr><td colspan="2">Ocena</td><td>30</td><td>25</td><td>20</td><td>10</td><td colspan="3">0</td></tr>
<tr><td rowspan="4">5.</td><td rowspan="3">Podzemna voda</td><td>Priliv na 10m dužine tunela (l/m)</td><td>Nema</td><td><10</td><td>10-25</td><td>25-125</td><td colspan="3">>125</td></tr>
<tr><td>Pritisak vode na spoju/glavni napon σ</td><td>0</td><td><0.1</td><td>0.1-0.2</td><td>0.2-0.5</td><td colspan="3">>0.5</td></tr>
<tr><td>Opšti uslovi</td><td>Potpuno suvo</td><td>vlažno</td><td>mokro</td><td>kaplje</td><td colspan="3">teče</td></tr>
<tr><td colspan="2">Ocena</td><td>15</td><td>10</td><td>7</td><td>4</td><td colspan="3">0</td></tr>
</table>

Vrednost RMR za dati primer je 59.

4. ZAKLJUČAK

Klasifikacija stenske mase je veoma važan korak pri konstruisanju tunela ili pri bilo kom drugom, sličnom poslu, kao što su rudarski radovi (miniranje, definisanje stabilnosti kosina na površinskim kopovima, definisanje sile kopanja itd.), u građevinarstvu i sl. Ocena stenskog masiva po Bienawskom (RMR) je bio baziran na slučajevima uzetim iz istorije građevinarstva.

LITERATURA

[1] N. R. Barton, R. Lien, J. Lunde, Engineering classification of rock masses for the design of tunnel support, Rock Mechanic 6(4), 189-239

[2] M. Ljubojev, R. Popovic, Basis of geomechanic, Copper Institute Bor, Indok center 2006, pages 1-533, ISBN 86-7827-020-9

[3] M. Ljubojev, R. Popovic, D. Ignjatovic, Tunnel analysis in fault zones and the effects of stress distribution on the support, Journal of Mining and Metallurgy, Section A: Mining 2009, Vol. 45, 2009, pages 49-57,ISBN 1450-5959

[4] M. Ljubojev, M. Avdic, D. Ignjatovic, L. Đ. Ignjatovic, Influence from flotation tailings, field 2, on Krivelj River tunnel stability, Mining works Journal, 2/2009, pages 21-28

[5] M. Ljubojev, D. Ignjatovic, V. Ljubojev, L. Đ. Ignjatović, D. Rakić, Deformabilnost i nosivost nasutog materijala u neposrednoj blizini otvora okna na P. K. „Zagrađe“ – Kop – 2, Rudarski radovi, br. 2/2010, str. 107-114

UDK: 551.49:622.26(045)=20

Dragan Ignjatović, Milenko Ljubojev*, Lidija Đurđevac Ignjatović*, Jelena Petrović**

ROCK MASS CLASSIFICATION BEFORE THE TUNNEL CONSTRUCTION (PER WICKHAM AND BIENAWSKI)**

Abstract

Before any activity in the tunnel construction, it is necessary to determine the rock condition, through which will new tunnel be constructed.

Rock mass classification scheme was developed for over than 100 years since Ritter (1879) attempted to formalize an empirical approach to tunnel design, in particular for determining support requirements. While the classification schemes are appropriate for their original application, a care has to be paid on their use the classification of rock mass or other engineering problems.

Key words: *rock mass classification, tunnel construction*

1. INTRODUCTION

During the feasibility and preliminary design stages of a project, when very little detailed information are available on the rock mass and its stress and hydrological characteristics, the use of rock mass classification scheme can be of considerable benefit. Simply, this may involve using the classification scheme as a check-list to ensure that all relevant information was considered. On the other side, one or more rock mass classification schemes can be used to make a view of composition and characteristics the rock mass to provide initial evaluation of support requirements, and to provide evaluations of t strength and deformation properties of the rock mass.

It is important to understand the limitations of rock mass classification schemes (Palmstrom and Broch, 2006) and that their use does not (and cannot) replace some design procedures. However, the use of these design procedures require relatively detailed information on in situ stresses, rock mass properties and planned excavation sequence, none of which may be available at an early stage in the project. As this information becomes available, the use of the rock mass classification schemes should be updated and used in conjunction with the specific s.

2. ROCK STRUCTURE RATING (RSR)

Wickham et al (1972) described a quantitative method for description the quality of rock mass and selection the appropriate support based on their Rock Structure Rating (*RSR*) classification. Most of the historical

* *Mining and Metallurgy Institute Bor*

** *This paper is produced from the project no. 33021 which is funded by means of the Ministry of Education and Science of the Republic of Serbia*

cases, used in this system development, were for relatively small tunnels supported by means of steel sets, although historically this system was previously before concrete supports. In spite of these limitations, it is worth to test the *RSR* system in some details since it demonstrates the logic involved in development the quasi-quantitative rock mass classification system.

The significance of the *RSR* system, in the context of this discussion, is that it presents a rating concept of each components listed below as the numerical value of $RSR = A + B + C$ would be obtained.

1. Parameter A, Geology: General rating of geological structure, based on:

a. Rock type origin (igneous, metamorphic, and sedimentary)

b. Rock hardness (hard, medium, soft and decomposed)

Table 1. *Rock structure rating: Parameter A: General area geology*

	Basic Rock Type				Geological Structure			
	Hard	Medium	Soft	Decomposited	Massive	Slightly Folded or Faulted	Moderately Folded or Faulted	Intensively Folded or Faulted
Igneous	1	2	3	4				
Metamorphic	1	2	3	4				
Sedimentary	2	3	4	4				
Type 1					30	22	15	9
Type 2					27	20	13	8
Type 3					24	18	12	7
Type 4					19	15	10	6

c. Geologic structure (massive, slightly faulted/folded, moderately faulted/folded, intensively faulted/folded).

2. Parameter B, Geometry: Effect of discontinuity pattern regarding to the direction of the tunnel drive, based on:

a. Joint spacing

b. Joint orientation (strike and dip)

c. Direction of tunnel drive

3. Parameter C: Effect of groundwater inflow and joint conditions, based on:

a. Overall rock mass quality based on combination of A and B parameters

b. Joint condition (good, fair, poor)

c. Amount of water inflow (in gallons per minute per 1000 feet of tunnel)

Table 2. *Rock structure rating: Parameter B: Joint pattern, direction of drive*

Average joint spacing	Strike ⊥ to Axis					Strike ‖ to Axis		
	Direction of drive					Direction of drive		
	Both	With dip		Against dip		Either direction		
	Dip of Prominent Joints					Dip of Prominent Joints		
	Flat	Dipping	Vertical	Dipping	Vertical	Flat	Dipping	Vertical
1. Very closely jointed, <2in	9	11	13	10	12	9	9	7
2. Closely jointed, 2-6 in	13	16	19	15	17	14	14	11
3. Moderately jointed, 6-12 in	23	24	28	19	22	23	23	19
4. Moderate to blocky, 1-2 ft	30	32	36	25	28	30	28	24
5. Blocky to massive, 2-4 ft	36	38	40	33	35	36	24	28
6. Massive, >4 ft	40	43	45	37	40	40	38	34

Table 3. *Rock structure rating: Parameter C: Groundwater, joint condition*

Anticipated water inflow gpm/1000 ft of tunnel	Sum of Parameters A+B					
	13-44			45-75		
	Joint condition					
	Good	Fair	Poor	Good	Fair	Poor
None	22	18	12	25	22	18
Slight, <200 gpm	19	15	9	23	19	14
Moderate, 200-1000 gpm	15	22	7	21	16	12
Heavy, >1000 gpm	10	8	6	18	14	10

For example, hard metamorphic rock which is slightly folded or faulted has a rating of A=22 (from Table 1). The rock mass is moderately joined, with joints striking perpendicular to the tunnel axis which is driven east-west, and dipping between 20˚ and 50˚.

Table 2 gives the rating for B=24 for driving with dip (defined below). The value of A+B=46 and this means that, for joints of fair condition (slightly weathered and altered) and a moderate water inflow between 200 and 1000 gallons per minute. Table 3 gives the rating for C=16. Hence, the final value of the rock structure rating RSR=A+B+C=62.

3. GEOMECHANICS CLASSIFICATION

Bieniawski (1976) published the details of a rock mass classification called the Geomechanics Classification or the Rock Mass Rating (*RMR*) system. Over the years, this system has been successively refined as more case records have been examined and Bieniawski made significant changes in the ratings assigned to different parameters. The following six parameters are used to classify a rock mass using the *RMR* system:

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

In applying this classification system, the rock mass is divided into a number of structural regions and each region is classified separately. The boundaries of the structural regions are usually matched with the major structural feature such as fault or with change in the rock type. In some cases, significant changes in discontinuity spacing or characteristics, within the same rock type, may require division of the rock mass into a number of small structural regions.

The Rock Mass Rating system is presented in Table 4, giving the ratings for each of the six parameters listed above. These ratings are summed to give the *RMR* value. The following example illustrates the use of these tables to obtain the *RMR* value.

A tunnel has to be driven through slightly weathered granite with a dominant joint set dipping at 60^o towards the drive direction. Testing index and logging of diamond drilled core give a typical Point-load strength index value of 8 MPa and average *RQD* value of 70%. The slightly rough and slightly weathered joints with a separation of < 1 mm, are spaced at 300 mm. Tunneling conditions are anticipated to be wet.

Table 4: *Rock Mass Rating System (After Bieniawski 1989)*

<table>
<tr><td colspan="10">A. Classification parameters and their ratings</td></tr>
<tr><td colspan="3">Parameter</td><td colspan="7">Range of values</td></tr>
<tr><td rowspan="3">1.</td><td rowspan="2">Strength of intact rock material</td><td>Point-load strength index</td><td>>10 MPa</td><td>4-10 MPa</td><td>2-4 MPa</td><td>1-2 MPa</td><td colspan="3">For this low range uniaxial compressive test is prefered</td></tr>
<tr><td>Uniaxial comp. strength</td><td>>250 MPa</td><td>100-250 MPa</td><td>50-100 MPa</td><td>25-50 MPa</td><td>5-25 MPa</td><td>1-5 MPa</td><td><1MPa</td></tr>
<tr><td colspan="2">Range</td><td>15</td><td>12</td><td>7</td><td>4</td><td>2</td><td>1</td><td>0</td></tr>
<tr><td rowspan="2">2.</td><td colspan="2">Drill core Quality RQD</td><td>90%-100%</td><td>75%-90%</td><td>50%-75%</td><td>25%-50%</td><td colspan="3"><25%</td></tr>
<tr><td colspan="2">Rating</td><td>20</td><td>17</td><td>13</td><td>8</td><td colspan="3">3</td></tr>
<tr><td rowspan="2">3.</td><td colspan="2">Spacing of discontinuities</td><td>>2m</td><td>0.6-2m</td><td>200-600mm</td><td>60-200mm</td><td colspan="3"><60mm</td></tr>
<tr><td colspan="2">Rating</td><td>20</td><td>15</td><td>10</td><td>8</td><td colspan="3">5</td></tr>
<tr><td rowspan="2">4.</td><td colspan="2">Condition of discontinuities</td><td>Very rough surfaces Not continuous No separation Unweathered wall rock</td><td>Slightly rough surface Separation <1mm Slightly weathered walls</td><td>Slightly rough surface Separation <1mm Highly weathered walls</td><td>Slickensided surfaces or Gouge <5mm thick or Separation 1-5mm Continuous</td><td colspan="3">Soft gouge >5mm thick or Separation >5mm continuous</td></tr>
<tr><td colspan="2">Rating</td><td>30</td><td>25</td><td>20</td><td>10</td><td colspan="3">0</td></tr>
<tr><td rowspan="4">5.</td><td rowspan="3">Underground water</td><td>Inflow per 10m tunnel length (l/m)</td><td>None</td><td><10</td><td>10-25</td><td>25-125</td><td colspan="3">>125</td></tr>
<tr><td>Joint water press/Major principal σ</td><td>0</td><td><0.1</td><td>0.1-0.2</td><td>0.2-0.5</td><td colspan="3">>0.5</td></tr>
<tr><td>General conditions</td><td>Completely dry</td><td>Damp</td><td>Wet</td><td>Dripping</td><td colspan="3">Flowing</td></tr>
<tr><td colspan="2">Rating</td><td>15</td><td>10</td><td>7</td><td>4</td><td colspan="3">0</td></tr>
</table>

The *RMR* value for the given example is present as follows:

4. CONCLUSION

Rock mass classification is very important step in the tunneling construction or any other similar works, such as mining works (blasting, defining slope stability in the open pits, defining of excavation force, etc), building construction etc. Rock Mass Rating by Bienawski (RMR) was originally based upon case histories drawn from the civil engineering.

This paper indicates that before any activity in the construction, it is necessary to define the Rock mass classification as the base for further investigations.

REFERENCES

[1] N. R. Barton, R. Lien, J. Lunde, Engineering classification of rock masses for the design of tunnel support, Rock Mech. 6(4), 189-239

[2] M. Ljubojev, R. Popovic, Basis of geomechanic, Copper Institute Bor, Indok center 2006, pages 1-533, ISBN 86-7827-020-9

[3] M. Ljubojev, R. Popovic, D. Ignjatovic, Tunnel analysis in fault zones and the effects of stress distribution on the support, Journal of Mining and Metallurgy, Section A: Mining 2009, Vol. 45, 2009, pages 49-57,ISBN 1450-5959

[4] M. Ljubojev, M. Avdic, D. Ignjatovic, L. Đ. Ignjatovic, Influence from flotation tailings, field 2, on Krivelj River tunnel stability, Mining works Journal, 2/2009, pages 21-28

[5] M. Ljubojev, D. Ignjatovic, V. Ljubojev, L. Đ. Ignjatović, D. Rakić, Deformation and bearing capacity of burried material near the shaft opening at the open pit mine „Zagrađe“ – Open Pit – 2, Mining Engineering, No. 2-2010, pp. 115-122

INSTITUT ZA RUDARSTVO I METALURGIJU BOR — YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA — UDK: 622

UDK:551.49:622.26(045)=861

Milenko Ljubojev, Dragan Ignjatović*, Lidija Đurđevac Ignjatović**

PREDLOG POPREČNOG PRESEKA TUNELA KRIVELJSKE REKE**

Izvod

Novi tunel Kriveljske reke predstavlja kapitalni objekat od velikog ekološkog značaja kako za Bor i okolinu, tako i za deo crnomorskog sliva (reka Dunav). Tunel Kriveljske reke će biti izrađen u sredini koja je veoma nepovoljna za izradu i postojanost prostorije, kako tokom izrade, tako i za period korišćenja. Zato je potrebno detaljno uraditi ispitivanje stanja stenskog masiva i na osnovu tih podataka odrediti koji tip poprečnog preseka treba uzeti kao optimalno rešenje.

***Ključne reči**: tunel Kriveljske reke, poprečni presek tunela*

1. UVOD

Potrebu za izradom novog tunela Kriveljske reke, uslovilo je loše stanje postojećeg kolektora Kriveljske reke. Njegova izrada bila bi delom kroz flotacijsko jalovište, a delom kroz stenski masiv. Novim tunelom, regulisao bi se tok Kriveljske reke, iz razloga što dasadašnji tok može biti prekinut rušenjem starog kolektora Kriveljske reke.

Po lokaciji trase tunela biće izvedena terenska istražna bušenja sa ciljem determinisanja stenskog materijala.

Sledeći istražni radovi će biti izvedeni:

- istražno bušenje sa jezgrovanjem
- detaljno inženjersko-geološko kartiranje izvođenog jezgra
- odabir reprezentativnih uzoraka svakog izdvojenog litološkog člana
- određivanje ispucalosti stenske mase i izdeljenosti dobijenog jezgra (RQD).

---

* *Institut za rudarsvo i metalurgiju*
** *Ovaj rad je proistekao iz Projekta br. 33021 koji je finansiran sredstvima Ministarstva za prosvetu i nauku Republike Srbije*

Sl. 1. *Satelitski snimak flotacijskih jalovišta*

2. IZRADA NOVOG TUNELA KRIVELJSKE REKE

Predložena trasa novog tunela Kriveljske reke bila bi u dužini od oko 2530 m. Tunel bi bio izrađen od kote K+246 (na spoju sa Kriveljskom rekom van flotacijskog jalovišta), do K+272, po usponu od 1% (do spoja sa starim tunelom).

Izrada tunela bi bila po usponu, iz razloga odvodnjavanja otkopnih radova. i bi bila podeljena u dve deonice:

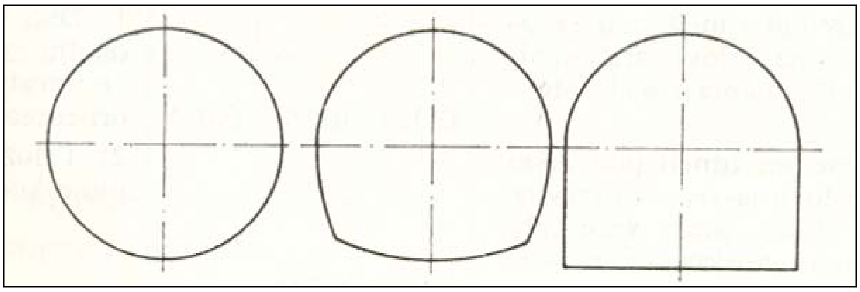
- I DEONICA- bila bi izrada tunela od njegovog budućeg izlaza, pored Kriveljske reke, po usponu od 1 %, do K+267, odnosno u dužini od oko 2000 m (slika br.1). Izrada tunela je kroz slabu i veoma slabu stensku sredinu, sa stalnim prilivom vode.
- II DEONICA- bila bi izrada tunela od kraja I deonice do spoja sa starim tunelom. Dužina ove deonice bila bi oko 500 m, odnosno, od K+267, do K+272. Izrada ove deonice bila bi kroz flotacijsku jalovinu Polja 2, a delom po aluvijonu nekadašnjeg korita Kriveljske reke.

3. OBLIK POPREČNOG PRESEKA HIDROTEHNIČKIH TUNELA

Oblik poprečnog preseka hidrotehničkih tunela zavisi od više činilaca, među kojima se ističu:

- statički uslovi rada obloge tunela
- hidraulički uslovi rada tunela i
- uslovi pod mojima će se tunel izrađivati.

Od obloge hidrotehničkih tunela zahteva se da bude tako dimenzionisana da po zapremini bude najmanja a istovremeno i da bez deformacija izdržava unutrašnje i spoljašnje pritiske. Osim toga obloga hidrotehničkih tunela mora biti i vodonepropusna.



Sl. 2. *Oblici slobodnih preseka hidrotehničkih tunela*

Prema uslovima hidrauličkog rada oblik poprečnog preseka treba da obezbedi najveći protok kod određenog pada i najmanjeg poprečnog preseka. Ovim uslovima najbolje udovoljavaju kružni, visokozasvođeni i potkovičasti oblici (slika br.2).

Vodeći računa o svim uticajnim činiocima, na izbor oblika poprečnog preseka hidrotehničkih tunela ipak preovlađuje uslov vezan za statički rad obloge tunela. S obzirom na preimućstva koja ima kružni oblik, u vezi statičkih i hidrauličkih uslova, njegovi nedostaci vezani za izradu se nadoknađuju prednostima iz predhodna dva uslova. Ovo je i razlog što se sada kod projektovanja ovakvih objekata veoma često projektanti odlučuju na kružni oblik.

3.1. Dimenzionisanje gravitacionih tunela

Već je naglašeno da kao najbolji oblici poprečnog preseka hidrotehničkih tunela su oblici koji imaju najmanji hidraulični otpor. Ovakav uslov najbolje zadovoljavaju kružni, visokozasvođeni i potkovičasti odlici.

Vrednosti koeficijenata rapavosti η kod gravitacionih tunela biće prikazane u tabeli 1.

Tabela 1. *Vrednosti koeficijenata hrapavosti η kod gravitacionih tunela*

Tip br.	Karakteristika površine dovoda	Vrednost za η			Napomena
		srednje	najveće	najmanje	
1.	2.	3.	4.	5.	6.
1.	*Gravitacioni tuneli u neobeleženoj steni* **a)** Gravitacioni tuneli pod srednjim uslovima-zidovi izravnjani pomoću otklanjanja ispada stene **b)** Gravitacioni tuneli pod nepovoljnim uslovima-veoma neravna površina stene, malo veće izbijanje prema projekt. profilu	0,030 0,040	0,038 0,045	0,038 -	**1)** Dati uslovi su individualni zato se navedene vrednosti daju samo radi orjentacije **2)** Taloženje nanosa smanjuje koeficijenat rapavosti

2.	*Gravitacioni tuneli u delimično malterisanoj steni* **a)** Pri torketiranju ili malterisanju stene, bez izrade žleba u donjem delu preseka **b)** Pri izradi žleba u donjem delu preseka i delimičnom malterisanju	0,030 0,023	0,022 -	- 0,019	
3.	*Gravitacioni tuneli obloženi sa običnom betonskom oblogom bez malterisanja i glačanja* **a)** Pri glatkom betonu koji se dobija pomoću dobro izrendisane oplate, bez ispada i rupa **b)** Pri rapavom betonu koji nosi na sebi tragove oplate (udubljenja, tragove vlakna)usled lošeg naleganja dasaka oplate, i za tip 3-a kada se na dnu oplate taloži pesak i šljunak	0,014 0,016	0,015 0,018	0,013 0,015	**1)** Dati uslovi su individualni zato se navedene vrednosti daju samo radi orjentacije. **2)** Ako ima mahovine (bez nanosa) koeficijenat rapavosti povećava se za 0,002
4.	*Gravitacioni tunel sa obrađenom, omalterisanom ili uglačanom površinom betona* **a)** Po visokom kvalitetu radova sa površinom omalterisanom cementnim malterom i uglačanom **b)** Pri dobrom kvalitetu radova površina je dobro uglačana i izravnana, spojnice su uglačane	0,011 0,012	- 0,013	0,010 0,011	
5.	*Gravitacioni tuneli sa torketiranom površinom* **a)** Pri pažljivom čišćenju četkom od čeličnih žica i pažljivom glačanju **b)** Pri čišćenju četkom od čeličnih žica i sprečavanju obrazovanja ,,rubova,, između torket-betona i malterisane površine **c)** Običan torket-beton bez preduzimanja nekih specijalnih mera	0,013 0,018 0,019	0,015 - 0,023	0,012 0,016 -	

Prema *Pernatu,* kod gravitacionih tunela, neophodno je da visina vode u tunelu bude nešto niža od visine samog tunela s obzirom na formiranje talasa, i treba da se kreće u granicama od:

$$1{,}7 \cdot r \leq t \leq 1{,}85 \cdot r$$

gde su:

r – polovina najveće širine u proseku
t – visina punjenja tunela

Na slici br. 3, prikazan je visokozasvođeni i potkovičasti oblik sa dimenzijama koje, prema *Pernatu,* najbolje udovoljavaju hidrauličke zahteve i kojih se treba pridržavati.

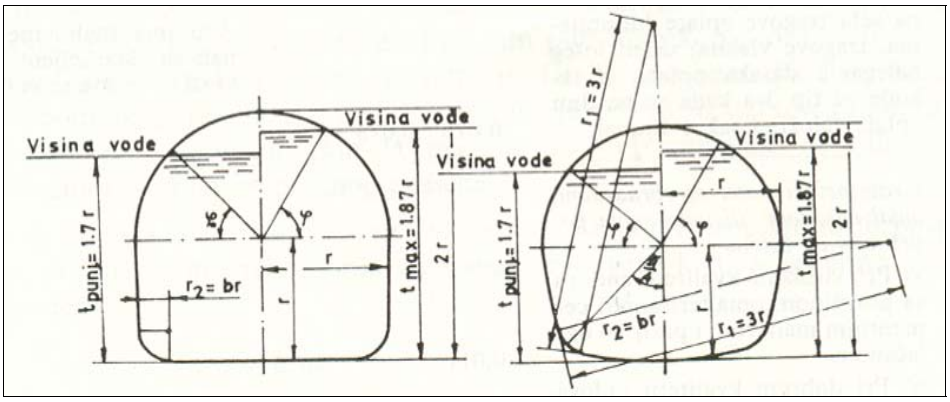
Propusna moć hidrotehničkog tunela može se proračunati po obrascu Forhajmera:

$$Q = 1/n \cdot \omega \cdot R^{0,7} \cdot I^{0,5},$$

gde su:

n – koeficijent rapavosti prema Gangije-Kuteru, (tabela 1)
ω – površina slobodnog preseka, pod vodom, m^2,
R – hidraulički radijus,
I – pad dna tunela.

Na primer, čvrsta metamorfna stena koja je blago poremećena ili naborana ima ocenu A=22 (iz Tabele 1). Stenska masa je umereno sastavljena, sa spojevima koji su upravni na osu tunela koji se pruža pravcem istok-zapad, i padom između 20˚ i 50˚.



Sl. 3. *Optimalni oblici hidrotehničkih tunela (prema Pernatu)*

Ovaj opšti obrazac za propusnu moć hidrotehničkih tunela preradio je Pernat specijalno za proračunavanje kapaciteta gravitacionih tunela i ovako prerađen obrazac glasi:

$$Q = P \cdot 1/n \cdot r^{2,7} \cdot I^{0,5},$$

gde je:

P – koeficijenat koji zavisi od vrednosti odnosa t : r (odnos punjenja i poluširine tunela) – (slika 3)

Ovaj obrazac omogućava da se, ukoliko se poznaju ostale vrednosti u obrascu, odredi poluširina tunela r, na osnovu koje se može, koristeći sliku 3, izvršiti dimenzionisanje željenog profila.

4. ZAKLJUČAK

Shodno predhodnim stavovima, donosi se zaključak da je za izbor oblika poprečnog preseka tunela Kriveljske reke potrebna detaljna analiza, da bi postojanost ovog hodnika bila optimalna.

LITERATURA

[1] Projekat 17004 MN
[2] Elaborat o geološkim istraživanjima i fizičko-mehaničkim ispitivanjima stena trase tunela za izmeštanje ''Kriveljske reke'', Institut za rudarstvo i metalurgiju Bor, Bor, 2008.
[3] Milovan Antunović Kobliška, Opšti rudarski radovi, Izdavačko preduzeće ''Građevinska knjiga'', Beograd, 1973.
[4] Prof. dr Petar Jovanović, Izrada podzemnih prostorija velikog profila, Izdavačko preduzeće ''Građevinska knjiga'', Beograd, 1978.
[5] M. Ljubojev, Z. Stojanović, D. Mitić, D. Ignjatović, Predlog načina izrade novog tunela Kriveljske reke, ''Inovacije i razvoj'', broj 1, Bor, 2009.
[6] M. Ljubojev, D. Ignjatović, L. Đ. Ignjatović, S. Krstić, Z. Stojanović, Ocena stabilnosti stena trase tunela Kriveljske reke nastala upoređivanjem klasifikacija čvrstoće stene, ''Bakar'', broj 1, Bor, 2009.
[7] S. Čosić, K. Okanović, Modeliranje naponsko-deformacijskog stanja numeričkim metodama kod širokočelnog otkopavanja, Rudarski radovi, br. 2-2010, str. 53-72

UDK: 551.49:622.26(045)=20

Milenko Ljubojev[*], *Dragan Ignjatović*[*], *Lidija Đurđevac Ignjatović*[*]

PROPOSAL OF CROSS SECTION FOR THE KRIVELJ RIVER TUNNEL**

Abstract

New Krivelj River tunnel represents the capital object of great ecological importance for Bor and its environment and also for a part of the Black Sea catchment (River Danube). The Krivelj River tunnel will be developed in an environment that is very unfavorable for development and persistence of the room, both during construction and period of usage. Therefore, it is necessary to carry out a detailed investigation of the rock mass condition and based on these data to determine which type of cross section should be taken as the optimum solution.

Key words: *Krivelj River tunnel, tunnel cross section.*

1. INTRODUCTION

Necessity for creation the new Krivelj River tunnel was caused by the poor condition of the existing Krivelj River collector. Its development would be partly through the flotation tailing dump, and partly through the rock massif. The new tunnel will regulate the flow of the Krivelj River, because the existing flow may be interrupted by destruction the old Krivelj River collector.

[*] *Mining and Metallurgy Institute Bor*
** *This paper is produced from the project no. 33021 which is funded by means of the Ministry of Education and Science of the Republic of Serbia*

Figure 1. *Satellite image of flotation tailing dumps*

Along the tunnel route, the field prospecting drillings will be carried out with the aim of determination the rock material.

The following investigation works will be done:

- Prospecting drilling with coring,
- Detailed engineering-geological core mapping,
- Selection of representative samples of each isolated lithologic unit,
- Determination of rock mass cracks and dividity of the obtained core (RQD).

2. CONSTRUCTION OF THE NEW KRIVELJ RIVER TUNNEL

The proposed new route for the Krivelj River tunnel would be in a length of about 2530 m. Tunnel would be made from altitude K +246 (in contact with the Krivelj river out of the tailing dump) to +272 K, per ascent of 1% (to the circuit with the old tunnel).

Construction of the tunnel would be by ascent, and it is going to be divided into two parts:

- I PHASE - production would be from its future out, along the Krivelj river, with ascent of 1%, to K+267, in the total length of about 2000 m (Figure 1). Development of the tunnel is through the weak rock environment, with constant water inflow.
- II PHASE – will be from the end of the first phase to the circuit section of the old tunnel. The length of these shares would be around 500 m, and from K+267 to K+272. Construction of these shares would be through the flotation tailing dump, Field 2, and partly by the former river bed.

3. CROSS-SECTION FORMS OF HYDROTECHNICAL TUNNEL

A cross-section form of hydrotechnical tunnel depends on several factors, among which are:

- Static conditions of the tunnel cover,
- Hydraulic conditions of the tunnel, and
- Conditions of the tunnel construction

Hydrotechnical cover of the tunnel has to be such sized that it has the smallest volume and, at the same time, the ability to support internal and external pressures without deformations. Beside all this, it has to be waterproof.

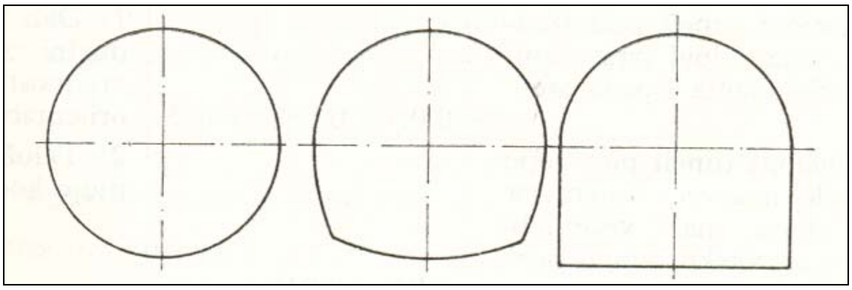


Figure 2. *Forms of free sections of hydrotechnical tunnel*

According to the hydraulic conditions, the form of cross section should provide the highest flow in a certain fall and the lowest cross section. High camera form, circular and horse–shoe form are the best forms that satisfy these conditions (Figure2).

Taking care of all influential factors, the choice of form the tunnel cross-section still overcomes the hydrotechnical condition related to the tunnel lining static work. Given the priority that a circular form has, the static and hydraulic conditions, the disadvantages related to compensate the advantages of previous two conditions.

4. SIZING THE GRAVITATIONAL TUNNEL

It has been emphasized yet that the best forms of cross-section of the hydrotechnical tunnel are the forms that have the smallest hydraulic resistance, such as circular, high camera and horse–shoe form.

Values of roughness coefficient η, for gravitational tunnels, will be shown in Table1.
value of 8 MPa and average *RQD* value of 70%. The slightly rough and slightly weathered joints with a separation of < 1 mm, are spaced at 300 mm. Tunneling conditions are anticipated to be wet.

Table 1. *Value of the roughness coefficient η for gravitational tunnels*

Type number	Characteristics of lead	Value for η			Note
		secondary	highest	lowest	
1	2	3	4	5	6
1	*Gravitational tunnels in the unmarked rock* **a)** Gravitational tunnels under conditions of medium-leveled walls with persistent removal of rocks	0.030	0.038	0.038	**1)** Conditions are given individual because the values are given only for orientation
	b) Gravitational tunnels under unfavorable conditions; very uneven surface of the rock, a little more outbreak from projected profile	0.040	0.045	-	**2)**Reduce sediment deposition of roughness coefficients

2	*Gravitational tunnels in the partially mortared rock* **a)** During gunite or mortaring of rock**,** without making groove in the bottom of the section **b)** During groove creating in the bottom of the section and partial rendering	0.030 0.023	0.022 -	- 0.019	
3	*Gravitational tunnels fitted with the plain concrete coverings without rendering and polishing* **a)** Smooth concrete that comes with good planning of shuttering, without the persistent and hole **b)** Roughness concrete that bears the traces of self shuttering (recess, trace fiber) due to poor shuttering contact boards, and type 3-a when sand and gravel are settling on the bottom of the shuttering	0.014 0.016	0.015 0.018	0.013 0.015	**1)** Conditions are given individually because the values are given only for orientation. **2)** If there is moss (no drift) the roughness coefficient increase for 0.002
4	*Gravitational tunnel with the process, plastered or polished concrete surface* **a)** By high quality works from the surface of plastered cement mortar and polished **b)** In the good quality work, the surface is well polished and flattened; connectors are polished	0.011 0.012	- 0.013	0.010 0.011	
5	*Gravitational tunnels with concrete lining area* **a)** During the careful cleaning with steel wire brush and careful polishing **b)** During cleaning with steel wire brush and prevention for creating “edges” between the concrete and gunite-mortaring surface **c)** Gunite-concrete without any special measures	0.013 0.018 0.019	0.015 - 0.023	0.012 0.016 -	

According to *Pernat,* for gravitational tunnels, it is necessary that the water level in the tunnel is a little bit lower than the tunnel height, considering the formation of waves, and should be within the range of:

$$1.7 \cdot r \leq t \leq 1.85 \cdot r$$

where:

r – half of the maximum width (average value)

t – height of the tunnel filling.

Figure 3 shows the high arched and horse–shoe form with the sizes that fulfill all necessary hydraulic requirements, according to Pernat.

Gap power of hydrotechnical tunnel can be calculated by the Forhaymer form:

$$Q = 1/n \cdot \omega \cdot R^{0.7} \cdot I^{0.5},$$

where:

n – roughness coefficient according to Gangy - Kutter, (Table 1)

ω – area of free cross-section, under the water, m^2,

R – hydraulic radius,

I – fall of the tunnel bottom.

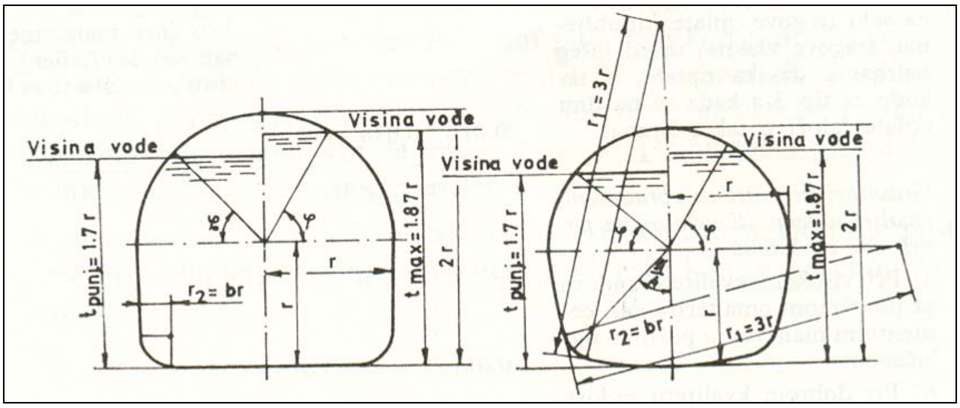


Figure 3. - *Optimum forms of the hydrotechnical tunnel (according to Pernat)*

This general form for the transference of hydrotechnical tunnel was revised by Pernat specifically for calculation the capacity of the gravitational tunnel:

$$Q = P \cdot 1/n \cdot r^{2.7} \cdot I^{0.5},$$

where:

P – coefficient, depending on the values of t : r (ratio of charge and half width of the tunnel) - (Figure 3)

This form allows determining the half width of the tunnel r, based on which, using Figure 3, the sizing of wanted profile could be done. It is possible only knowing the other values in the form.

5. CONCLUSION

Pursuant to the foregoing paragraphs, it can be concluded that the choice of ross-section forms for the Krivelj River tunnel needs more detailed analysis, which will result in the optimum existence of this corridor.

REFERENCES

[1] Project 17004 Ministry of Science (in Serbian)

[2] Project Report on geological explorations and physical - mechanical testing of rocks for the tunnel route of relocation of the “Krivelj River“, Mining and Metallurgy Institute Bor, Bor, 2008 (in Serbian)

[3] Milovan Antunović Kobliška, General Mining Works, Publishing Company ’’Gradjevinska knjiga’’, Belgrade, 1973 (in Serbian)

[4] Prof. Ph. D. Petar Jovanović, Development of the Large Profile Underground Rooms, Publishing Company’’Gradjevinska knjiga’’, Belgrade, 1978 (in Serbian)

[5] M. Ljubojev, Z. Stojanović, D. Mitić, D. Ignjatović, Proposal of Construction Method the New Tunnel of the Krivelj River, Innnovations and Development, No. 1, Bor, 2009 (in Serbian)

[6] M. Ljubojev, D. Ignjatović, L. Dj. Ignjatović, S. Krstić, Z. Stojanović, Evaluation the Rock Stability of the Tunnel Route of the Krivelj River Obtained by Comparison the Classification of the Rock Strength, Copper, No. 1, Bor, 2009 (in Serbian)

[7] S. Čosić, K. Okanović, Modeling of stress-deformation state using the numerical nethods in the wide face mining, Mining Engineering, No. 2-2010, pp. 73-92

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:622.271:681.51(045)=861

Alen Baraković[*] Ekrem Bektašević[**], Izudin Sjerotanović[**]

ODREĐIVANJE FAKTORA SIGURNOSTI U STIJENSKOM MATERIJALU NA PRIMJERU BORSKOG LEŽIŠTA NUMERIČKOM METODOM „SWASE“[***]

Izvod

U ovom radu dat je jedan pristup problemu analize stabilnosti kosina u stijenskom materijalu primjenom metode računarskog modeliranja. Numeričke metode, modeliranje i računar postali su gotovo sinonim odnosno planiranje, izrada, razvoj i korištenje pojedinih objekata (površinski kopovi, kamenolomi, saobraćajnice itd.) mora se "posmatrati pod lupom računara". Takođe je potrebno istaknuti da primjena računarske tehnike i odgovarajućih softvera u analizi stabilnosti mijenja i ulogu inženjera. Težište inženjerskog rada se premiješta sa dugotrajnog numeričkog rada na korektnu obradu osnovnih podataka, dobijenih geotehničkim istraživanjima terena, s ciljem određivanja mjerodavnih ulaznih podataka za softver. Takođe, potrebno je staviti akcenat na interpretaciju rezultata dobijenih uz pomoć računara i provjeri njihove saglasnosti sa ulaznim podacima. Na osnovu toga jasno je da adekvatno korišćenje računara zahtijeva vrlo složenu inženjersku analizu koja podrazumijeva poznavanje kako geotehničkog aspekta problema tako i potencijala softvera koji se koristi. Kao ishod takve analize postiže se da rezultati imaju jedan novi, viši kvalitet nego što je to ranije bio slučaj.

Ključne riječi: *stabilnost kosina, stijenski materijal, numeričke metode, proračun*

1. UVOD

Svako pomjeranje u padinama i kosinama podrazumijeva da je došlo do prekoračenja čvrstoće materijala pod uticajem smičućih napona, a ono se može manifestovati na razne načine i u različitom intenzitetu, od gotovo neprimjetnog puzanja, sporijeg ili bržeg klizanja, pa do vrlo brzih procesa odronjavanja. Prirodni nagibi padina su formirani tokom dugog vremenskog perioda i prilagođeni stvarnoj čvrstoći smicanja stijenske mase.

Kod izrađenih objekata kao što su: etaže na površinskim kopovima, usjeci, nasipi, kanali, propusti, putevi, željezničke

[*] *BBM VAREŠ d.o.o. Vareš, BiH*
[**] *GRAMAT d.o.o. Gračanica, BiH*
[***] *Ovaj rad je proistekao iz magistarskog rada pod naslovom: „Modeliranje metoda stabilnosti kosina u stijenskom materijalu“, Tuzla 05.12.2003 godine, Rudarsko-geološko-građevinsk fakultet Tuzla*

pruge i sl., često se dešava da se obruše dijelovi kosine ili čak cjelokupne kosine. Takođe može doći do potpunog obustavljanja proizvodnje na površinskim kopovima i kamenolomima kao i do obustavljanja saobraćaja na saobraćajnicama.

Pri formiranju kosina površinskog kopa nastaju deformacije stijenskog masiva oko obodnih dijelova kopa, kao posljedica narušavanja prirodne ravnoteže. Zona uticaja rudarskih radova oko obodnih dijelova kopa uslovljena je prirodno-geološkim i rudarsko-tehničkim uslovima eksploatacije. Ponovno uravnotežavanje potkopanog stijenskog masiva može dovesti do pojave klizišta koje može ugroziti bezbjednost proizvodne mehanizacije i stabilnost kosina i etaža na kopu. Projektovane kosine moraju zadovoljiti uslov stabilnosti koji garantuje sigurno i bezbjedno izvođenje rudarskih radova. U toku procesa eksploatacije ležišta neki parametri (hidrogeološki uslovi, pojave diskontinuiteta itd.) mogu biti izmijenjeni, što može uzrokovati smanjenje faktora sigurnosti na kritičnu vrijednost ($F \leq 1$) i stvoriti uslove za pojavu klizišta.

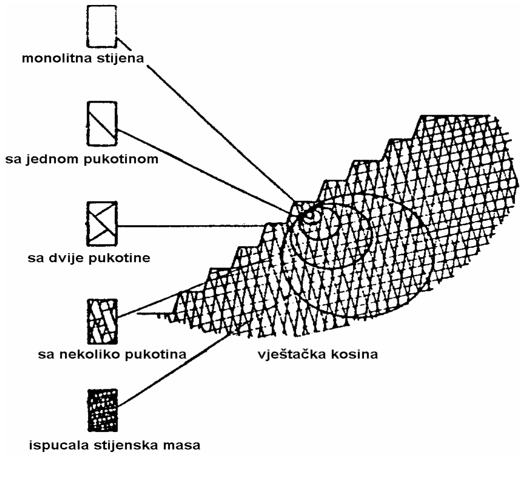
U procesu računarskog modeliranja stijena, javljaju se problemi vezani za složenost stijenske mase (nehomogenost, anizotropija, diskontinuiranost itd.).

Zbog toga je postavka numeričkog modela veoma kompleksna i to nije zadatak koji bi se mogao rutinski realizovati.

2. ČVRSTOĆA STIJENSKE MASE

Stijenske mase posjeduju svojstvo da se odupru dejstvu spoljnih sila. Ako se opterećenje postepeno uvećava u jednom trenutku dostići će graničnu vrijednost koju stijenska masa može da prihvati, pružajući otpor bez uvećanih vidljivih deformacija. Smičuća čvrstoća predstavlja najveći smičući napon koji se može nanijeti strukturi stijene u određenom pravcu. Kada je dostignut najveći mogući smičući napon, praćen plastičnim deformacijama, kaže se da je došlo do loma, pri čemu je mobilisana sva smičuća čvrstoća stijenske mase. Tada smičući naponi imaju tendenciju da pomjere dio mase u odnosu na ostalu masu stijene ukoliko je lom lokalizovan samo u ravni smicanja tj. gdje se pojavljuje klizna površina.

U zavisnosti od razmjere posmatranja stijensku masu možemo tretirati kao: monolitnu, sa jednom familijom pukotina, sa dvije familije pukotina, sa nekoliko familija pukotina i kao jako izlomljenu. Svi ovi tipski modeli stijenskih masa u pogledu čvrstoće bitno se međusobno razlikuju.



Sl. 1. *Uprošćeni prikaz uticaja razmjere pri izboru modela stijene*

U praksi često nailazimo na stijenske mase sa jasno izraženom jednom familijom pukotina (škriljavosti, slojevitosti, klivaža). Za takav slučaj E. Hoek predlaže izraz za određivanje čvrstoće na smicanje u obliku:

$$\sigma'_1 = \sigma'_3 + \frac{2(c'_i + \sigma'_3 \cdot tg\varphi'_i)}{(1 - tg\varphi'_i \cdot tg\beta)\sin 2\beta} \qquad (1)$$

gdje je:

c'_i – prividna kohezija,

φ'_i – prividni ugao trenja duž pukotina,

β - nagib površine pukotine prema većem glavnom naponu,

σ'_1 – maksimalni efektivni glavni napon pri lomu,

σ'_3 – minimalni efektivni glavni napon pri lomu.

2.1. COULOMB – MOHR-ov (linearan) Zakon loma

Prvi upotrebljiv zakon loma pripisuje se Coulomb-u, 1776 godine, koji definiše smičuću čvrstoću i dat je izrazom:

$$\tau_f = c + \sigma_n \tan\varphi \qquad (2.)$$

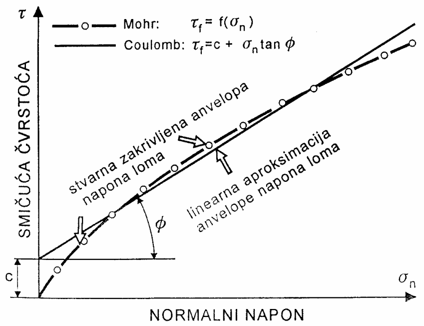
Ovaj izraz se često naziva i Mohr – Coulomb-ov zakon loma jer Mohr-ova hipoteza podrazumijeva da smičuća čvrstoća zavisi od normalnog napona, što se u opštem obliku piše kao $\tau_f = f(\sigma_n)$, a Coulomb-ov zakon jednostavno kaže da je $f(\sigma_n)$, linearna funkcija normalnog napona, dovoljno tačan opis veličina napona pri lomu. Priroda ove aproksimacije, u odnosu na realnost, je prikazana na slici 2. Smičuća čvrstoća se od kasnih dvadesetih godina ovog vijeka opisuje empirijskim linearnim zakonom u funkciji efektivnih napona izrazom:

$$\tau_f = c` + (\sigma_n - u) \tan \varphi` = \\ = c` + \sigma_n` \tan \varphi` \qquad (3.)$$

gdje se za parametre obično koriste sljedeći nazivi:

c` - kohezija za efektivne napone ili prividna kohezija,

φ` - ugao trenja za efektivne napone ili ugao smičuće otpornosti.



Sl. 2. *Zavisnost smičuće čvrstoće od normalnog napona*

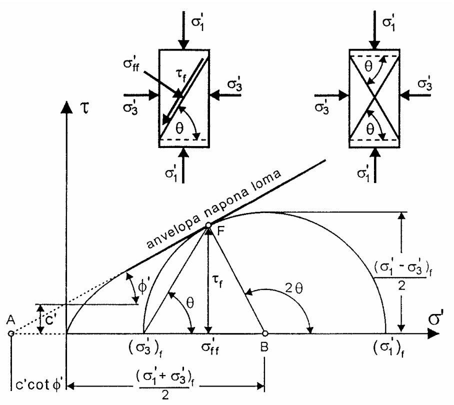
Smičuća čvrstoća stijenskog materijala se može takođe izraziti i preko glavnih efektivnih napona $\sigma_1`$ i $\sigma_3`$ pri lomu u posmatranoj tački. Prava linija opisana gornjom jednačinom će tangirati Mohr-ove krugove efektivnih napona, kao što je prikazano na slici 3.

Koordinate tangentne tačke F (τ_f, $\sigma_f`$) su:

$$\tau_f = \frac{(\sigma'_1 - \sigma'_3)_f}{2} \sin 2\theta \qquad (4.)$$

$$\sigma'_f = \frac{(\sigma'_1 + \sigma'_3)_f}{2} + \frac{(\sigma'_1 - \sigma'_3)_f}{2} \cos 2\theta \qquad (5.)$$

gdje je θ teorijski ugao između ravni u kojoj djeluje maksimalni glavni napon i ravni loma.



Sl. 3. *Mohr-ov dijagram napona loma*

S obzirom na simetričnost Mohr-ovog dijagrama u odnosu na osu normalnih napona, postoje dvije takve ravni koje zaklapaju jednake uglove u odnosu na pravac najvećeg glavnog napona. Veličina ugla se može odrediti iz geometrijskih odnosa i uslova da zbir unutrašnjih uglova trougla ABF iznosi 180°. Ugao u tjemenu A ima veličinu φ`, u tjemenu B ugao je 180° - 2θ, a u tjemenu F ugao je 90°. Iz uslova da je φ` + 90° + (180° - 2θ) = 180° proizilazi da je:

$$\theta = \pm (45^{\circ} + \varphi`/2) \qquad (6)$$

Sa slike 3. može se dobiti veza između efektivnih glavnih napona i parametara čvrstoće:

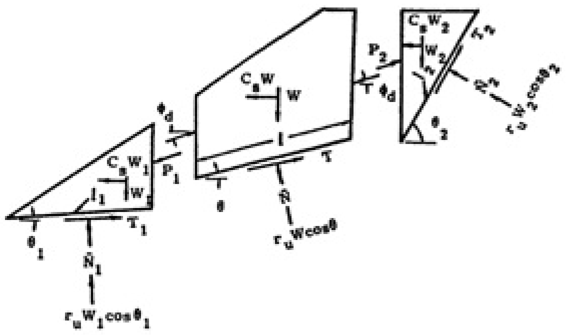
$$\sin\varphi' = \frac{\dfrac{(\sigma_1' - \sigma_3')_f}{2}}{c'\cos\varphi' + \dfrac{(\sigma_1' + \sigma_3')_f}{2}} \qquad (7)$$

tako da je razlika glavnih napona pri lomu:

$$\left(\sigma_1' - \sigma_3'\right)_f = \left(\sigma_1' + \sigma_3'\right)_f \sin\varphi' + 2c'\cos\varphi' \qquad (8)$$

2.2. Analiza stabilnosti kosina u stijenskom materijalu metodom blokova

Postoji više metoda proračuna stabilnosti kosina, a jedna od njih je metoda blokova koja je analizirana u ovom radu.



Sl. 4. *Grafički prikaz metode blokova*

Sile za svaki blok,

$$N_i = \{w_1[\cos\phi_d - r_u \cos\theta_i \cos(\phi_d - \theta_i) - \\ - C_s \sin\phi_d] + c_i l_i [\sin(\phi_d - \theta_i)] / Fs\} / \\ / \{\cos(\phi_d - \theta_i) - [tg\phi_i \sin(\phi_d - \theta_i)] / Fs\} \quad (9)$$

$$P_i = (N_i \sin\theta_i - T_i \cos\theta_i + \\ + r_u W_i \cos\theta_i + \sin\theta_i + C_S W) / \cos\phi_d \quad (10)$$

Faktor sigurnosti se određuje na osnovu rješavanja sistema nelinearnih jednačina, a broj tih jednačina je zavisan od broja blokova sa kojim modeliramo klizno tijelo. Svaki blok sastoji se od dvije jednačine (Pi i Ni). Tako da se faktor sigurnosti metodom blokova može prikazati slijedećom jednačinom:

$$Fs = \frac{N_1 \, tg\phi_1 + N_2 \, tg\,\phi_2}{W \sin ß} = \\ = \frac{\sin\theta_2 \, tg\,\phi_1 + \sin\theta_1 \, tg\,\phi_2}{\sin(\theta_1 + \theta_2) tg\, ß} \quad (11.)$$

Zavisno od zahtjeva projektanta i odgovarajućih propisa za kosine u stijenskom materijalu kao i u tlu može se analizirati i uticaj seizmičkih efekata bilo od zemljotresa ili uticaj miniranja.

3. PRIMJER PRORAČUNA I ANALIZA METODOM "SWASE" (METODA BLOKOVA)

Za proračun i analizu stabilnosti kosine u stijenskom materijalu upotrijebljen je karakterističan poprečni profil (profil 2) koji prolazi kroz površinski kop "Bor", a svi potrebni ulazni podaci (inženjersko-geološke i fizičko-mehaničke karakteristike stijenskog materijala) uvršteni su u programske pakete koji su specijalno modifikovani za kvalitetno izvođenje navedene analize. Prilikom proračuna i analize korišten je karakterističan profil koji obuhvata tri vrste materijala odnosno tri litološka člana i to:

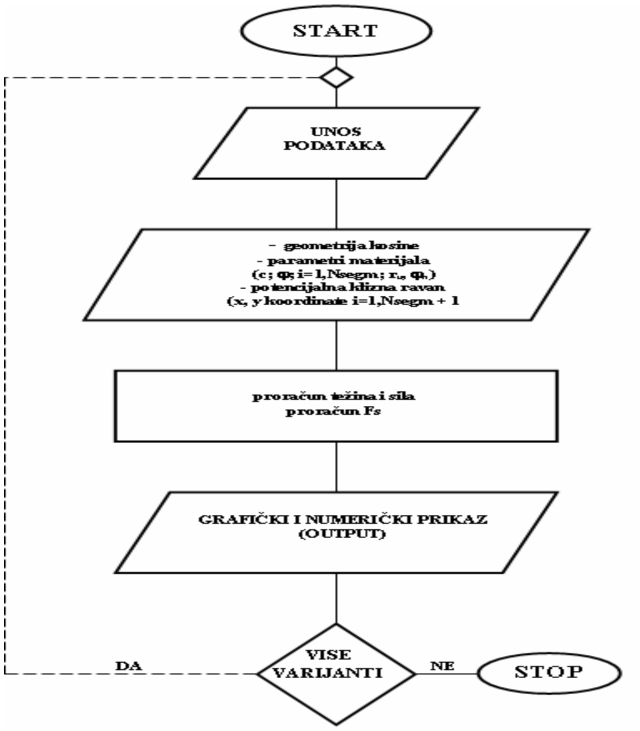
- materijal 1 – silifikacija i ruda,
- materijal 2 – kaolinisani andezit,
- materijal 3 – piroklastiti.

Tabela 1. *Prikaz parametara materijala korištenih u proračunu*

Vrsta stijene	φ^o	γ (kN/m^3)	c (kN/m^2)
Kaolinisani andezit	42,0	26,5	251,00
Hloritisani andezit i piroklastiti	42,4	26,5	300,00
Svjež andezit i konglomerati	43,3	26,5	350,00
Silifikacija i ruda	44,0	27,8	418,00

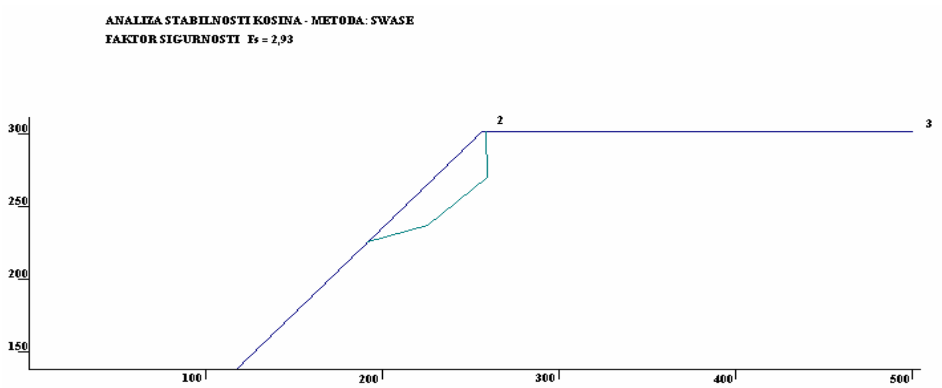
Proračun je izvršen sa programskim paketom "SWASE". Za razliku od izvornog koda datog u literaturi proračun je izvršen sa modifikovanim programom, a modifikacije su učinjene u cilju povećanja kvaliteta, tačnosti, olakšanja kod unosa podataka kao i poboljšanja grafičkog prikaza rezultata proračuna. Prvobitna šema toka programa sastojala se od unosa podataka od kojih se većina unosi manuelno. Modifikovana verzija programa koja je korištena u ovom radu je sa mnogo jednostavnijim načinom unosa podataka preko koordinata kako terena tako i pretpostavljnih kliznih ravni.

Takođe i izlazna lista se sastoji od tekstualnog dijela, rezultata proračuna i grafičkog prikaza što omogućava daleko veći broj razmatranih varijanti u veoma kratkom vremenu. Ako bi se izvršilo poboljšanje programa uvođenjem upravljanja "mišem" omogućio bi se daleko brži i lakši rad u analizi stabilnosti kosina. Na slijedećoj slici prikazan je tok proračuna metodom „SWASE".

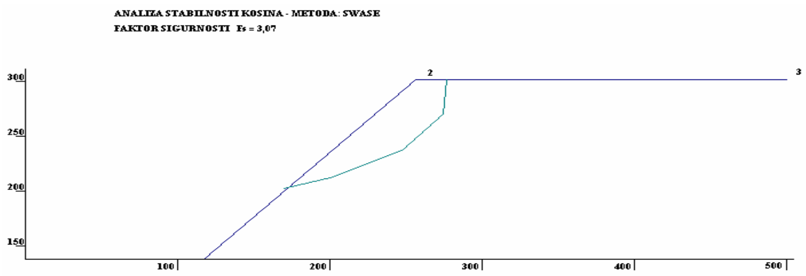


Sl 5. *Šema toka proračuna*

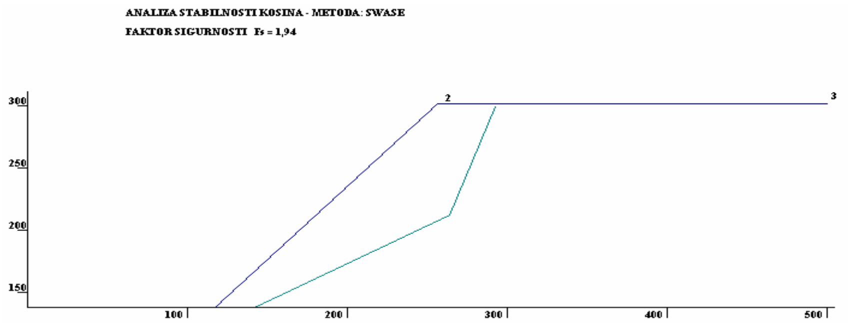
Na slici 6., 7., i 8. prikazani su rezultati koji su dobijeni proračunom za tri različite potencijalne klizne ravni.



Sl. 6. *Prikaz klizne ravni 1*



Sl. 7. *Prikaz klizne ravni 2*



Sl. 8. *Prikaz klizne ravni 3*

Izlazna lista proračuna:

SET	DSBM	DSTP	DSME	LNS2	LNS	RU	SEIC	GAMMA	ANOUT
1	000	1.235	0.515	91.082	148.223	0.100	0.000	27.800	0.831

BW	F
257.000	**1.937**

SET	DSBM	DSTP	DSME	LNS2	LNS	RU	SEIC	GAMMA	ANOUT
2	0.305	1.603	0.727	31.016	48.166	0.100	0.000	27.800	0.831

BW	F
257.000	**2.933**

SET	DSBM	DSTP	DSME	LNS2	LNS	RU	SEIC	GAMMA	ANOUT
3	0.283	0.835	0.464	43.186	55.902	0.100	0.000	27.800	0.831

BW	F
257.000	**3.066**

4. ANALIZA REZULTATA PRORAČUNA

U radu je korišten klasični metod odnosno metoda blokova – "Swase", koja je uzela u obzir fizičko mehaničke osobine materijala, i to parametre smicanja, a koji su dobijeni laboratorijskim putem i redukovani na bazi klasifikacije stijenske mase prema geomehaničkom sistemu. Proračun i analiza je vršena po blokovima. Blokovi su modelirani na bazi inženjersko-geoloških podataka a na osnovu rezultata proračuna uočljivo je da se faktori sigurnosti kreću u granicama koje omogućavaju nesmetan rad na površinskom kopu. Proračun je rađen za projektovano stanje odnosno za ugao završne kosine 53^0.

5. ZAKLJUČNA RAZMATRANJA

Projektovane nagibe radnih i završnih kosina treba posmatrati kao promjenljive parametre koji se kreću između dvije krajnosti – ono što je sigurno nije ekonomično i obratno. Ovi se parametri u toku eksploatacije prilagođavaju stvarnom stanju uslova stijenskog masiva. Ovakav pristup zahtijeva da se u toku eksploatacije raspolaže sa dovoljno kvantitativnih podataka, kako o stijenskom masivu, tako i o posljedicama pojedinih zahvata na kopu, u pogledu stabilnosti. Pri ovome se podrazumijeva da su parametri miniranja prilagođeni vrsti stijenskog masiva i da su završne kosine pravilno urađene. Sa ovim programskim paketom prilikom analize stabilnosti kosina mogu se uzeti u obzir i uticaji seizmičkog efekta na stabilnost kosina a koji se javljaju prilikom izvođenja radova masovnog miniranja stijenskog materijala kao i seizmički efekti koji nastaju kao posljedica zemljotresa.

LITERATURA:

[1] S. Vujić; M. Berković; D. Kuzmanović; P. Milanović; A. Sedmak; M. Mičić, Primena MKE kod geostatičkih proračuna u rudarstvu, Univerzitet u Beogradu, 1990.

[2] Yang H. Huang, Stability analysis of earth slopes, University of Kentucky, 1983.

[3] Undeground Excavations in Rock, Hoek & E.T.Brown, Institution of Mining and Metallurgy, London, 1980

[4] R. Milanović, Stabilnost kosina površinskih kopova u stenskim masivima, Institut za bakar Bor

[5] Slope Analysis, R.N.Chowdhury, Developments in Geotechnical engineering, vol 22, 1978.

YU ISSN: 1451-0162
UDK: 622

UDK: 622.271:681.51(045)=20

Alen Baraković[*]*, Ekrem Bektašević*[**]*, Izudin Sjerotanović*[**]

DETERMINATION OF SAFETY FACTOR IN THE ROCK MATERIALS ON EXAMPLE OF THE BOR DEPOSIT USING THE "SWASE" NUMERIC METHOD[***]

Abstract

In this paper, an approach is given for analysis the slope stability in rock materials using the computer modeling method. Numeric methods, modeling and computer became almost synonym, more exactly planning, preparation, development and use of certain structures (surface mines, quarries, roads, etc.) should be "observed under the magnification of computer". Also, it is necessary to emphasize that the use of computerized technique and suitable software in stability analysis also changes the role of engineers. The center of engineering work is transferred from a long numeric work to correct the basic data processing, provided by geotechnical researches on the field, , with the aim of determination of proper entry data for software. Also, it is necessary to emphasize the role of interpretation the results obtained by computer and check up of their adjustment with entry data. Based on this, it is clear that the computer use requires very complex engineering analysis that implies understanding the geotechnical aspects of problems and potential of software that is used. As the result of such analysis, it is achieved that the results have a new higher quality in comparison to the previous procedures.

Key words: *slope stability, rock materials, numeric methods, calculation*

1. INTRODUCTION

Every movement in the slopes and inclinations implies that material strength was overloaded under the influence of shear tensions, and that can be manifested in different ways and different intensity, from almost invisible creeping, slower or faster sliding, to very fast processes of rock falling. Natural inclinations of slopes are formed during very long period of time and adjusted to the realistic shear strength of rock masses.

At the constructed structures as the benches at the open pits, cuts and notches, embankments, canals, culverts, roads, railway tracks, and other, it often happens that the parts of slopes or even whole

[*] *BBM VAREŠ d.o.o. Vareš, BiH*
[**] *GRAMAT d.o.o. Gračanica, BiH*
[***] *This paper is the result of Masters paper under title "Modeling of Slope Stability in Rock materials", Tuzla 05 December 2003, Mining-Geology-Civil Engineering Faculty Tuzla*

slopes collapse. Also, production at the open pits and quarries can be completely stopped as well as the traffic can be stopped on the roads.

During formation of slopes at the open pit, deformations of rock massif are created around parts of excavations as the result of disturbance of natural balance. Zone of influence the mining works around edge parts of the open pit is conditioned by the natural-geological and mining-technical conditions of exploitation. Newly created balance of mined rock masses can result in the landslide occurrence which can endanger the safety of production mechanization and stability of slopes and benches at the open pit. Designed slopes have to satisfy the stability conditions that guarantee secure and safe performance of mining operations. During the exploitation process of deposit, some of the parameters (hydrogeological conditions, occurrence of discontinuities, etc.) can be changed, what can result in reduction of safety factor to the critical value ($F \leq 1$) and create conditions for landslide occurrence.

In the process of computer modeling the rocks, the problems, related to complexity of rock masses, occur (inhomogeneity, anisotropy, discontinuity, etc.).

Due to this, the placement of numeric model is very complex and that is not a task that could be realized as a routine.

2. STRENGTH OF THE ROCK MASS

Rock masses have a characteristic to resist the influence of external forces. If the load is gradually increased, at one point of time it will reach the limit value that rock masses is able to accept, giving the resistance without increased visible deformations. Shear strength presents the highest shear tension that can be implied to the rock structure in certain direction. When the highest shear tension is reached, which is followed with plastic deformations, it can be said that the break has occurred, at what point all shear strength of rock mass was mobilized. Than shear tensions have tendency to move a part of the mass in relation to the other mass of the rock if break is localized only in the plane of shear, more exactly, where sliding plane occurs.

In relation to the scope of review, the rock mass can be treated as: monolithic, with one family of cracks, two families of cracks, several families of cracks and very cracked. All those typical models of rock masses, in the aspect of strength, are different from each other.

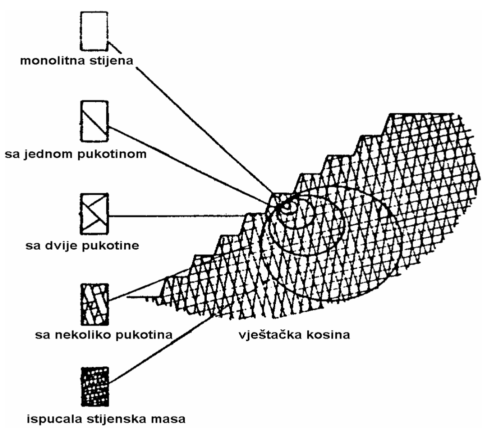


Figure 1. *Simplified presentation of ration impact in the selection of rock models*

In practice, the rock masses are often found with clearly expressed one family of cracks (schistosity, stratification, cleavage). For that case E. Hoek proposes expression for determination of strength to shear in the expression:

$$\sigma'_1 = \sigma'_3 + \frac{2(c'_i + \sigma'_3 \cdot tg\varphi'_i)}{(1 - tg\varphi'_i \cdot tg\beta)\sin 2\beta} \qquad (1)$$

where:

$c_i^{'}$ – imaginary cohesion,

$\varphi_i^{'}$ – imaginary angle of friction along the cracks,

β - inclination of crack surface towards higher main tension,

$\sigma_1^{'}$ – maximum effective principal tension at break,

$\sigma_3^{'}$ – minimum effective principal tension at break.

2.1. THE COULOMB – MOHR (linear) LAW OF BREAK

The first useable law of break is implied to Coulomb, in 1776, which defines the strength to shear and is given by the expression:

$$\tau_f = c + \sigma_n \tan\varphi \qquad (2.)$$

This expression is often called the Mohr – Coulomb law of break because the Mohr hypothesis implies that shear strength depends on normal tension, what in general case is written as $\tau_f = f(\sigma_n)$, and the Coulomb law simply indicates that f (σ_n) is, linear function of normal tension, sufficiently correct description of tensions at the occurrence of break. Nature of this approximate, in relation to reality, is present in Figure 2. Shear strength is described empirically, since the twenties of last century, by the linear law in a function of effective tension by the expression:

$$\tau_f = c` + (\sigma_n - u) \tan \varphi` =$$
$$= c` + \sigma_n` \tan \varphi` \qquad (3.)$$

Where the following terms are usually used for parameters:

c` - cohesion for effective tensions or imaginary cohesion,

φ` - angle of friction for effective tensions or angle of shear resistance.

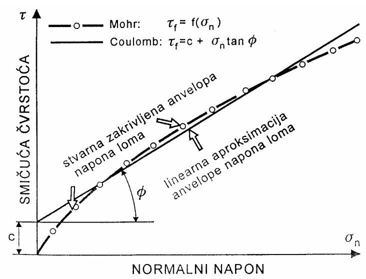


Figure 2. *Dependence of shear strength on normal tension*

Shear strength of rock material can also be expressed through main effective tensions $\sigma_1`$ and $\sigma_3`$ that occur at break in the observed point. Straight line that is described in above formula will be a tangent on the Mohr circles of effective tensions as it is present in Figure 3.

Coordinates of tangent point F (τ_f, $\sigma_f`$) are:

$$\tau_f = \frac{(\sigma'_1 - \sigma'_3)_f}{2} \sin 2\theta \qquad (4.)$$

$$\sigma'_f = \frac{(\sigma'_1 + \sigma'_3)_f}{2} + \frac{(\sigma'_1 - \sigma'_3)_f}{2} \cos 2\theta \qquad (5.)$$

Where θ is a theoretical angle between planes in which the maximum principle tension acts as well as the planes of break.

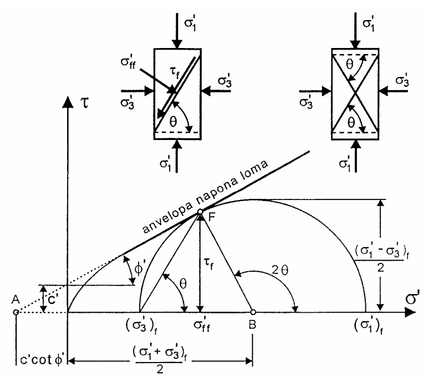


Figure 3. *The Mohr diagram of break tension*

Regarding the symmetry of the Mohr diagram in relation to the axe of normal tensions, there are two such planes that close same angles in relation to a direction of the highest principle tension. Size of angle can be defined from geometric relations and condition that the sum of inner angles of triangle ABF is 180°. Angle in point A has a value φ`, at point B angle is 180° - 2θ, and point F angle is 90°. From the condition that φ` + 90° + (180° - 2θ) = 180° it results that:

$$\theta = \pm (45^o + \varphi`/2) \qquad (6)$$

It can be concluded from Figure 3 that the relation between principle effective tensions and strength parameters:

$$\sin\varphi' = \frac{\dfrac{(\sigma'_1 - \sigma'_3)_f}{2}}{c'\cos\varphi' + \dfrac{(\sigma'_1 + \sigma'_3)_f}{2}} \qquad (7)$$

so the difference of principle tensions at break is:

$$\left(\sigma'_1 - \sigma'_3\right)f = \left(\sigma'_1 + \sigma'_3\right)f \sin\varphi' + 2c'\cos\varphi' \qquad (8)$$

2.2. Analysis of slope stability in the rock material using the method of blocks

There are several methods of slope stability calculation in this paper and one of them is a method of blocks that is analyzed in this paper.

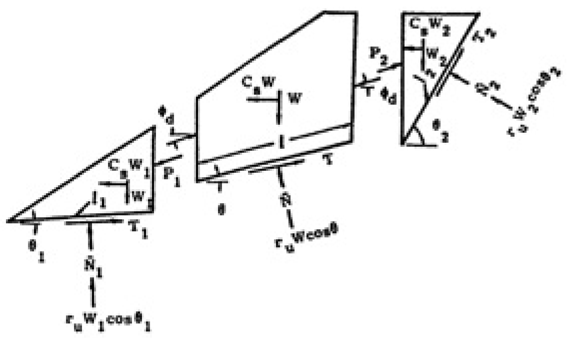


Figure 4. *Graphic presentation the method of blocks*

Forces for each block,

$$N_i = \{w_1[\cos\phi_d - r_u \cos\theta_i \cos(\phi_d - \theta_i) - \\ - C_s \sin\phi_d] + c_i l_i [\sin(\phi_d - \theta_i)] / Fs\} / \\ / \{\cos(\phi_d - \theta_i) - [tg\phi_i \sin(\phi_d - \theta_i)] / Fs\} \quad (9)$$

$$P_i = \big(N_i \sin\theta_i - T_i \cos\theta_i + \\ + r_u W_i \cos\theta_i + \sin\theta_i + C_S W\big) / \cos\phi_d \quad (10)$$

Safety factor is determined based on solving the system of nonlinear equations and number of such equations depends on number of blocks by which the sliding body was modeled. Each block is composed of two equations (Pi and Ni). Such as the safety factor of block method can be present by the following equation:

$$Fs = \frac{N_1 tg\phi_1 + N_2 tg\phi_2}{W \sin\beta} = \\ = \frac{\sin\theta_2 tg\phi_1 + \sin\theta_1 tg\phi_2}{\sin(\theta_1 + \theta_2) tg\beta} \quad (11.)$$

Depending on the requests of designers and appropriate regulations for slopes in the rock material as well as in the soil, the seismic effects of earthquake or blasting influence can be analyzed.

3. AN EXAMPLE OF CALCULATION AND ANALAYSIS USING THE “SWASE” METHOD (METHOD OF BLOCKS)

For calculation and analysis the slope stability in rock materials, a characteristic cross section (profile 2) is use, which crosses the open ppit “Bor”, and all required entering data (engineering geological and physical-mechanical characteristics of rock materials) are taken into the program packages that are specifically modified for carrying out the given analysis. In calculation and analysis, a characteristic profile is used that takes three types of materials, i.e. three lithological members, as follows:

material 1 – Silification and mineral,
material 2 – Andesite kaolinized,
material 3 – Pyroclastite.

Table 1. *Review of parameters used in calculation*

Type of rock	**φ°**	**γ (kN/m³)**	**c (kN/m²)**
Andesite kaolinized	42.0	26.5	251.00
Chloritinized andezite and pyroclastite	42.4	26.5	300.00
Fresh andesite and conglomerates	43.3	26.5	350.00
Silification and mineral	44.0	27.8	418,00

Calculation was done using the software package “SWASE”. As a difference from the source code given in literature, the calculation was done using the modified software and modifications were done to the aim of increase the quality, accuracy and easier way of data entry as well as the improvement of graphic presentation of calculation results. The original scheme of program flow was composed of data entries that were mostly entered manually. The modified version of program, used in this paper, is with much easier way of data entry using coordinates both of the field and assumed sliding planes.

Also, the output list is composed of textual part, calculation results and graphic presentation what provides greater number of reviewed options in a very short period of time. If the program improvement will be done by introduction of “mouse”, that would provide much faster and easier work in analysis of slope stability. Following Figure presents a flow of calculation using the “SWASE“ method.

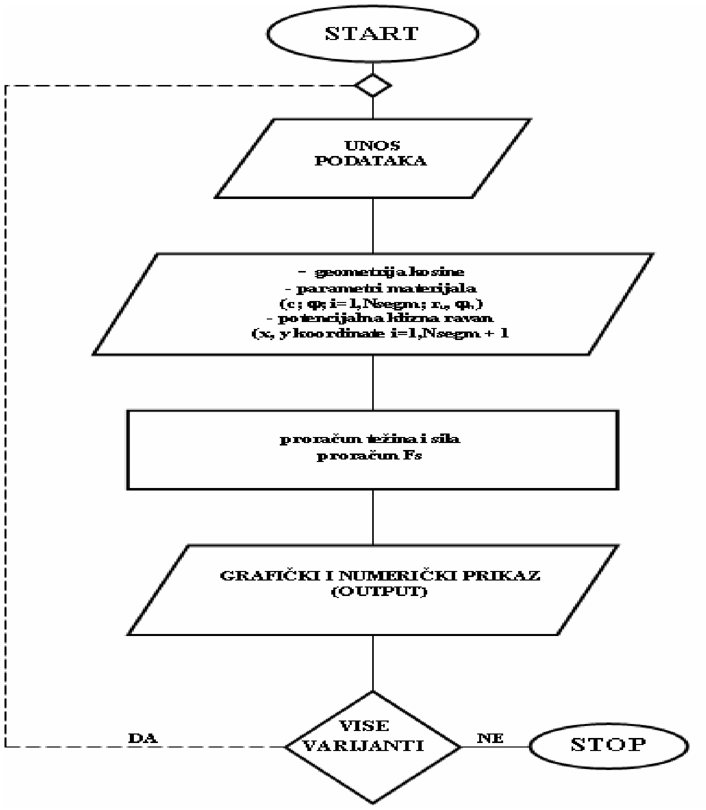


Figure 5. *Scheme of calculation flow*

Figures 6,7 and 8 presents the results provided by calculation of three different sliding planes.

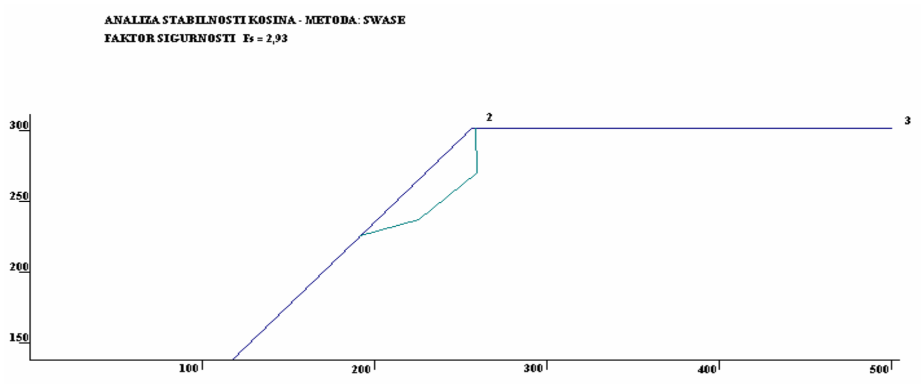


Figure 6. *Presentation of sliding plane 1*

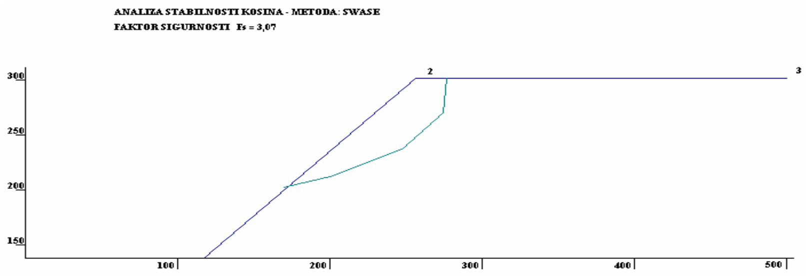


Figure 7. *Presentation of sliding plane 2*

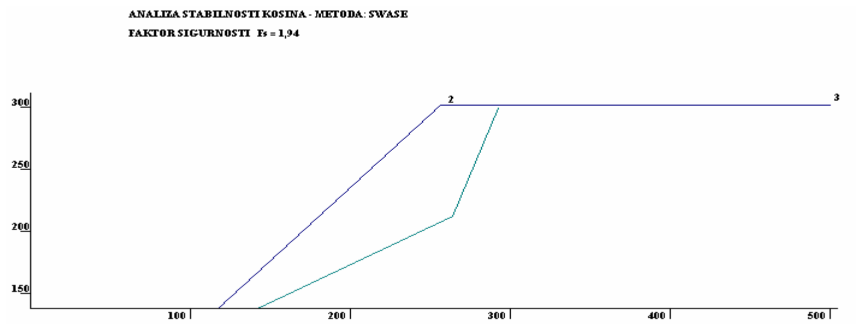


Figure 8. *Presentation of sliding plane 3*

Output list of calculation:

SET	DSBM	DSTP	DSME	LNS2	LNS	RU	SEIC	GAMMA	ANOUT
1	000	1.235	0.515	91.082	148.223	0.100	0.000	27.800	0.831

BW	F
257.000	**1.937**

SET	DSBM	DSTP	DSME	LNS2	LNS	RU	SEIC	GAMMA	ANOUT
2	0.305	1.603	0.727	31.016	48.166	0.100	0.000	27.800	0.831

BW	F
257.000	**2.933**

SET	DSBM	DSTP	DSME	LNS2	LNS	RU	SEIC	GAMMA	ANOUT
3	0.283	0.835	0.464	43.186	55.902	0.100	0.000	27.800	0.831

BW	F
257.000	**3.066**

4. ANALYSIS THE CALCULATION RESULTS

In this paper, the classic method was used, in other words the method of blocks – "Swase", that has taken into consideration the physical-mechanical properties of material, that is parameters of shear, provided in laboratory and reduced based on classification of rock masses in accordance with the geomechanical system. Calculation and analysis were done for each block. Blocks were modeled base on the engineering geological data and, based on the calculation results, it is obvious that safety factors are in the range that provides undisturbed work at the open pit. Calculation was done for designed condition, more exactly for the final slope angle of 53^0.

5. CONCLUDING DISCUSSIONS

Designed inclination of working and final slopes should be reviewed as variable parameters that are in the range of two extremes – something that is safe is not economical and vice versa. Those parameters are adjusted during exploitation to the real condition of rock massif. Such approach requires that during exploitation, there is sufficient quantity of quantitative data both on a rock massif and the results of certain operations at the open pit, in respect of stability. This means that parameters of blasting are adjusted to the type of rock masses and that final slopes are properly constructed. Using this software at stability analysis of slopes, there could also be considered influences of seismic effects on stability of slopes that occur during operations of massive blasting of rock materials as well as seismic effects that occur as the result of earthquakes.

REFERENCES

[1] S. Vujić; M. Berković; D. Kuzmanović; P. Milanović; A. Sedmak; M. Mičić, Use of MKE for Geostatic Calculations in Mining, University of Belgrade, 1990 (in Serbian)

[2] Yang H. Huang, Stability analysis of earth slopes, University of Kentucky, 1983.

[3] Undeground Excavations in Rock, Hoek & E.T.Brown, Institution of Mining and Metallurgy, London, 1980

[4] R. Milanović, Stability of Slopes of the Surface Mines in the Rock Masses, Copper Institute Bor (in Serbian)

[5] Slope Analysis, R.N.Chowdhury, Developments in Geotechnical Engineering, Vol. 22, 1978.

INSTITUT ZA RUDARSTVO I METALURGIJU BOR — YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA — UDK: 622

UDK: 622.83(045)=861

Milenko Ljubojev, Ratomir Popović*, Dragoslav Rakić***

RAZVOJ DINAMIČKIH POJAVA U STENSKOJ MASI***

Izvod

U radu je analiziran energetski bilans ugljenih naslaga, koji se sastoji od energije zarobljenog gasa u sloju, energije zarobljene u obrušenom materijalu i dela energije ostale u stenskoj masi.

Ključne reči: *energija gasa, energija u materijalu, energija u steni, energija rušenja stene, prirast kinetičke energije*

UVOD

Dinamičke pojave predstavljaju trenutno oslobađanje akumulirane mehaničke energije. Kod svih dinamičkih pojava, pomeranje stenske mase je brzinom do desetine metara u sekundi. Ako su takvim pomeranjima zahvaćeni veliki prostori stenskih masa, posledice takvog stanja mogu biti katastrofalne.

Povezujući pomeranje stenske mase reda desetine metara u sekundi sa zakonom o održanju energije, tako da pri γ = 20,0 [kN/m^3] akumulirana energija iznosi oko 10^5 [J/m^3]. Iz navedenog proizilazi opšti stav, da kod svih dinamičkih pojava oslobađa se velika mehanička energija po jedinici zapremine. Ukupna energija se sastoji od energije elastičnih deformacija i energija pritiska gasa zarobljenog u porama i šupljinama stenske mase. Ovde se mora naglasiti, da sve stene na određenoj dubini poseduju znatnu energiju. Gorski udari se javljaju samo u nekim stenama i pri specifičnim uslovima. Istraživanje tih uslova omogućuje rudarskim inženjerima da opasne i katastrofične situacije prevedu u neopasne uslove, tj. primenom odgovarajućih metoda prognoze.

U procesu dinamičkih pojava, energija raznih delova stenskog masiva menja se i međusobno se preraspoređuje. Ta izmena energije se ne odvija proizvoljno i u potpunosti je u saglasju sa zakonom o održanju energije, tako da je suma promena jednaka nuli. Pratiti u detalje put tih promena je veoma otežano. Sa praktične (pragmatične) tačke gledišta, preobražaj

* *Institut za rudarstvo i metalurgiju Bor*
** *Rudarski geološki fakultet Beograd*
*** *Ovaj rad je proistekao iz Projekta br. 33021 i 36014 koji je finansiran sredstvima Ministarstva za prosvetu i nauku Republike Srbije*

energije u detalje nije nužan. Pri ovim pojavama sasvim dovoljno je izučiti integralne karakteristike prelaza od početnog stanja ka stanju posle burnog rušenja. Pri tome je neophodno izjednačiti stanje sistema do i posle gubitka stabilnosti. Takvo integralno upoređivanje omogućuje izučavanje energetskog bilansa.

Iz njega proizilazi ocena kinetičke ene*-rgije, neophodne za preduzimanje odgovarajućih mera po upozorenju, ogra-ničavajući jačinu, a po mogućnosti i potpu-nom isključenju posledica od dinamičkih pojava.

1. ENERGETSKI BILANS

Izdvojena energija se sastoji iz dela energije širenja gasa W_g; dela zarobljene energije u obrušenom materijalu W_M i dela energije smeštene u steni $-\Delta э$. Navedeni bilans energije se troši na: energiju rušenja stene W_R; prirast kinetičke energije delovima obrušenog materijala ΔK, gubitak dela energije koju apsorbuju bokovi stene u neposrednoj blizini dinamičke pojave W_B i mali deo energije, do 10[%], troši se na oblikovanje seizmičkih talasa W_C, ostatak energije se troši na obrazovanje udarnih vazdušnih talasa W_V. Napred rečeno može se predstaviti sledećom jednačinom:

$$W_g + W_M + (-\Delta э) = W_R + \Delta K + W_B + W_C + W_V \quad (1)$$

Levi deo izraza (1) predstavlja izdvojenu energiju, a desni njenu apsorbciju.

1.1. Energija gasa Wg

Bilans te sumarne energije, neophodno je uzeti u obzir pri pojavi gasno-dinamičkih izboja. U procesu izboja rad može izvršiti samo slobodni gas. Imajući to u vidu i pri proračunu gasne energije W_g neophodno je saznanje o slobodnom gasu V_f u jedinici zapremine stene, i srazmerno dopunskoj količini gasa, koja je izdvojena, a zatim se širi pri desorbciji izazvanu padom pritiska. Važno je naglasiti da u stenama sa malom sorbcionom sposobnošću, poslednji efekat se može zanemariti.

Pri politropnom sa pokazateljem politropije χ_n, a pri širenju gasa V_0 od pritiska P do P_0, izdvaja se sledeća energija

$$W_{go} = \frac{P_a V_o}{\chi_n - 1}\left[1 - \left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right] \quad (2)$$

Pri adijabatskom procesu χ_n jednak je adijabati χ_g ; χ_g za metan iznosi 1,31.

Pri izotermičkom širenju pri $\chi_n = 1$ sledi:

$$W_{go} = P_a V_o \cdot \ln\frac{P}{P_a}$$

Razmatrajući vrednost energije W_f' slobodnog gasa V_f koji se sadrži u jedinici zapremine materije, preračun na normalne uslove provodi se na sledeći način:

$$\frac{V_n}{V_f} = \frac{P_a T}{P T_a} \quad (3)$$

gde je:

- V_n, zapremina šupljine (pore).

Uzimajući u obzir izraz (2) sledi:

$$W_f' = \frac{P_a W_f}{\chi_n - 1}\left[1 - \left(\frac{V_n}{V_f}\cdot\frac{T_a}{f}\right)^{1-\frac{1}{\chi_n}}\right] \quad (4)$$

Umesto V_f u (4) moguće je uvrstiti razliku sadržaja gasa u jedinici zapremine materije V_g i zapremine apsorbovanog gasa V_s, tj. V_g-V_s. Koristeći izraz (3) može se napisati:

$$W_f' = \frac{P V_n}{\chi_n - 1}\cdot\frac{T_a}{T}\left[1 - \left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right] \quad (5)$$

Pri adijabatskom procesu:

$$W_f' = \frac{P_a}{\chi_n - 1}\left(V_g - \frac{P a_s b_s}{1 + b_s P}\right)\left[1 - \left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right] \quad (6)$$

- a_s i b_s, konstante sorbcije, određuju se eksperimentalno za svaki materijal,

$a_s \approx 25-70\left[m^3/m^3\right]$ za ugljeve, tu vrednost Vs praktično dostiže pri 5 do 10 [MPa]. Veličina b_s za različite ugljeve se menja od 0,2 do 3,0 MPa^{-1}.

Pri izotermičkom širenju sledi:

$$W_f^{'} = P_a\left(V_g - \frac{Pa_s b_s}{1+b_s P}\right)\ln\frac{P}{P_a}$$

Iz navedenih izraza očigledno je da pri $V_g < a_s$ i pri velikim pritiscima moguće je da deo u maloj zagradi teži nuli, tada se istovremeno menja i energija $W_f^{'}$ Navedene jednačine ograničene su poroznošću materijala m. Tada minimalnu energiju W_f određuje sledeći izraz:

$$W_f^{'} = \frac{Pm}{\chi_n - 1}\cdot\frac{T_a}{T}\left[1-\left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right] \qquad (7)$$

Pri pritisku P = 2 [MPa] i poroznosti m = 0,08, izdvojena minimalna energija za metan iznosi:

$$W_f^{'} = 0{,}26\cdot 10^6\left[J/m^3\right]$$

a pri P = 5 [MPa] i m = 0,08, minimalna energija je: $W_f^{'} = 0{,}78\cdot 10^6\left[J/m^3\right]$.

Za nesorbirajuće stene $a_s = 0$ i $V_n = m$ izraz (5) poprima oblik izraza (7).

Pri adijabatskom procesu i padu pritiska od P do P_a sledi:

$$W_s^{'} = \frac{P_a\cdot a_s b_s}{\chi_n - 1}\int_{P_a}^{P} k_d\left[1-\left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right]\frac{d_p}{(1+b_s P_T)} \qquad (8)$$

Veličina $W_s^{'}$ teži nuli pri $b_s = 0$ i $b_s = \infty$, tj. kako pri usporenom, tako i pri vrlo brzom dostizanju granične sorbcije s porastom pritiska gasa.

Iz izraza (8) proizilazi:

$$\frac{W_s^{'}}{k_d\cdot a_s} \approx P_a\cdot P\cdot b_s$$

pri $Pb_s = 1,\ k_d = 0,\ a_s = 40\left[m^3/m^3\right]$, $P_a = 0{,}1[MPa]$, tada je minimalna izdvojena energija gasa $W_s^{'} = 0{,}4\cdot 10^6\left[J/m^3\right]$.

Opšta energija gasa je:

$$W_g = \left(W_f^{'} + W_s^{'}\right)\cdot V_p \qquad (9)$$

- $W_f^{'}$ i $W_s^{'}$, određuju izrazi (6) i (8)
- V_p, zapremina izbačenog obrušenog materijala.

1.2. Energija elastičnih deformacija obrušenog materijala W_M

Energija elastičnih deformacija jedinice zapremine materije pri malim deformacijama izražava se na sledeći način:

$$E_e = \int_0^{-e_{ij1}} \sigma_{ij1}\cdot d\varepsilon_{eij1} \qquad (10)$$

gde je:

- σ_{ij1}, tenzor napona,
- ε_{eij1}, povratna (elastična) tenzorska deformacija.

Izraz (10) pretpostavlja nelinearnu zavisnost između napona i elastične deformacije.

U slučaju linearne veze između σ_{ij1} i ε_{eij1}, tada je:

$$E_e = \frac{1}{2}\sigma_{ij1}\cdot\varepsilon_{eij1}$$

Za izotropni materijal, zavisnost elastičnih deformacija od napona u koordinatnom sistemu XOYZ je:

$$\varepsilon_{ex_1} = \frac{1}{E}\left[\sigma_{x_1} - \nu\left(\sigma_{y_1} + \sigma_{z_1}\right)\right]$$

$$\varepsilon_{exy_1} = \frac{1+\nu}{E}\sigma_{xy_1}$$

Izraz (10), tj. energija jedinice zapremine materije se može napisati u sledećem obliku:

$$E_e = \frac{1}{2E}\Big[\sigma_{x_1}^2 + \sigma_{y_1}^2 + \sigma_{z_1}^2 - 2\nu \cdot$$

$$\cdot\left(\sigma_{x_1}\sigma_{y_1} + \sigma_{x_1}\sigma_{z_1} + \sigma_{y_1}\sigma_{z_1}\right) + \quad (11)$$

$$+2(1+\nu)\left(\sigma_{xy_1}^2 + \sigma_{xz_1}^2 + \sigma_{yz_1}^2\right)\Big]$$

- ν, koeficijent Poisson-a,
- E, modul elastičnosti obrušenog materijala

Energija elastične deformacije W_M je: $W_M = \int_{V_M} \varepsilon_e \cdot dV$, i sa uzimanjem u obzir izraza (10) dobije se:

$$W_M = \int_{V_M}\left(\int_0^{\varepsilon_{ij1}} \sigma_{ij1} \cdot d\varepsilon_{eij1}\right) \cdot dV$$

Ako zapremina V_M predstavlja deo sloja s moćnošću 2h i sa površinom S_M dobije se:

$$W_M = \int_{S_M}\left(\int_{-h}^{h} E_e \cdot dy\right) \cdot ds \qquad (12)$$

Razmatrajući slučaj, kada se normalni naponi $\sigma_{x_1}, \sigma_{y_1}$ i σ_{z_1} malo menjaju duž moćnosti sloja, a smičući naponi σ_{xy_1} menja se linearno po $y\left(\sigma_{xy_1} = \frac{\tau_y}{h}\right)$, a $\sigma_{yz} = \sigma_{xz} = 0$.

Korišćenjem izraza (12) i s uzimanjem u obzir izraza (11), dobije se:

$$W_M = \frac{h}{E}\int_{S_M}\Big[\sigma_{x_1}^2 + \sigma_{y_1}^2 + \sigma_{z_1}^2 -$$

$$-2\nu\left(\sigma_{x_1}\sigma_{y_1} + \sigma_{x_1}\sigma_{z_1} + \sigma_{y_1}\sigma_{z_1}\right)\Big] \cdot$$

$$\cdot ds + \frac{2}{3}\frac{1+\nu}{E} \cdot h \cdot \tau \cdot S_M$$

Proračun energije u zapremini susedne prostorije duž dela G njene zapremine i ograničene površine na odstojanju $\xi(G)$ od udaljenosti

$$W_M = \frac{h}{E}\int_G dG \int_0^{\xi(G)}\Big[\sigma_{x_1}^2 + \sigma_{y_1}^2 + \sigma_{z_1}^2 -$$

$$-2\nu\left(\sigma_{x_1}\sigma_{y_1} + \sigma_{x_1}\sigma_{z_1} + \sigma_{y_1}\sigma_{z_1}\right)\Big]$$

$$\cdot d\xi + \frac{2}{3}\frac{1+\nu}{E} \cdot h \cdot \tau^2 \cdot S_M \qquad (13)$$

Za uslove ravnijske deformacije raspodela napona do tačke maksimuma oslonačkog pritiska pri $\xi \le a$,

- a, rastojanje od čela radilišta do maksimuma oslonačkog pritiska.

Energija elastične deformacije W_{M_1} za površ širine 1 [m] određuje se sledećim izrazom:

$$W_{M_1} = \frac{1}{E}\sigma_{kub.}^2 \xi \cdot h \cdot$$

$$\cdot\left[\frac{23}{12} - \frac{1}{3}\nu + (1-2\nu)\frac{\xi}{h}\left(\frac{3}{2} + \frac{\xi}{h}\right)\right] \quad (14)$$

pri $\sigma_{kub.} = 7{,}5\,[MPa]$; $\xi = 1{,}0\,[m]$; $2h = 2{,}0\,[m]$, $\nu = 0{,}4$; $E = 10^3[MPa]$, tada je:

$$W_{M_1} = 114.000\left[\frac{J}{m}\right]$$

U većim slučajevima pri određivanju energije u materijalu do rušenja predstavlja imperativ izračunavanja njene zapremine koja je podvrgnuta intenzivnim nepovratnim deformacijama, tj. deo sloja koji je u granično naponskoj zoni. Pri $\xi = a$ i korišćenjem (14) i (13) dobije se:

$$W_{M_1} = 0{,}96\frac{\sqrt{\sigma_{kub.}^2 h^5 k_1^2}}{E} f_a(b) \cdot$$

$$\cdot\left\{\frac{23}{12} - \frac{1}{3}\nu + 0{,}96(1-2\nu) f_a(b) \cdot\right.$$

$$\cdot\sqrt[3]{\frac{k_1^2}{\sigma_{kub.}^2 \cdot h}}\left[\frac{3}{2} + 0{,}96 \cdot\right. \qquad (15)$$

$$\left.\left.\cdot\sqrt[3]{\frac{k_1^2}{\sigma_{kub.}^2 \cdot h}} f_a(b)\right]\right\}$$

Za približnu ocenu energije u površi jedinične dužine moguće je usvojiti $f_a(b) \approx 1$ i tada je:

$$W_{M_1} \approx 0{,}91(1-2\nu)\frac{k_1^2}{E} \cdot h \qquad (16)$$

Za podzemnu prostoriju dužine 2l = 200 [m], širine $2x_0$ = 100 [m], na dubini od 600

[m] u sloju moćnosti 2h = 2,0 [m]; ν = 0,4; E = 10^3 [MPa], pri tome je:

$|k_1| = 1{,}84 \cdot 10^8 \left[N/m^{3/2} \right]$ i

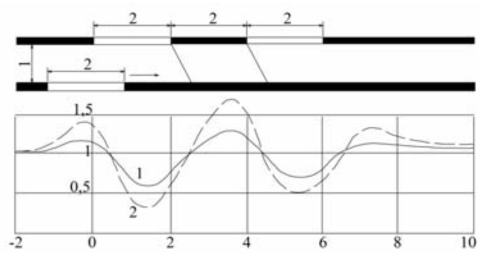
$W_{M_1} = 6 \cdot 10^6 [J/m]$

Puna energija sadržana u granično naponskoj zoni sloja određuje se na sledeći način:

$$W_M = \int_G W_{M_1} \cdot dG$$

Prethodni izraz spojen sa izrazima (14) i (16) koristi se za definisanje energetskog bilansa, izraz (1).

Kako proizilazi iz navedenih izraza, energija W_M raste s porastom čvrstoće i moćnosti sloja i s porastom koeficijenta intenziteta napona i sa smanjenjem modula elastičnosti.

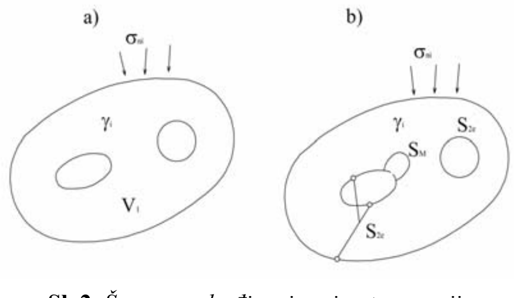


Sl. 1. *Promena koeficijenta intenzivnosti napona i energije pri eksploataciji dela sloja sa rasponom jednak dva*

$$\text{-1: } \frac{k_1}{k_1^{'}} \text{; -2: } \frac{W_{M_1}}{W_{M_1}^{'}} \text{; } \frac{W_{M_1}}{W_{M_1}^{'}} = \frac{k_1^2}{k_1^{'2}}$$

1.2.1. Prirast energije sadržane u stenskoj masi

Razmotrimo proizvoljnu zapreminu V' stenskog masiva sa prostorijama, sl. 2.a. Sada jedna od njih povećava svoju zapreminu na V_P, tako da nova zapremina stenskog masiva je V'', sl. 2.b. Površinu zapremine V'' ćemo predstaviti u vidu sume površina S_2, zadržavajući bez izmene novu površinu S. Polazno stanje odgovara sl. 2.a. Pomeranje, deformacije i naponi su obeleženi indeksom jedan U_{i1}, ε_{ij1} i σ_{ij1}. Novom stanju, sl. 2.b., odgovaraju priraštaji ΔU_i, $\Delta\varepsilon_{ij}$ i $\Delta\sigma_{ij}$, tako da su nove komponente povezane na sledeći način:



Sl. 2. *Šema za određivanje prirasta energije*

- a, telo u početnom stanju

- b, telo posle povećanja zapremine

$$\left.\begin{aligned} U_{i2} &= U_{i_1} + \Delta U_i \\ \varepsilon_{ij2} &= \varepsilon_{ij_1} + \Delta\varepsilon_{ij} \\ \sigma_{ij2} &= \sigma_{ij_1} + \Delta\sigma_{ij} \end{aligned}\right\} \qquad (17)$$

Prirast energije $-\Delta$ э preko površine S pri povećanju zapremine prostorije na ΔV jednak je razlici između priraštaja rada ΔA spoljnih sila u zapremini V'' i na površini S_{2e} i priraštajem ΔU unutarnje energije zapremine V''.

Iz napred navedenog proizilazi:

$$\Delta A = \int_{V''} \gamma_i \Delta U_i dV + \int_{S_{2e}} \left(\int_{U_{i1}}^{U_{i2}} \sigma_{ni} dy_i \right) dS$$

$$\Delta U = \int_{V''} \left(\int_{S_{ij1}}^{S_{ij2}} \sigma_{ij} d\varepsilon_{ij} \right) dV$$

gde je:

- γ_i, odgovarajući vektor zapreminske sile u elementarnoj zapremini

Tada je:

$$-\Delta \text{э} = \int_{S_{2e}} \left(\int_{U_{i_1}}^{U_i} \sigma_{ni} \cdot dU_i \right) dS + \int_{V''} \gamma_i \Delta U_i dV - \int_{V''} \left(\int_{\varepsilon_{ij_1}}^{\varepsilon_{ij2}} \sigma_{ij} \cdot d\varepsilon_{ij} \right) dV \qquad (18)$$

Koristeći izraz (17) i Gausa i Ostrogorskog i jednačina ravnoteže ima sledeći oblik

$$\int_{S_{2e+S*}} \Delta U_i ds + \int_{V''} \gamma \Delta U_i dV - \int_{V''} \sigma_{ij_2} \Delta\varepsilon_{ij} dV = 0 \qquad (19)$$

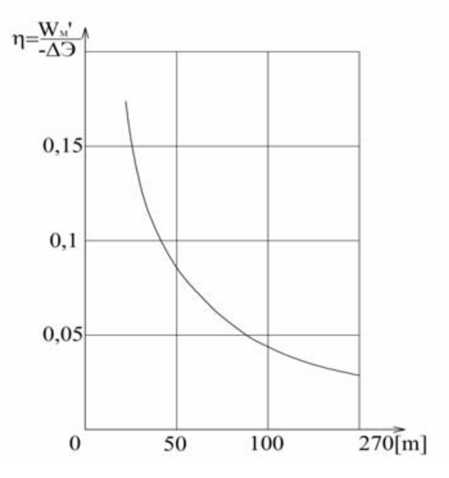
Oduzimanjem levog dela izraza (19) od desnog izraza (18), dobija se

$$-\Delta \text{э} = \int_{S_{2e}} \left(\int_{U_{i_1}}^{U_{i2}} \Delta\sigma_{ni} \cdot dU_i \right) dS - \int_{V''} \left(\int_{\varepsilon_{ij_1}}^{\varepsilon_{ij2}} \Delta\sigma_{ij} \cdot d\varepsilon_{ij} \right) dV - \int_{S*} \sigma_{ni_1} \cdot \Delta U_i dS$$

Preobražaj zapreminskog integrala u površinski daje osnovni izraz prirasta energije

$$-\Delta \text{э} = \int_{S*} \left[\int_{U_{i_1}}^{U_{i2}} \sigma_{ni} \cdot dU_i \right] dS \qquad (20)$$

Izraz (20) ne uključuje izmenu unutarnje energije zapremine V_P. Ta izmena predstavlja rad utrošen na rušenje W_R.



Sl. 3. *Odnos energije elastične deformacije W_M prema unutrašnjoj energiji $-\Delta$ э pri dinamičkim pojavama u zoni graničnog stanja*

Brojčane vrednosti $W_1 W_M \cdot (-\Delta э)$ za sledeće parametre:

$E = 2 \cdot 10^4\,[MPa]$; $\nu = 0{,}25$; $\sigma_{kub} = 10\,[MPa]$

$H = 600\,[m]$; $2h = 2\,[m]$; $\xi = 1{,}0\,[m]$.

Za raspon prostorije od $2X_0 = 100\,[m]$.

tada je $-\Delta э = 1{,}6 \cdot 10^6\,[J/m]$, što je skoro 15 puta više od W_M. U zaštićenoj zoni na jedinicu dužine prostorije $\frac{-\Delta э}{\Delta S_1}$ ostvaruje se energija $5 \div 7 \cdot 10^5 \left[J/m^2\right]$, a pod oslona-čkim stubom dostiže vrednost $2{,}7 \cdot 10^6 \left[J/m^2\right]$, tj. vrlo blisko ostvarenoj energiji gasa za sloj moćnosti od 1 [m], koji sadrži gas pod pritiskom većim od 1 [MPa].

1.2.2. Energija utrošena na rušenje W_R

Ovaj vid energetskog bilansa pri gorskom udaru i izboju gasa učestvuje u intenzivnom drobljenju materijala pri izboju. Energija utrošena na rušenje pri dinamičkim pojavama a pri aktivnom učešću i gasa određuje se na sledeći način:

$$W_R = g \cdot S_R \qquad (21)$$

- g, efektivna površinska energija,
- S_R, sumarna površina čestica obrušenog materijala.

Za gorske udare, energija rušenja određena je sledećom zavisnošću:

$$W_R = 2g_0 \cdot \Delta S_1 \qquad (22)$$

- $2g_0$, apsorbovana energija po jedinici preseka obrušenog materijala,
- ΔS_1, povećanje površine podine podzemne prostorije.

Veličina $2g_0$ za ugljeve sklone gor-skom udaru je od 0,31 do $1{,}0 \cdot 10^6$ [J/m^2].

1.2.3. Kinetička energija

Pri aktuelnom gorskom udaru, kinetička energija se određuje na bazi prosečne udaljenosti S_r odbačenog obrušenog materijala na horizontalnu podlogu. Srednje vreme pada obrušenog odbačenog materijala je:

$$t_r = \sqrt{2hg_T}$$

gde je:

- g_T, ubrzanje pri slobodnom padu

Pretpostavka je da je pomeranje posle pada obrušenog materijala na podlogu znatno usporeno.

$$S_r = v_r \sqrt{2h \cdot g_T} \qquad (23)$$

$$v_r = \frac{S_r}{\sqrt{2hg_T}} \qquad (24)$$

Prosečna vrednost kinetičke energije je:

$$\Delta K = \rho_1 \cdot V_r \frac{v_r^2}{2} \qquad (25)$$

- $\rho_1 = g_T \cdot \gamma$, gustoća obrušenog materijala

Korišćenje izraza (24) i (25) efektno je samo posle gorskog udara. Maksimalna brzina V_{max} i maksimalna udaljenost odbačenog materijala S_{rmax} proizašli iz dinamičke pojave, određuju se preko potencijalne energije, koja prethodi kinetičkoj zanemarujući energetski gubitak.

$$v_{r\max} = \sqrt{2\frac{W_M + (-\Delta э)}{\rho_1 \cdot V_r}}$$

U prethodnom izrazu mora se odrediti $(-\Delta э)$. Tada pri rušenju sloja moćnosti 2h, maksimalna brzina je:

$$v_{r\max} = \sqrt{-\left(\frac{d\,э}{dS_1}\right) \cdot \frac{1}{\rho_1 \cdot h}}$$

Koristeći sledeće izraze određuje se $v_{r\max}$:

$$-\frac{d\,э}{dS} \approx \frac{1-\nu^2}{E_1} - k_1^2$$

$$-\Delta э \approx \frac{1-\nu^2}{E_1} \int_{\Delta S_1} k_1^2\, dS$$

$$v_{r\max} = |k_1| \sqrt{\frac{1-\nu_1^2}{E_1 \cdot \rho_1 \cdot h}}$$

Pri $|k_1| = 2 \cdot 10^3 \left[N/m^{3/2}\right]$;

$\rho_1 = 15 \left[kN/m^3\right]$; $2h = 2\,[m]$; $\nu = 0{,}25$;

$E_1 = 2 \cdot 10^4\,[MPa]$, tada je

$v_{max} = 35\,[m/s]$, a koristeći izraz (23), maksimalna udaljenost odbačenog obrušenog materijala je: $S_{max} = 16$ [m].

Osnovna pretpostavka zasnovana je na tome da pri izboju suma $W_B + W_C + W_V$ je vrlo mala u poređenju sa W_r i ΔK. Osnovni deo delujuće energije $W_g + W_M + +(-\Delta э)$ troši se na rušenje i predaju kinetičku energiju obrušenim česticama.

Prirast kinetičke energije ΔK određuje sledeći izraz:

$$\Delta K \approx W_g + (-\Delta э) + W_M - W_r \quad (26)$$

Ako je desni deo izraza (26) manji od nule, tj. $W_g + W_M + (-\Delta э) - W_r < 0$, tada raspoloživa energija nije dovoljna da izazove rušenje i odbacivanje obrušenog materijala.

LITERATURA

[1] Proskurlkov N. M., Fomina V. D., Rožkov V. K., Gazodinamičeskije javljenija na Soligorskih kalijnih rudnikah Minsk, Polimja, 1974.

[2] Petuhov I. M., Linjkov A. M., Mehanika gornih udarov i vibrosov Moskva, Njedra, 1983.

[3] Rabota E. N., Isledovanije naprjaženovo sostajanija masiva gornih parod i određivanja granične zone zašćišćenih od gornih udarov VNIMI, 1975.

[4] SYDS.PENG, Ph. D., Coal Mine Ground Control New York, 1990.

[5] Henrik Filcek, Zdistaw Kteczek, Andrzej Zorychta Pogladi i rozviazania dotyczace tapan w kopalnih wegla kamienmego Krakow, 1984.

[6] Milenko Ljubojev, Ratomir Popović, Mevludin Avdić, Lidija Đurđevac-Ignjatović, Vesna Ljubojev, Defining the legality of gray sandstone rock strength testing in a complex state of stress Tehnics Tehnologies Education Management, Published by DRUNPP, Sarajevo, Vol. 5, Number 3, 2010. ISSN 1840-1503

[7] Ratomir Popović, Milenko Ljubojev, Dragan Ignjatović, Lidija Đurđevac-Ignjatović Geomeghanical laboratory conditions of rock fracture Tehnics Tehnologies Education Management, Published by DRUNPP, Sarajevo, Vol.5, Number 3, 2010. ISSN 1840-1503

[8] R. Popović, M. Ljubojev, Osnove rušenja stena primenjenom mehanizacijom pri eksploataciji čvrstih mineralnih sirovina, Monografija, Bor, 2011.

[9] D. Petrović, Z. Damnjanović, D. Đenadić, R. Pantović, V. Milić, Primena modernih računarskih uređaja i alata za smanjenje akcidentnih situacija u rudarskim sistemima, Rudarski radovi, br. 2-2010, str. 29-34

MINING AND METALLURGY INSTITUTE BOR YU ISSN: 1451-0162
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS UDK: 622

UDK:622.83(045)=20

Milenko Ljubojev, Ratomir Popović*, Dragoslav Rakić***

DEVELOPMENT OF DYNAMIC PHENOMENA IN THE ROCK MASS***

Abstract

This paper analyzes the energy balance of coal deposits, which consists of energy the trapped gas in the layer, the energy trapped in the collapsed material and a part of present energy in the rock mass.

Key words: *gas energy, material energy, rock energy, energy of rock destruction, kinetic energy increase*

INTRODUCTION

Dynamical phenomena present the instantaneous release of accumulated mechanical energy. In all dynamic phenomena, movement of rock mass is with rate up to tens of meters per second. If such movements affect large areas of rock masses, the consequences of such a situation can be disastrous.

Linking the movement of rock masse of order tens of meters per second with the law of energy conservation, so that at γ = 20.0 [kN/m^3], the stored energy is about 105 [J/m^3]. The general statement follows, from the all mentioned, that in all dynamic phenomena, high mechanical energy per unit volume is released. The total energy consists of energy of elastic deformations and energy of gas pressure trapped in pores and cavities of the rock mass. It must be stressed here that all rocks at certain depth have substantial energy.

The rock bursts only occur in some rocks and under the specific conditions. Investigations of these conditions allow the mining engineers to transform the dangerous and catastrophic situations into non-hazardous conditions, i.e. using the appropriate methods of forecasting.

In the process of dynamic phenomena, the energy of various parts of rock mass is changed and redistributed at each other. This energy change does not take place arbitrarily, and it is fully compliant with the law of energy conservation, so that the sum of changes is zero. Detailed following the way of these changes is very difficult.

* *Mining and Metallurgy Institute Bor*
** *University of Belgrade, Faculty of Mining and Geology Serbia*
*** *This paper is produced from the project no. 33021 and 36014 which is funded by means of the Ministry of Education and Science of the Republic of Serbia*

From the practical (pragmatic) point of view, the transformation of energy into the details is not necessary. During these phenomena, it is quite enough to study the integral characteristics of transition from the initial state to state after a whirlwind of destruction. During this, it is necessary to equalize the system state before and after loss of stability. Such integral comparison allows the study of energy balance.

The assessment of kinetic energy results from it, that is necessary for taking the appropriate actions upon notice, limiting the volume, and preferably the full exclusion of consequences from dynamic phenomena.

1. ENERGY BALANCE

Extracted energy consists of a part of gas spreading energy Wg; a part of trapped enerfy in the collapsed material and WM and a part of energy stored in the rock - $\Delta э$. The above given energy balance is spent on: energy for rock collapse WR; increase of kinetic energy from parts of collapsed material ΔK, loss of a part of energy absorbed by the rock hips in the vicinity of dynamical phenomena WB and a small part of the energy, up to 10 [%], is spent on a formation of seismic waves WC, the rest of the energy is spent on formation the impact air waves WV. The above outlined can be present by the following equation:

$$W_g + W_M + (-\Delta э) = W_R + \Delta K + W_B + W_C + W_V \quad (1)$$

Left part of equation (1) presents the extracted energy, and the right - its absorption.

1.1. Gas energy Wg

It is necessary to take into account the balance of this summary energy during the occurrence of gas-dynamic discharge. In the process of discharge, work can be performed only by free gas. Having this in mind and in calculation the gas energy W_g, it is necessary to know about the free gas V_f in the unit of rock volume and, proportionally, supplementary amount of gas, which is separated, and then expands at desorption induced by the pressure drop. It is important to note that in the rocks with small sorption ability, the last effect can be neglected.

In polytrope with the indicator of polytropy χ_n, and at gas expansion V_0 from pressure P to P_0, the following energy is extracted.

$$W_{go} = \frac{P_a V_o}{\chi_n - 1}\left[1 - \left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right] \quad (2)$$

In the adiabatic process χ_n is equal to the adiabatic χ_g; χ_g; for methane is 1.31.

During the isothermal expansion $\chi_n = 1$ follows

$$W_{go} = P_a V_o \cdot \ln \frac{P}{P_a}$$

Considering the energy value W_f' of free gas V_f which contains in the volume unit of material, a pre-calculation of normal conditions is carried out as follows

$$\frac{V_n}{V_f} = \frac{P_a T}{P T_a} \quad (3)$$

where:

- V_n, pore volume

Taking into account the expression (2), it follows

$$W_f' = \frac{P_a W_f}{\chi_n - 1}\left[1 - \left(\frac{V_n}{V_f} \cdot \frac{T_a}{f}\right)^{1-\frac{1}{\chi_n}}\right] \quad (4)$$

Instead of V_f in (4), it is possible to include a difference of gas content in the volume unit of matter V_g and volume of absorbed gas V_s, i.e. V_g-V_s. Using the expression (3), it could be written

$$W_f' = \frac{P V_n}{\chi_n - 1} \cdot \frac{T_a}{T}\left[1 - \left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right] \quad (5)$$

In the adiabatic process:

$$W_f^{'} = \frac{P_a}{\chi_n - 1}\left(V_g - \frac{P a_s b_s}{1 + b_s P}\right)\left[1 - \left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right] \quad (6)$$

- a_s and b_s, sorption constants are determined experimentally for each material

$a_s \approx 25 - 70\left[m^3/m^3\right]$ for coal, this value Vs practically attains at 5 to 10 [MPa]. Value b_s for different types of coal is changed from 0.2 to 3.0 MPa^{-1}.

At isothermal expansion, it follows:

$$W_f^{'} = P_a\left(V_g - \frac{P a_s b_s}{1 + b_s P}\right)\ln\frac{P}{P_a}$$

From given expressions, it is obvious that at $V_g < a_s$ and high pressures it is possible that a part in a small bracket tends to zero, and then the energy $W_f^{'}$ is also changed simultaneously. The above equations are limited by the material porosity m. Then the minimum energy W_f is determined by the following expression

$$W_f^{'} = \frac{Pm}{\chi_n - 1}\cdot\frac{T_a}{T}\left[1 - \left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right] \quad (7)$$

At pressure P = 2 [MPa] and porosity m = 0.08, the extracted minimum energy for methane is:

$$W_f^{'} = 0{,}26\cdot 10^6\left[J/m^3\right]$$

And at P = 5 [MPa] and m = 0.08, minimum energy is: $W_f^{'} = 0{,}78\cdot 10^6\left[J/m^3\right]$

For nonsorbted rocks $a_s = 0$ and $V_n = m$ expression (5) takes the form of expression (7).

At adiabatic process and pressure drop from P to P_a, it follows:

$$W_s^{'} = \frac{P_a\cdot a_s b_s}{\chi_n - 1}\int_{P_a}^{P} k_d\left[1 - \left(\frac{P_a}{P}\right)^{1-\frac{1}{\chi_n}}\right]\frac{d_p}{(1 + b_s P_T)} \quad (8)$$

Value $W_s^{'}$ tends to zero at $b_s = 0$ and $b_s = \infty$, i.e. both slow and very fast reaching the limit sorption with increase the gas pressure.

From expression (8) it follows:

$$\frac{W_s^{'}}{k_d\cdot a_s} \approx P_a\cdot P\cdot b_s$$

at $Pb_s = 1,\ k_d = 0,\ a_s = 40\left[m^3/m^3\right]$, $P_a = 0{,}1[MPa]$, then the minimum extracted gas energy $W_s^{'} = 0{,}4\cdot 10^6\left[J/m^3\right]$

General gas energy is:

$$W_g = \left(W_f^{'} + W_s^{'}\right)\cdot V_p \quad (9)$$

- $W_f^{'}$ and $W_s^{'}$, determined by the expressions (6) and (8)
- V_p, volume of discharged caved material

1.2. Energy of elastic deformations of caved material W_M

Energy of elastic deformations of matter volume unit at low deformations is expressed in the following way:

$$E_e = \int_0^{-e_{ij1}} \sigma_{ij1}\cdot d\varepsilon_{eij1} \quad (10)$$

where:

- σ_{ij1}, stress tensor
- ε_{eij1}, return (elastic) deformation tensor

Expression (10) assumes nonlinear dependence between stress and elastic deformation.

In a case of linear link between σ_{ij1} and ε_{eij1}, then:

$$E_e = \frac{1}{2}\sigma_{ij1}\cdot\varepsilon_{eij1}$$

For isotropic material, the dependence of elastic deformations on stress in the coordinate system XOYZ is:

$$\varepsilon_{ex_1} = \frac{1}{E}\left[\sigma_{x_1} - \nu\left(\sigma_{y_1} + \sigma_{z_1}\right)\right]$$

$$\varepsilon_{exy_1} = \frac{1+\nu}{E}\sigma_{xy_1}$$

Expression (10), i.e. energy of matter volume unit can be written in the following form:

$$E_e = \frac{1}{2E}\Big[\sigma_{x_1}^2 + \sigma_{y_1}^2 + \sigma_{z_1}^2 - 2\nu \cdot \cdot\left(\sigma_{x_1}\sigma_{y_1} + \sigma_{x_1}\sigma_{z_1} + \sigma_{y_1}\sigma_{z_1}\right) + 2(1+\nu)\left(\sigma_{xy_1}^2 + \sigma_{xz_1}^2 + \sigma_{yz_1}^2\right)\Big] \quad (11)$$

- ν, the Poisson coefficient

- E, elasticity module of caved material

Elastic deformation energy W_M is: $W_M = \int_{V_M} \varepsilon_e \cdot dV$ with exception into account the expression (10), the following is obtained:

$$W_M = \int_{V_M}\left(\int_0^{\varepsilon_{ij1}} \sigma_{ij1} \cdot d\varepsilon_{eij1}\right) \cdot dV$$

If volume V_M presents a part of layer with thickness 2h and surface S_M, the following is obtained:

$$W_M = \int_{S_M}\left(\int_{-h}^{h} E_e \cdot dy\right) \cdot ds \quad (12)$$

Considering the case when the normal stresses $\sigma_{x_1}, \sigma_{y_1}$ and σ_{z_1} are changed slightly along layer thickness, and transverse stresses σ_{xy_1} it is changed linearly per $y\left(\sigma_{xy_1} = \frac{\tau y}{h}\right)$, and $\sigma_{yz} = \sigma_{xz} = 0$.

Using the expression (12) and taking into account the expression (11), the following is got:

$$W_M = \frac{h}{E}\int_{S_M}\Big[\sigma_{x_1}^2 + \sigma_{y_1}^2 + \sigma_{z_1}^2 - 2\nu\left(\sigma_{x_1}\sigma_{y_1} + \sigma_{x_1}\sigma_{z_1} + \sigma_{y_1}\sigma_{z_1}\right)\Big] \cdot \cdot ds + \frac{2}{3}\frac{1+\nu}{E} \cdot h \cdot \tau \cdot S_M$$

Energy calculation in a volume of next room along part G of its volume and limited surface at distance $\xi(G)$

$$W_M = \frac{h}{E}\int_G dG \int_0^{\xi(G)}\Big[\sigma_{x_1}^2 + \sigma_{y_1}^2 + \sigma_{z_1}^2 - 2\nu\left(\sigma_{x_1}\sigma_{y_1} + \sigma_{x_1}\sigma_{z_1} + \sigma_{y_1}\sigma_{z_1}\right)\Big] \cdot d\xi + \frac{2}{3}\frac{1+\nu}{E} \cdot h \cdot \tau^2 \cdot S_M \quad (13)$$

For conditions of flat deformation, stress distribution to the point of maximum pressure at $\xi \le a$

- a, distance from the forehead of operation site to the maximum pressure

Energy of elastic deformation W_{M_1} zfor surfaces 1 [m] is determined by the following expression

$$W_{M_1} = \frac{1}{E}\sigma_{kub.}^2 \xi \cdot h \cdot \cdot\left[\frac{23}{12} - \frac{1}{3}\nu + (1-2\nu)\frac{\xi}{h}\left(\frac{3}{2} + \frac{\xi}{h}\right)\right] \quad (14)$$

at $\sigma_{kub.} = 7{,}5\,[MPa]$; $\xi = 1{,}0\,[m]$; $2h = 2{,}0\,[m]$, $\nu = 0{,}4$; $E = 10^3\,[MPa]$, then, it is:

$$W_{M_1} = 114.000\left[\frac{J}{m}\right]$$

In the majority cases in determining the energy in material to the caving presents an imperative for calculation of its volume that undergone the intensive noreturnable deformations, i.e. a part of layer that is in the limit stress zone. At $\xi = a$ using (14) and (13) the following is obtained:

$$W_{M_1} = 0{,}96\frac{\sqrt{\sigma_{kub.}^2 h^5 k_1^2}}{E} f_a(b) \cdot \cdot\left\{\frac{23}{12} - \frac{1}{3}\nu + 0{,}96(1-2\nu) f_a(b) \cdot \cdot\sqrt[3]{\frac{k_1^2}{\sigma_{kub.}^2 \cdot h}}\left[\frac{3}{2} + 0{,}96 \cdot \cdot\sqrt[3]{\frac{k_1^2}{\sigma_{kub.}^2 \cdot h}} f_a(b)\right]\right\} \quad (15)$$

For an approximate evaluation of surface energy in the unit length, $f_a(b) \approx 1$ can be adopted and then:

$$W_{M_1} \approx 0{,}91(1-2\nu)\frac{k_1^2}{E} \cdot h \qquad (16)$$

For an underground room, length 2l = 200 [m], width $2x_0$ = 100 [m], at depth of 600 [m], in a layer of thickness 2h = 2.0 [m]; ν = 0.4; E = 10^3 [MPa], where it is:

$|k_1| = 1{,}84 \cdot 10^8 \left[N/m^{3/2}\right]$ and

$W_{M_1} = 6 \cdot 10^6 [J/m]$

Full energy contained in the limit stress zone is determined by the following way:

$$W_M = \int_G W_{M_1} \cdot dG$$

Previous expression linked with the expressions (14) and (16) is used to define the energy balance, expression (1).

As follows from the above expressions, W_M energy increases with increase the strength and thickness of layer and with increase the coefficient of stress intensity and decrease the elasticity module.

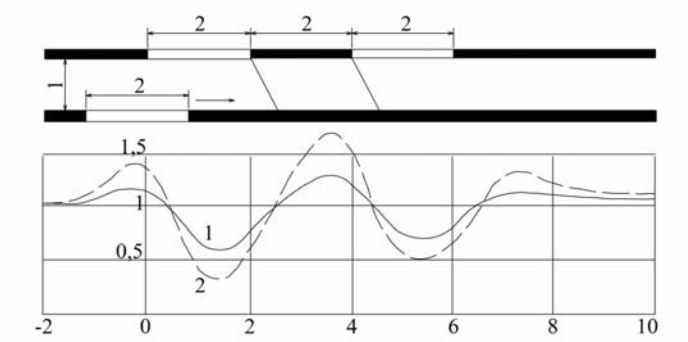


Figure 1. *Change the intensity coefficient of stress and energy in exploitation a part of layer in the range equal two*

$$\textbf{-1: } \frac{k_1}{k_1'}; \textbf{ -2: } \frac{W_{M_1}}{W'_{M_1}}; \ \frac{W_{M_1}}{W'_{M_1}} = \frac{k_1^2}{k_1'^2}$$

1.2.1. Energy increase contained in the rock mass

Consider an arbitrary volume V' of the rock massif with the rooms, Figure 2 a). Now, one of them increases its volume at V', so the new volume of rock massif is V'', Figure 2 b). Surface of volume V'' will be present in the form of sums of sufaces S_2, keeping without changing the new surface S. Initial condition corresponding to Figure 2 a). Displacement, deformation and stress are marked by the index one U_{i1}, ε_{ij1} and σ_{ij1}. To the new condition, Figure 2 b), corresponds increments of ΔU_i, $\Delta\varepsilon_{ij}$ and $\Delta\sigma_{ij}$, so that new components are linked as follows:

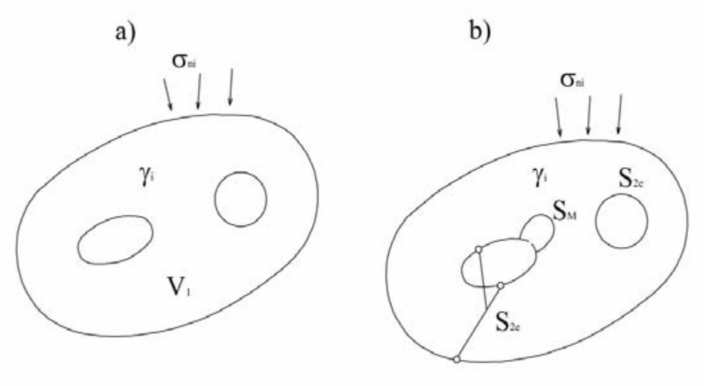


Figure 2. *Scheme for determining the energy increase*
- a) body in the initial condition
- b) body after volume increase

$$\left.\begin{aligned} U_{i2} &= U_{i_1} + \Delta U_i \\ \varepsilon_{ij2} &= \varepsilon_{ij_1} + \Delta\varepsilon_{ij} \\ \sigma_{ij2} &= \sigma_{ij_1} + \Delta\sigma_{ij} \end{aligned}\right\} \qquad (17)$$

Energy increase $-\Delta\,Э$ over the surface S at the increase of room volume in ΔV is equal to the difference between the growth of ΔA external forces in the volume V'' and on the surface S_{2e} and increase ΔU of internal volume energy V''.

From the above follows

$$\Delta A = \int_{V''} \gamma_i \Delta U_i dV + \int_{S_{2e}} \left(\int_{U_{i1}}^{U_{i2}} \sigma_{ni} dy_i \right) dS$$

$$\Delta U = \int_{V''} \left(\int_{S_{ij1}}^{S_{ij2}} \sigma_{ij} d\varepsilon_{ij} \right) dV$$

where:

- γ_i corresponding vector of volume force in the elementary volume

Then:

$$- \Delta\,Э = \int_{S_{2e}} \left(\int_{U_{i_1}}^{U_i} \sigma_{ni} \cdot dU_i \right) dS + \int_{V''} \gamma_i \Delta U_i dV - \int_{V''} \left(\int_{\varepsilon_{ij_1}}^{\varepsilon_{ij2}} \sigma_{ij} \cdot d\varepsilon_{ij} \right) dV \qquad (18)$$

Using the expression (17) and the Gaus and Ostrogorski, and equilibrium equitation has the following form

$$\int_{S_{2e+S*}} \Delta U_i ds + \int_{V''} \gamma \Delta U_i dV - \int_{V''} \sigma_{ij_2} \Delta\varepsilon_{ij} dV = 0 \qquad (19)$$

Subtracting the left part of expression (19) from the right expression (18), it is got

$$- \Delta\,Э = \int_{S_{2e}} \left(\int_{U_{i_1}}^{U_{i2}} \Delta\sigma_{ni} \cdot dU_i \right) dS - \int_{V''} \left(\int_{\varepsilon_{ij_1}}^{\varepsilon_{ij2}} \Delta\sigma_{ij} \cdot d\varepsilon_{ij} \right) dV - \int_{S*} \sigma_{ni_1} \cdot \Delta U_i dS$$

Transformation of volume integral into the surface gives the basic expression of energy increase

$$- \Delta\,Э = \int_{S*} \left[\int_{U_{i_1}}^{U_{i2}} \sigma_{ni} \cdot dU_i \right] dS \qquad (20)$$

Expression (20) does not include the internal energy change of volume V_P. This change represents the work spent on demolition of W_R.

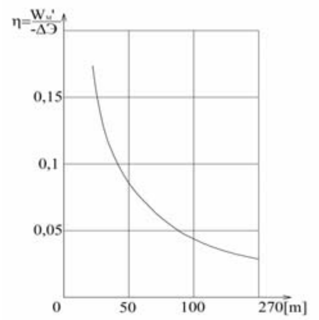


Figure 3. *The ratio of energy of elastic deformation WM according to the internal energy $-\Delta Э$ at the dynamic phenomena in the zone of limit conditions*

Numeric values $W_1 W_M \cdot (-\Delta Э))$ for the following parameters:
$E = 2 \cdot 10^4 \, [MPa]$; $\nu = 0,25$; $\sigma_{kub} = 10 \, [MPa]$
$H = 600 \, [m]$; $2h = 2 \, [m]$; $\xi = 1,0 [m]$.

For the room range of $2X_0 = 100 \, [m]$.

then $-\Delta Э = 1,6 \cdot 10^6 \, [J/m]$, what is almost 15 times higher than W_M. In the protected zone per unit length of room $\frac{-\Delta Э}{\Delta S_1}$, the energy is realized $5 \div 7 \cdot 10^5 \left[J/m^2\right]$ and under the support pillar reaches a value $2,7 \cdot 10^6 \left[J/m^2\right]$, i.e. very close to the realized gas energy for the layer thickness of 1 [m], which contains gas under pressure higher than 1 [MPa].

1.2.2. Energy spent on demolition W_R

This form of energy balance in the rock burst and gas discharge participate in intensive crushing of material at discharge. The spent energy in the destruction at the dynamic phenomena and the active gas participation is determined as follows:

$$W_R = g \cdot S_R \tag{21}$$

- g, effective surface energy
- S_R, summary surface of caved material particles

For the rock bursts, the demolition energy is determined by the following dependence:

$$W_R = 2g_0 \cdot \Delta S_1 \tag{22}$$

- $2g_0$, absorbed energy per section unit of caved material
- ΔS_1, surface increase of underground room floor

Value $2g_0$ for coal tends to the rock burst is from 0.31 to $1,0 \cdot 10^6$ [J/m^2].

1.2.3. Kinetic energy

In the current rock burst, the kinetic energy is based on the average distance S_r of rejected caved material on a horizontal surface. The average time of fall the rejected material is:

$$t_r = \sqrt{2hg_T}$$

where:

- g_T, acceleration in free fall

The assumption is that the movement after the fall of caved material on the surface is significantly slowed down.

$$S_r = v_r \sqrt{2h \cdot g_T} \tag{23}$$

$$v_r = \frac{S_r}{\sqrt{2hg_T}} \tag{24}$$

Average value of kinetic energy is:

$$\Delta K = \rho_1 \cdot V_r \frac{v_r^2}{2} \tag{25}$$

- $\rho_1 = g_T \cdot \gamma$, density of caved material

Using the expressions (24) and (25) is effective only after rock burst. Maximum rate V_{max} and maximum distance of rejected material S_{rmax} ,resulted from dynamic appearance, are determined by the potential energy, which is preceded by the neglecting kinetic energy loss.

$$v_{r\max} = \sqrt{2\frac{W_M + (-\Delta э)}{\rho_1 \cdot V_r}}$$

In the previous expression, it has to be determined $(-\Delta э)$. The, during demolition of the layer thickness 2h maximum rate is:

$$v_{r\max} = \sqrt{-\left(\frac{d\,э}{dS_1}\right) \cdot \frac{1}{\rho_1 \cdot h}}$$

Using the following expressions, $v_{r\max}$ is determined:

$$-\frac{d\,э}{dS} \approx \frac{1-v^2}{E_1} - k_1^2$$

$$-\Delta э \approx \frac{1-v^2}{E_1} \int_{\Delta S_1} k_1^2 dS$$

$$v_{r\max} = |k_1| \sqrt{\frac{1-v_1^2}{E_1 \cdot \rho_1 \cdot h}}$$

At $|k_1| = 2 \cdot 10^3 \left[N/m^{3/2}\right]$; $\rho_1 = 15 \left[kN/m^3\right]$; $2h = 2\,[m]$; $v = 0{,}25$; $E_1 = 2 \cdot 10^4\,[MPa]$, then $v_{\max} = 35\,[m/s]$, and using the expression (23), the maximum distance of rejected caved material is:
S_{max} = 16 [m].

The basic assumption is based on the fact that the discharge amount $W_B + W_C + W_V$ is very small compared to W_r and ΔK.The main part of the acting energy $W_g + W_M + (-\Delta э)$ is spent on demolition and transfer of kinetic energy to the collapsed particles.

Increase of kinetic energy ΔK determines the following expression:

$$\Delta K \approx W_g + (-\Delta э) + W_M - W_r \qquad (26)$$

If the right part of the expression (26) is less tha zero, i.e. $W_g + W_M + (-\Delta э) - W_r < 0$, then the existing energy is not sufficient to cause a demolition and rejection of the caved material.

REFERENCES

[1] Proskurlkov N. M., Fomina V. D., Rožkov V. K. Gazodinamičeskije javljenija na Soligorskih kalijnih rudnikah Minsk, Polimja, 1974.

[2] Petuhov I. M., Linjkov A. M. Mehanika gornih udarov i vibrosov Moskva, Njedra, 1983.

[3] Rabota E. N. Isledovanije naprjaženovo sostajanija masiva gornih parod i određivanja granične zone zašćišćenih od gornih udarov VNIMI, 1975.

[4] SYDS.PENG, Ph. D. Coal Mine Ground Control New York, 1990.

[5] Henrik Filcek, Zdistaw Kteczek, Andrzej Zorychta Pogladi i rozviazania dotyczace tapan w kopalnih wegla kamienmego Krakow, 198

[6] M. Ljubojev, R. Popović, M. Avdić, L. Dj. Ignjatović, V. Ljubojev, Defining the legality of gray sandstone rock strength testing in a complex state of stress Tehnics Tehnologies Education Management, Published by DRUNPP, Sarajevo, Vol.5, Number 3, 2010. ISSN 1840-1503

[7] R. Popović, M. Ljubojev, D. Ignjatović, L. Dj. Ignjatović Geomeghanical laboratory conditions of rock fracture Tehnics Tehnologies Education Management, Published by DRUNPP, Sarajevo, Vol.5, Number 3, 2010. ISSN 1840-1503

[8] R. Popović, M. Ljubojev, Basis of rock destruction by aplied mechanization in exploitation of solid mineral raw materials Monograph, Bor, 2011.

[9] D. Petrović, Z. Damnjanović, D. Đenadić, R. Pantović, V. Milić, Use of modern computer equipment and tools to reduce the occurrence of accidents in the mining sistems, Mining Engineering, No. 2-2010, pp. 35-40

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:622.274.4(045)=861

Ljubinko Savić, Radiša Janković**, Srđa Kovačević***

OTKOPAVANJE SIGURNOSNIH STUBOVA U RUDNIKU ''TREPČA'' – STARI TRG

Izvod

U podzemnoj eksploataciji u rudniku ''Trepča'' – Stari Trg, uglavnom se primenjivala metoda krovnog otkopavanja sa zasipavanjem otkopnih prostora. U primarnoj fazi eksploatacije ostavljani su sigurnosni stubovi (dimenzija 10 x 10 m) u šahovskom poretku. Ostavljeno je više od 70 sigurnosnih stubova. U sekundarnoj fazi eksploatacije planirano je otkopavanje sigurnosnih stubova u kojima se nalazi oko 15% neotkopanih rudnih masa od ukupnih rudnih rezervi.

U ovom radu prikazan je izbor parametara bušačko minerskih radova i način eksploatacije jednog sigurnosnog stuba, koji će poslužiti kao primer za eksploataciju ostalih sigurnosnih stubova.

Ključne reči: *podzemna eksploatacija, sigurnosni stubovi, parametri.*

UVOD

Ležište Trepča po svom lokalitetu pripada središnjem delu Vardarske tektomagmatske zone, čija se istočna granica u ovom delu terena zapaža na liniji Gnjilane – Kačendol – Šatorica, a zapadna na liniji Novi Pazar – Rogozna i dalje ka jugu gde se gubi pod tercijarnim basenom Kosova.

Ležište Trepča, sastoji se od niza cevastih rudnih tela nepravilnog oblika sa površinama koje se kreću od nekoliko stotina do 7000 m^2. Eksploatacija se uglavnom obavljala primenom metode krovnog otkopavanja sa zasipavanjem otkopnog prostora. Centralno rudno telo formirano je na kontaktu centralna breča – škriljac – krečnjak, koje je u otvorenom delu ležišta praćeno po dubini oko 1.100 m, gde mu se površina kreće od 4.000 do 7.000 m^2.

Najveći broj rudnih tela izgrađuju sulfidni minerali a manji deo ležišta čine tzv. oligonitna rudna tela, izgrađena od gvožđevito – magmatskih karbonata sa većim ili manjim sadržajem olovo – cinkanih sulfida.

Laboratorijskim ispitivanjem uzoraka, uzorkovanih u jami Trepča – Stari Trg, utvrđene su sledeće fizičko – mehaničke karakteristike rude i pratećih stena, koje su prikazane u tabeli 1.

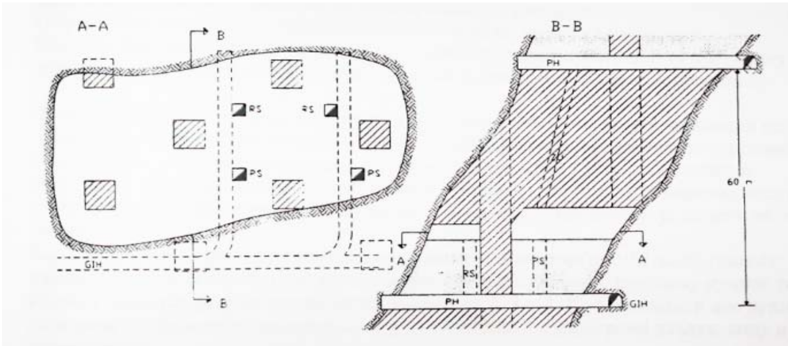
* *FTN Kosovska Mitrovica*
** *JPPK ''Kosovo'' Obilić*

Tabela 1. *Fizičko – mehaničke karakteristike rude i pratećih stena*

RUDA / STENA	γ_1 [t/m^3]	γ_2 [t/m^3]	σ_c MPa	σ_i MPa	**f**	**c MPa**	ν
Sulfidi	4,26	3,70	78,0	5,9	7,80	12,3	0,19
Oligoniti	3,67	3,48	82,1	7,4	7,43	13,6	0,19
Krečnjak	2,86	2,80	49,5	5,0	5,27	8,8	0,17
Škriljac	2,83	2,76	44,1	6,6	4,45	8,6	0,17
Breča	3,00	2,90	60,9	6,4	6,08	10,9	0,17

TREPČANSKA METODA OTKOPAVANJA

Na slici 1. prikazana je Trepčanska metoda otkopavanja.



Slika 1. *Trepčanska metoda otkopavanja*

Visinska razlika između horizonata iznosi 60 m. Na svakom horizontu u podinskom delu ležišta izrađuje se izvozni hodnik, a na svakih 30 m rade se prečni hodnici do krovine rudnog tela. Kada se ovim prečnim hodnicima dobiju tačne konture rudnog tela, koje su na višem horizontu poznate od ranije određuje se položaj sigurnosnih stubova. Sa nivoa horizonta počinje otkopavanje prve etaže čija visina iznosi 5,5 m. Bušenje kratkih minskih bušotina se vrši bušaćim čekićem RK – 28 ili VK – 30. Utovar rude se vrši direktno u vagonete sa CAVO utovarnom lopatom. Kada se otkopa ruda po celoj površini, delovi prečnog hodnika koji su bili u rudi izrađuju se u betonskoj oblozi, a ujedno se rade i rudne i rudno – prolazne sipke. Radi zasipavanja otkopa, izrađuju se u rudi zasipni uskopi do višeg horizonta. Sa nivoa višeg horizonta doprema se zasip koji se razastire po otkopu. Pod krovom otkopa ostavlja se otvorena visina od 2 m radi ventilacije i komuniciranja u otkopu. Otkopavanje sledeće etaže počinje od sipke prema krovinskom delu otkopa.

U jami rudnika Trepča – Stari Trg primenjuju se i sledeće metode: Uskopno frontalno otkopavanje i Magazinska metoda otkopavanja.

Metoda uskopnog frontalnog otkopavanja uglavnom se primenjuje za mala rudna tela površine do 50 m^2, sa uglom pada manjim od 40^o, pri čemu prateće stene moraju biti čvrste. Otkopavanje rudnog tela odvija se, odozdo na gore, po usponu (padu), u dve faze:

– Prva faza se sastoji u podsecanju rudnog tela na nivou horizonta, i
– Druga faza čini otkopavanje rudnog tela do gornjeg horizonta uz prethodnu pripremu objekata za prolaz ljudi, istakanje i transport rude i ventilaciju.

Razblaživanje kod ove metode je neznatno, a iskorišćenje rude i intenzitet otkopavanja su znatno veći nego kod Trepčanske metode.

Magazinskom metodom otkopavanja otkopan je veći broj manjih rudnih tela. Do sada su primenjivane dve varijante ove metode otkopavanja:

I varijanta: rudno telo se podseče na nivo horizonta, a zatim prva etaža zasipe i prelazi na magazinsko otkopavanje,

II varijanta: kada se sa postojeće Trepčanske metode prelazi na magazinsko otkopavanje.

Za rudna tela u rudniku Trepča, velikih površina i čiji je raspon od podinskog do krovinskog dela veliki, kao privremeno sredstvo osiguranja otkopa u primarnoj fazi eksploatacije, ostavjaju se sigurnosni stubovi raspoređeni u šahovskom poretku, dimenzija 10 x 10 m. Rastojanje između redova kreće se od 12 -16 m, a rastojanje između stubova u redu 16 -20 m. Sa sigurnošću se može konstatovati da u stubovima ostaje 15% rudne mase, koja će se eksploatisati u sekundarnoj fazi eksploatacije. Visina sigurnosnih stubova kreće se od 10 - 70 m, u zavisnosti od moćnosti rudnog tela. U ovim stubovima trenutno se nalaze rudne rezerve od oko 830.000 t, sa prosečnim sadržajem Pb – 6,75%, Zn – 3,76% i Ag – 206 g/t.

Na osnovu tehničke dokumentacije rudnika ''Trepča'' na četiri lokacije nalaze se 70 sigurnosnih stubova, za koje su proračunate rudne rezerve date u tabeli 2.

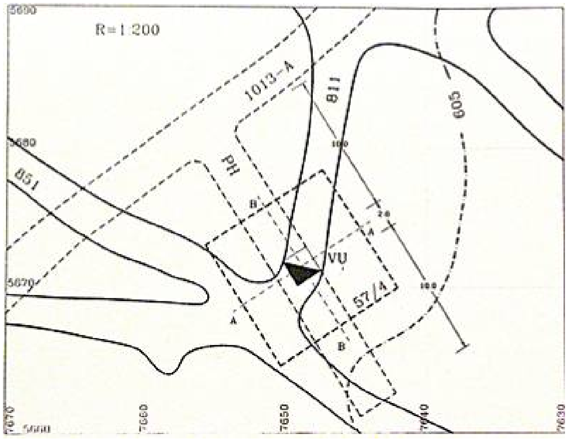
Tabela 2. *Prikaz rudnih rezervi u stubovima*

Redni broj	Lokacija stubova	Broj stubova	Rudne rezerve u stubovima [t]	Srednji sadržaj metala u rudi		
				Pb [%]	Zn [%]	Ag [g/t]
1.	Stari delovi	6	70.723	7.83	9.39	224
2.	Severno krilo jame	4	15.910	10.57	2.28	225
3.	Južno krilo jame	12	103.452	6.75	5.61	191
4.	Centralno rudno telo	48	639.982	6.75	2.86	205
Ukupno:		70	830.076	6.75	3.76	206

OTKOPAVANJE SIGURNOSNIH STUBOVA

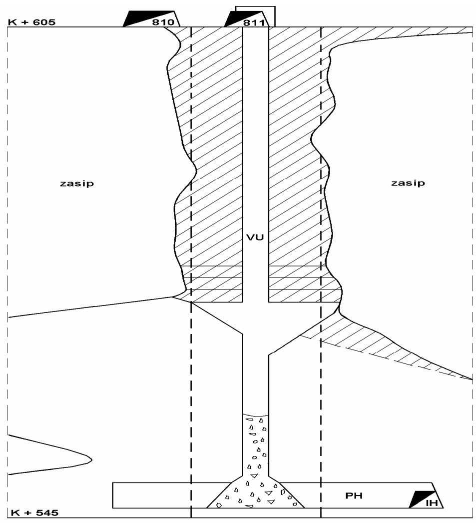
Uzimajući u obzir dosadašnje iskustvo pri otkopavanju sigurnosnih stubova i karakteristikama savremene bušaće opreme, razrađena je kombinovana uskopno magazinska – metoda otkopavanja sa masovnim obaranjem rude horizonralnim minskim bušotinama u lepezastom rasporedu.

Zbog velikog broja sigurnosnih stubova i sličnih geološko – tehničkih uslova eksploatacije, nije se smatralo celishodnim obrađivanje parametara bušačko minerskih radova svih sigurnosnih stubova pojedinačno, već je izabran jedan karakterističan stub (57/4), za koga su obrađeni parametri bušačko – minerskih radova i koji će služiti kao osnova za eksploataciju ostalih sigurnosnih stubova.



Slika 2. *Situacioni plan sa lokacijom stuba 57/4*

Po sredini sigurnosnog stuba prvo se pristupa izradi uskopa dimenzija 2 x 2 m. Na nivou nižeg horizonta vrši se podsecanje sigurnosnog stuba. Iz podsečenog dela počinje eksploatacija sigurnosnog stuba, bušenjem i miniranjem horizontalnih minskih bušotina u lepezastom rasporedu u punom krugu po visini sigurnosnog stuba. Time se omogućuje masovno miniranje većeg broja pojaseva miniranja, čime se reguliše željeni kapacitet eksploatacije. Nakon miniranja višak rude se tovari, a drugi deo rude služi kao podloga za naredno bušenje i miniranje. Sukcesivnim obaranjem rude i utovarom viška zapremine, eksploatiše se sigurnosni stub sve do nivoa višeg horizonta. Po završetku otkopavanja sigurnosnog stuba, vrši se pražnjenje rude i njegova zapremina koju je zauzimao ostaje prazna. Utvrđeno je da zasip i dalje ostaje da stoji čvrsto i ne zarušava se u prazan prostor, što je veoma značajno za otkopavanje narednih sigurnosnih stubova.



Slika 3. *Vertikalni presek sigurnosnog stuba sa lepezom minskih bušotina*

IZBOR VRSTE EKSPLOZIVA

Prilikom izbora vrste eksploziva potrebno je usaglasiti njegove eksplozivne karakteristike sa fizičko – mehaničkim karakteristikama stenske mase u kojoj se vrši miniranje. U konkretnim uslovima na otkopu izbor eksploziva se svodi na ANFO eksplozivne smeše koje se mehanizovanim postupkom mogu puniti u minske bušotine. Na osnovu izbora eksploziva i parametara miniranja usvaja se eksplozivna smeša ANFO – J1, iz proizvodnog programa DETONITA – Korporacije TRAYAL Kruševac.

Tabela 3. *Karakteristike eksplozivne smeše ANFO – J1*

Gustina [g/cm^3]	0,95 – 1,1
Brzina detonacije [m/s]	>2.000
Prenos detonacije	kontakt
Gasna zapremina [l/kg]	920
Toplota eksplozije [KJ/kg]	3.872
Osetljivost [g pentolita]	60
Kritični prečnik [mm]	30
Radna sposobnost [cm^3]	380
Stabilnost	6 meseci
Vodootpornost	slaba
Bilans kiseonika	uravnotežen

PRORAČUN PARAMETARA MINIRANJA

Dužina minskih bušotina, koji će se koristiti za otkopavanje sigurnosnih stubova, iznosi 4 – 6 m, u zavisnosti od poprečnog preseka sigurnosnog stuba, a proračun parametara miniranja vršiće se za prečnik minskih bušotina (d = 51 mm).

1. Specifična potrošnja eksploziva po Laresu:

$$q = q_1 \cdot \upsilon \cdot s \cdot \frac{e}{k_z} \cdot d \cdot k = 0{,}55\ [kg/m^3]$$

ili

$$q = \frac{q}{\gamma} = \frac{0{,}55\ [kg/m^3]}{3{,}7\ [t/m^3]} = 0{,}15\ [kg/m^3]$$

gde je:

q_1– koeficijent čvrstoće rude;

$$q_1 = \frac{\sigma_c}{2000} = 0{,}39$$

σ_c – jednoaksijalna pritisna čvrstoća rude (780 dN/cm^2);

υ – koeficijent stešnjenosti mina (1);

s – koeficijent sklopa stenske mase (0,9 - 1,1);

γ – zapreminska masa rude (3,7 t/m^3).

e – koeficijent relativne snage eksploziva;

$$e = \frac{480}{A_x} = 1{,}26$$

A_x – radna sposobnost eksploziva ANFO – J1 (380 cm^3);

k_z – koeficijent zbijenosti eksplozivnog punjenja (0,9);

d – koeficijent začepljenosti mina (1);

k – korekturni koeficijent zbijenosti eksplozivnog punjenja (1).

2. Linija najmanjeg otpora se izračunava:

$$W = 33 \cdot d_1 \cdot \sqrt{\frac{k_p}{q \cdot k_z}} = 1{,}9\ [m]$$

gde je:

d_1 – prečnik minske bušotine (51 mm);

k_p – koeficijent popunjenosti minske bušotine (0,7);

q – specifična potrošnja eksploziva (kg/m^3);

k_z – koeficijent zbližnjenja mina (1).

3. Rastojanje između bušotina u lepezi iznosi:

$$a_{min} = (0{,}5 \div 0{,}7) \cdot W = 0{,}95 \div 1{,}33\ [m]$$

$$a_{max} = (1{,}5 \div 1{,}7) \cdot W = 2{,}85 \div 3{,}23\ [m]$$

4. Masa rude čije se obaranje vrši jednom lepezom bušotina:

$$Q = (Ps - Pu) \cdot W \cdot \gamma = 675\ [t]$$

gde je:

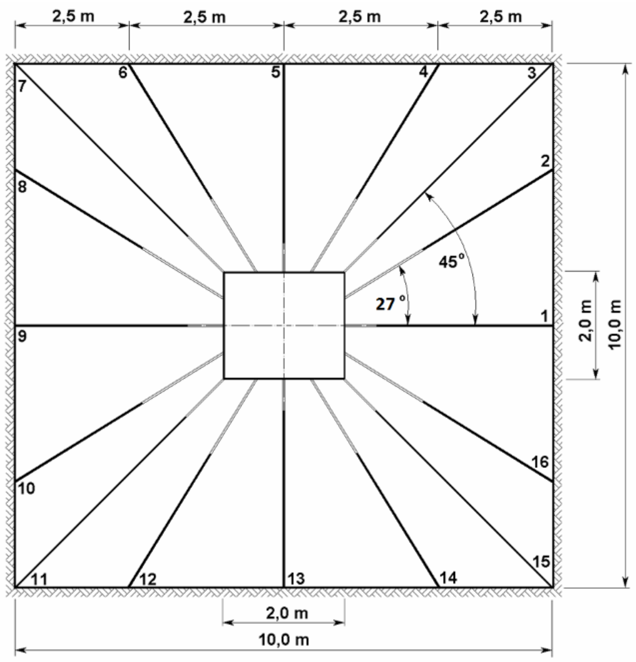
Ps – površina poprečnog preseka sigurnosnog stuba (m^2);

Pu – površina poprečnog preseka useka (m^2);

W – linija najmanjeg otpora (m).

5. Ukupna količina eksploziva koja se utroši za jedno miniranje lepeza bušotina:

$$Q_e = Q \cdot q = 675 \cdot 0{,}15 = 101{,}25\ [kg]$$



Slika 4. *Prikaz minskih bušotina sa lepezastim rasporedom*

Nakon izvršenog grafičkog rasporeda minskih bušotina izvršena je i njihova specifikacija koja je data u tabeli 4.

Tabela 4. Specifikacija minskih bušotina

Broj bušotine	Ugao [°]	Dužina [m]	Dužina punjenja [m]	Količina eksploziva [kg]
1	0	4,0	3,5	6,4
2	27	4,5	2,6	4,8
3	45	5,6	5,1	9,4
4	63	4,5	2,6	4,8
5	90	4,0	3,5	6,4
6	117	4,5	2,6	4,8
7	135	5,6	5,1	9,4
8	153	4,5	2,6	4,8
9	180	4,0	3,5	6,4
10	207	4,5	2,6	4,8
11	225	5,6	5,1	9,4
12	243	4,5	2,6	4,8
13	270	4,0	3,5	6,4
14	297	4,5	2,6	4,8
15	315	5,6	5,1	9,4
16	333	4,5	2,6	4,8
Ukupno:		74,4	55,2	101,6

6. U cilju ravnomerne raspodele energije eksploziva koristićemo različite koeficijente punjenja lepezastih bušotina koji iznose:
   - za minske bušotine 1, 5, 9 i 13

$$k_p = \frac{l_p}{l_b} = \frac{3,5}{4} = 0,88$$

   - za minske bušotine 2, 4, 6, 8, 10, 12 i 14

$$k_p = \frac{l_p}{l_b} = \frac{2,6}{4,5} = 0,58$$

   - za minske bušotine 3, 7, 11 i 15

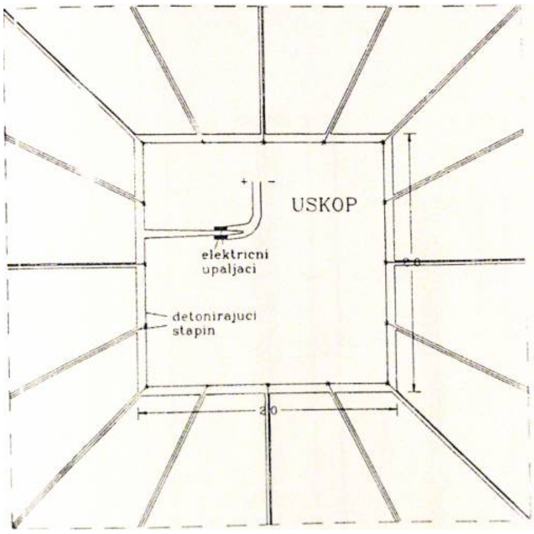
$$k_p = \frac{l_p}{l_b} = \frac{5}{5,6} = 0,88$$

7. Količina eksploziva za punjenje jedne minske bušotine:

$$q_e = \frac{\pi \cdot d^2}{4} \cdot l_p \cdot \rho \cdot k_z [kg]$$

8. Iniciranje minskih bušotina.

Na dno svake minske bušotine potrebno je ubaciti po dva pojačivača eksplozije (bustera), mase od po 80 gr. i povezati ih detonirajućim štapinom. Krajevi štapina treba da vire iz bušotine oko 0,5 m; radi povezivanja sa glavnim vodom detonirajućeg štapina. Šema vezivanja i iniciranja minskih punjenja prikazana je na slici 5.



Slika 5. *Šema vezivanja i iniciranja minskih bušotina*

Nakon smeštaja bustera i detonirajućeg štapina prelazi se na punjenje ANFO eksplozivne smeše pomoću pneumatske punilice.

Pošto se izvrši punjenje svih bušotina, vrši se povezivanje krajeva detonirajućeg štapina sa glavnim vodom. Oba kraja glavnog voda detonirajućeg štapina spajaju se na jednom mestu i tako se omogućuje prenos detonacije sa oba kraja. Za aktiviranje detonirajućeg štapina povezuju se dva električna detonatora radi sigurnijeg aktiviranja. Provodnici elektrodetonatora vezuju se na glavni električni kabal koji prolazi dužinom uskopa do sigurnog mesta za paljenje minskih bušotina.

ZAKLJUČAK

U radu je izvršen izbor i proračun parametara bušačko minerskih radova za otkopavanje sigurnosnog stuba (57/4), u rudniku ''Trepča'' – Stari Trg, gde se otkopavanje sigurnosnih stubova vrši bušenjem i miniranjem horizontalnih minskih bušotina u lepezastom rasporedu.

Nije se smatralo celishodnim određivanje parametara za sve sigurnosne stubove pojedinačno, već je izabran jedan karakterističan stub koji će služiti kao osnova za eksploataciju ostalih stubova.

LITERATURA

[1] P. Jovanović, Dimenzionisanje jamskih prostorija, Beograd, 1983.

[2] B. Gluščević, Otvaranje i metode podzemnog otkopavanja rudnih ležišta, Beograd, 1974.

[3] Lj. Savić, D. Đukanović, S. Krstić, Karakter odnosa između brzine bušenja i promene dužine bušotine, Podzemni radovi br. 12 (pp. 17-21) RGF Beograd, 2003.

[4] S. Trajković, V Nastić, S. Lutovac, Analiza postignutih rezultata pri izradi hodnika u rudniku "Rudnik" bušačko-minerskim radovima, Podzemni radovi br. 12 (pp. 01-10) RGF Beograd, 2003.

[5] P. Jovanović, D. Marković, S. Trajković, N. Vidanović, Studija procesa bušenja i miniranja na otkopima i hodnicima rudnika "Rudnik" RGF Beograd, 1985.

[6] Projektno tehnička dokumentacija rudnika ''Trepča'', Institut Zvečan.

MINING AND METALLURGY INSTITUTE BOR | YU ISSN: 1451-0162
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS | UDK: 622

UDK:622.274.4(045)=20

Ljubinko Savić[*], Radiša Janković[**], Srđa Kovačević[**]

MINING OF SAFETY PILLARS IN THE ''TREPCA'' - STARI TRG MINE

Abstract

In the underground mining in the "Trepca"- Stari Trg mine, the method of roof caving mining with back filling cavity was generally applied. In the primary stage of exploitation, the safety pillars (10 x 10 m) were left in a chess ranking. More than 70 safety pillars were left. Mining of the safety pillars, containing about 15% of non-mined ore masses of the total ore reserves, is planned in the secondary stage of mining operation.

This paper presents a selection of parameters for drilling and blasting works and the mining method of safety pillar, which will serve as an example for mining the other security pillars.

Key words: *underground mining, safety pillars, parameters*

INTRODUCTION

The Trepča deposit on its site belongs to the middle part of the Vardar tectonic-magmatic zone, whose eastern border in this part of the field is observed on the line Gnjilane - Kačendol - Šatorica, and the western border on the line Novi Pazar - Rogozna and further to the south where it loses under the Tertiary basin of Kosovo.

The Trepča deposit consists of a series the pipe ore bodies of irregular shape with surfaces that range from several hundred to 7,000 m^2. Exploitation was mainly carried out using the method of roof caving mining with back filling cavity. Central ore body was formed at the contact of the central breccia - shale - limestone, which is in the open part of deposit, followed by depth of about 1,100 m, where its area ranges from 4,000 to 7,000 m^2.

The largest number of ore bodies is built of sulphide minerals and smaller part of deposit consists of so called oligonite ore bodies, built of ferrous - magmatic carbonates with smaller or higher content of lead - zinc sulphides.

Laboratory testing of taken samples from the pit of mine Trepča - Stari Trg have confirmed the following physical - mechanical characteristics of the ore and associated rocks, shown in Table. 1.

[*] *FTN Kosovska Mitrovica*
[**] *JPPK ''Kosovo'' Obilić*

Table 1. *Physical - mechanical characteristics of ore and associated rocks*

ORE / ROCK	γ_1 [t/m^3]	γ_2 [t/m^3]	σ_c MPa	σ_i MPa	f	c MPa	ν
Sulfides	4.26	3.70	78.0	5.9	7.80	12.3	0.19
Oligonites	3.67	3.48	82.1	7.4	7.43	13.6	0.19
Limestone	2.86	2.80	49.5	5.0	5.27	8.8	0.17
Shale	2.83	2.76	44.1	6.6	4.45	8.6	0.17
Breccia	3.00	2.90	60.9	6.4	6.08	10.9	0.17

THE TREPČA MINING METHOD

The Trepča mining method is shown in Figure 1.

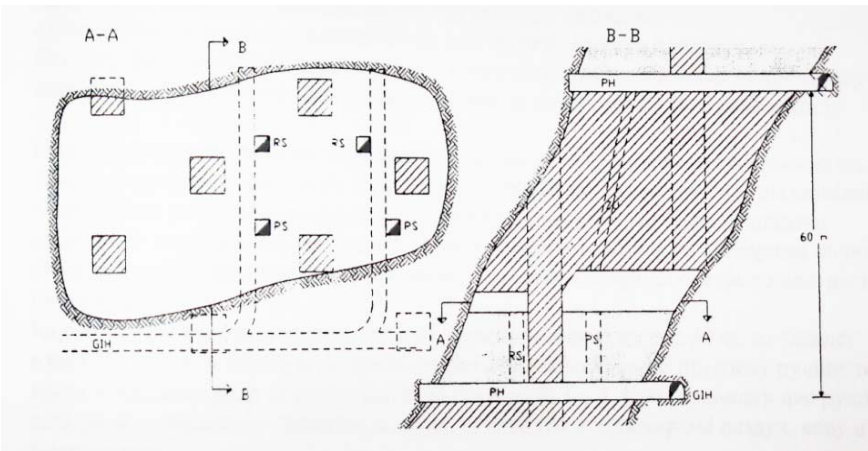


Figure 1. *The Trepča mining method*

Height difference between the horizons is 60 m. At each horizon in the foot wall of deposit, a haulage drift is driven, and the crosscut roadways are driven at every 30 feet to the roof of ore body. When the crosscut drifts give the accurate contours of the ore body, which are previously known at higher horizon, the position of safety pillars is determined. From the horizon level, the excavation of first bench starts, whose height is 5.5 m. Drilling of short blast holes is done using the drilling hammer RK - 28 or VK – 30. Loading of ore is carried out directly into wagons with CAVO loading shovel. When the ore is mined around the whole length, the parts of crosscut drift, which were in the ore, are made in concrete lining, and also the ore and ore - passing chutes. In order to backfill the stope, the stowing raises are made in the ore to higher horizon. From higher level of horizon, the stowing is delivered and distributed over stope. Under the hanging wall of stope, an open height of 2 m is left for ventilation and communication in the stope. Excavation of next bench begins from the chute towards the hanging wall of stope.

In the pit of mine Trepča - Stari Trg, the following methods are also applied: the raise frontal mining and block caving method.

The method of raise frontal mining is mainly applied for small ore bodies, area up to 50 m^2, with the angle of fall less than 40^o, where the associated rocks have to be solid. Excavation of ore body is carried out bottom-up per rise (fall), in two phases:

- The first phase consists of cutting the ore body at horizon level, and
- The second phase consists of excavation the ore body to the upper horizon with previous preparation of facilities for passage of people, unloading and transport of ore and ventilation.

Dilution in this method is slightly, and recovery of ore and mining intensity are significantly higher than in the Trepča method.

The block caving method of mining was used for excavation a large number of small ore bodies. Until now, two options of this mining method were used:

- I option: the ore body is cut at the horizon level, and then the first bench is stowed and the block caving is carried out,
- II option: when it is moved from the existing Trepča method to the block caving method.

For the ore bodies in the mine Trepča of large areas and wide wide range from the floor to roof, the safety pillars are left as a temporary means of stope ensuring in the primary phase of exploitation, in a chess order, size 10 x 10 m. The distance between the lines ranges from 12 -16 m, and the distance between pillars is in the order from 16 - 20 m. It could be concluded with certainity that 15% of ore mass is left in the pillars that will be exploited in the secondary phase of exploitation. Height of safety pillars varies from 10 - 70 m, depending on the ore body thickness. These pillars currently include the ore reserves of approximately 830,000 tons, with the average content of Pb – 6.75%, Zn – 3.76% and Ag – 206 g/t.

Based on the technical documentation of the mine “Trepča“, there are 70 safety pillars in four locations for which the ore reserves, given in Table 2, were calculated.

Table 2. *Review of ore reserves in pillars*

Order No.	Location of the pillars	Number of pillars	Ore reserves in pillars [t]	Mean metal content in the ore		
				Pb [%]	Zn [%]	Ag [g/t]
1	Old parts	6	70,723	7.83	9.39	224
2	North pit limb	4	15.910	10.57	2.28	225
3	South pit limb	12	103.452	6.75	5.61	191
4	Central ore body	48	639.982	6.75	2.86	205
	Total:	70	830.076	6.75	3.76	206

MINING OF SAFETY PILLARS

Taking into account the previous experience in mining the safety pillars and characteristics of modern drilling equipment, the combined block caving method with mass blowing down the ore by the horizontal blast holes in the fan-shaped arrangement.

Due to a large number of safety pillars and similar geological-technical conditions of exploitations, it was not considered as appropriate the processing of parameters of drilling-blasting works for all safety pillars individually, but a characteristic pillar (57/4) was selected for which the parameters of drilling – blasting works were processed, and that will serve as the base for exploitation the other pillars.

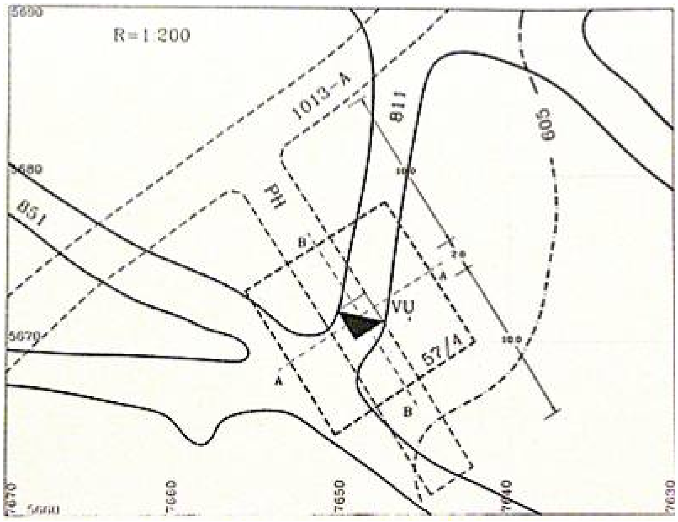


Figure 2. *Site plan with the location of pillar 57/4*

By middle of the safety pillar, the first approach is a raise driving, size 2 x 2 m. At the level of lower horizon, the safety pillar is cut. Exploitation of a cut part of safety pillar starts by drilling and blasting the horizontal blast holes in the fan-shaped arrangement in the full circle per height of safety pillar. This allows the mass blasting of a number of blasting belts, what regulates the desired capacity of exploitation. After blasting, the excess ore is loaded, and the second part of ore is used as the basis for subsequent drilling and blasting. By successive blowing down of ore and loading of excess capacity, the safety pillar is exploited down to the level of higher horizon. Upon completion of excavation the safety pillar, the ore is discharged and its volume is left empty. It was found that the stowing remains to stand firm and not caving in the empty space, what is very important for the excavation of next safety pillars.

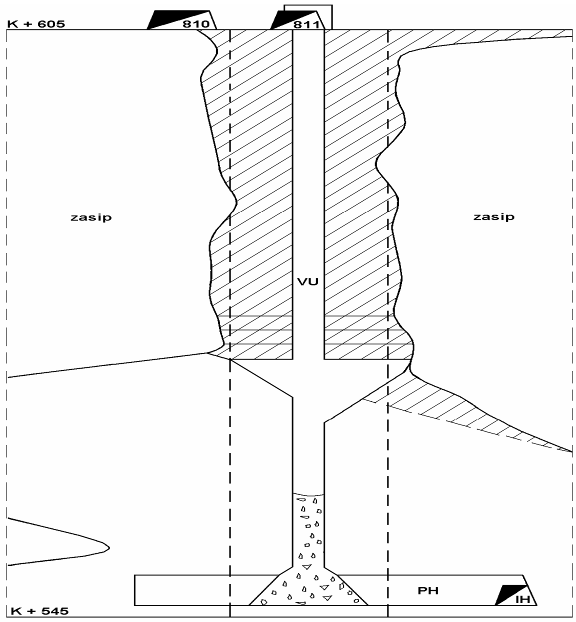


Figure 3. *Vertical section of the security pillar with a fan of blast holes*

SELECTION OF EXPLOSIVE TYPES

In the selection of explosive type, it is required to harmonize its explosive characteristics with physical - mechanical characteristics of rock mass where the blasting is carried out. In the specific terms, the selection of exsplosive at the stope comes down to the ANFO explosive mixtures which can be charged into the boreholes by mechanized method. Based on the selection of explosives and blasting parameters, the explosive mixture ANFO-J1 is adopted from the production program of DETONIT – TRAYAL Corporation, Kruševac.

Table 2. *Characteristics of the blastig mixture ANFO – J1*

Density [g/cm^3]	0.95 – 1.1
Detonation rate [m/s]	>2.000
Detonation transmission	contact
Gas volume [l/kg]	920
Blasting heat [KJ/kg]	3.872
Sensitivity [g pentolite]	60
Critical diameter [mm]	30
Working capacity [cm^3]	380
Stability	6 months
Waterproof	poor
Oxygen balance	balanced

CALCULATION OF BLASTING PARAMETERS

The length of blast holes, which will be used for excavation the safety pillars, is 4 - 6 m, depending on the cross section of safety pillar, and calculation of blasting parameters will be performed for the blast hole diameter (d = 51mm).

1. Specific consumption of explosive per Lares:

$$q = q_1 \cdot \upsilon \cdot s \cdot \frac{e}{k_z} \cdot d \cdot k = 0.55\ [kg/m^3] \text{ or}$$

$$q = \frac{q}{\gamma} = \frac{0.55\,[kg/m^3]}{3.7\,[t/m^3]} = 0.15\ [kg/m^3]$$

where:

q_1– coefficient of ore strength;

$$q_1 = \frac{\sigma_c}{2000} = 0.39$$

σ_c – uniaxial compressive strength of ore (780 dN/cm^2);

υ – coefficient of blast compactness (1);

s – coefficient of rock mass complex (0.9 - 1.1);

γ – volume mass of ore (3.7 t/m^3);

e – coefficient of relative strength of blasting agent;

$$e = \frac{480}{A_x} = 1.26$$

A_x – working capacity of blasting agent ANFO – J1 (380 cm^3);

k_z – coefficient of explosive charge compactness (0.9);

d – coefficient of blast stemming (1);

k – correction coefficient of explosive charge compactness (1).

2. The line of the least resistance is calculated:

$$W = 33 \cdot d_1 \cdot \sqrt{\frac{k_p}{q \cdot k_z}} = 1.9\,[m]$$

where:

d_1 – diameter of blast hole (51 mm);

k_p – charging coefficient of blast hole (0.7);

q – specific consumption of explosive (kg/m^3);

k_z – compactness coefficient of blasts (1).

3. Distance between blast holes in a fan is:

$$a_{\min} = (0.5 \div 0.7) \cdot W = 0.95 \div 1.33\,[m]$$

$$a_{\max} = (1.5 \div 1.7) \cdot W = 2.85 \div 3.23\,[m]$$

4. Ore mass that is blown down by one fan of the blast holes:

$$Q = (Ps - Pu) \cdot W \cdot \gamma = 675\,[t]$$

where:

Ps – cross-section area of safety pillar (m^2);

Pu – cross-section area of cut (m^2);

W – leat resistance line (m).

5. Total explosive quantity spent for one blasting of blast hole fans:

$$Q_e = Q \cdot q = 675 \cdot 0.15 = 101.25\,[kg]$$

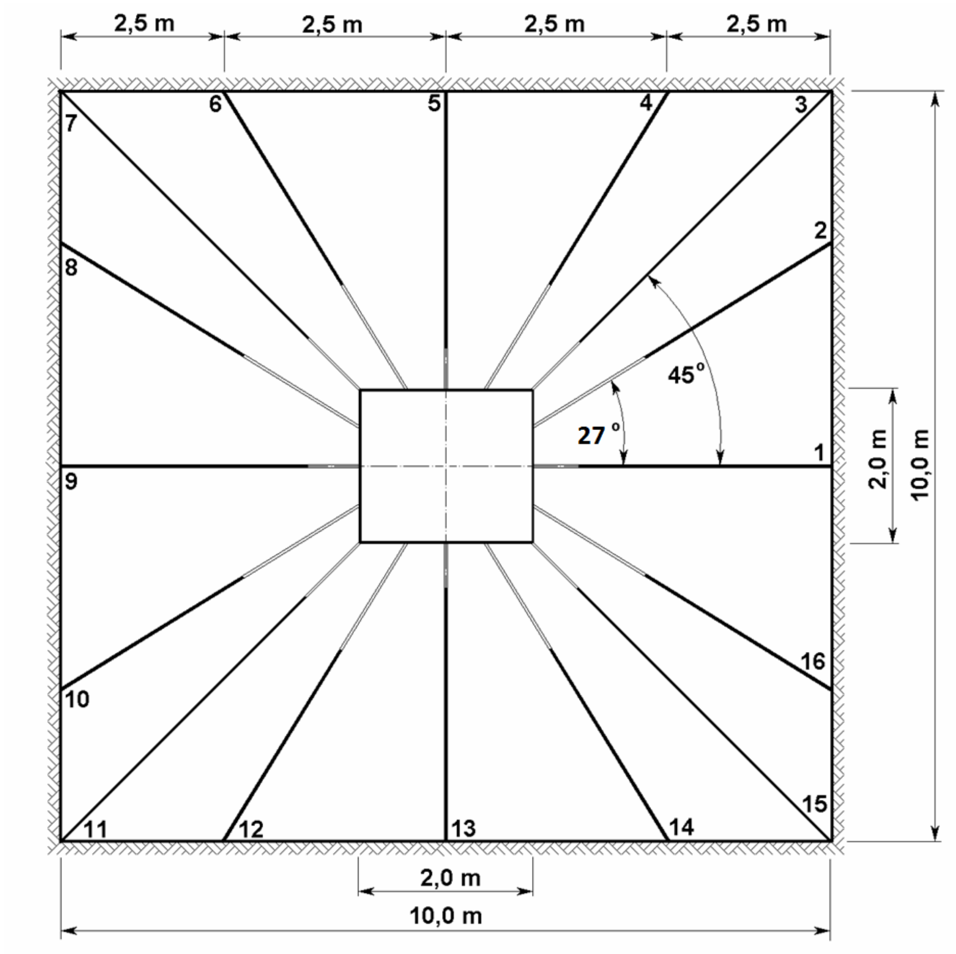


Figure 4. *Review of blast holes with a fan-shaped arrangement*

After the graphic layout of blast holes, their specification, given in Table 3, was also done.

Table 3. *Specification of the blast holes*

Blast hole No.	Angle [°]	Length [m]	Charge length [m]	Explosive quantity [kg]
1	0	4.0	3.5	6.4
2	27	4.5	2.6	4.8
3	45	5.6	5.1	9.4
4	63	4.5	2.6	4.8
5	90	4.0	3.5	6.4
6	117	4.5	2.6	4.8
7	135	5.6	5.1	9.4
8	153	4.5	2.6	4.8
9	180	4.0	3.5	6.4
10	207	4.5	2.6	4.8
11	225	5.6	5.1	9.4
12	243	4.5	2.6	4.8
13	270	4.0	3.5	6.4
14	297	4.5	2.6	4.8
15	315	5.6	5.1	9.4
16	333	4.5	2.6	4.8
	Total:	74.4	55.2	101.6

6. For the purpose of equal distribution of explosive energy, the different coefficients of charge the fan-shaped blast holes will be used and these are:
   – for blast holes 1, 5, 9 and 13

$$k_p = \frac{l_p}{l_b} = \frac{3.5}{4} = 0.88$$

   – for blast holes 2, 4, 6, 8, 10, 12 and 14

$$k_p = \frac{l_p}{l_b} = \frac{2.6}{4.5} = 0.58$$

   – for blast holes 3, 7, 11 and 15

$$k_p = \frac{l_p}{l_b} = \frac{5}{5.6} = 0.88$$

Quantitative of explosive for charging one blast hole:

$$q_e = \frac{\pi \cdot d^2}{4} \cdot l_p \cdot \rho \cdot k_z [kg]$$

7. Ignition of blast holes:

On the bottom of each blast hole, it is necessary to insert two boosters of blast, weight of 80 grams, and to connect them by detonating fuse. The ends of fuse should be sticking out of the hole, about 0.5 m, for connection to the main line of detonating fuse. Scheme of connection and ignition the explosive charges is shown in Figure 5.

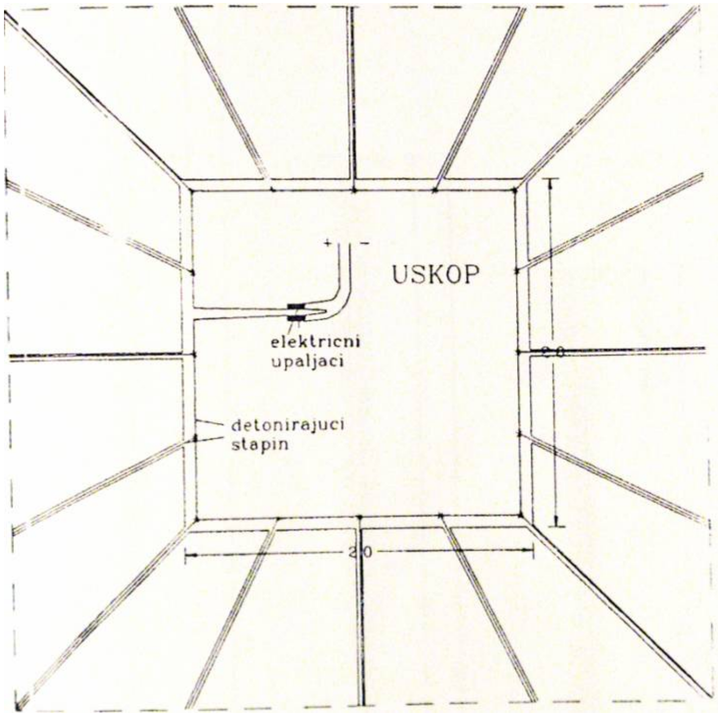


Figure 5. *Scheme of connection and ignition the explosive charges in the blast holes*

After placement of boosters and detonating fuse, the charging of ANFO explosive mixtures is done using the pneumatic filler.

After after charging of all blast holes, the connecting of end parts of detonating fuse is done with the main line. Both ends of the main line of detonating fuse are connected in one place and thus allows the transfer of detonation from both ends. To activate the detonating fuse, two electric detonators are connected due to the safe activation. Conductors of electric detonators are connected to the main electric cable that runs along the length of slope to the safe place for ignition the blast holes.

CONCLUSION

This paper gives a selection and calculation of parameters of drilling and blasting works for excavation the safety pillar (57/4), in the mine "Trepča"- Stari Trg, where the excavation of safety pillars is carried out by drilling and blasting the horizontal blast holes in the fan-shaped arrangement.

It was not considered as appropriate to establish the parameters for all safety pillars individually, but a characteristic pillar was selected to serve as the base for exploitation the other pillars.

REFERENCES

[1] P. Jovanović, Dimensioning the Mining Workings, Belgrade, 1983 (In Serbian)

[2] B. Gluščević, Opening and Methods of Underground Mining the Ore Deposits, Belgrade, 1974 (In Serbian)

[3] Lj. Savić, D. Đukanović, S. Krstić, Character of Relation the Drilling Rate and Change in Length, Underground Engineering No.12, (pgs. 17-21), Faculty of Mining and Geology Belgrade, 2003 (In Serbian)

[4] S. Trajković, V Nastić, S. Lutovac, Analysis of Realized Results in Drift Driving in the Mine "Rudnik" using the Drilling – Blasting Works, Underground Engineering No.12, (pgs.01-10), Faculty of Mining and Geology Belgrade, 2003 (In Serbian)

[5] P. Jovanović, D. Marković, S. Trajković, N. Vidanović, Study of Drilling and Blasting Process at Stops and Drifts of the Mine "Rudnik", Faculty of Mining and Geology Belgrade, 1985 (In Serbian)

[6] Project Technical Documentation of the Mine "Trepča", Institute of Zvečan (In Serbian)

INSTITUT ZA RUDARSTVO I METALURGIJU BOR | YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA | UDK: 622

UDK:551.49:622.26(045)=861

Milenko Ljubojev[*], *Dragan Ignjatović*[*], *Lidija Đurđevac Ignjatović*[*], *Vesna Ljubojev*[*]

PRIPREME ZA ISTRAŽIVANJE TRASE TUNELA I SNIMANJE TERENA[**]

Izvod

Neophodnost za povećanje kapaciteta flotacijskog jalovišta i takođe povećanje vrha brana 1A, 2A i 3A do kote K+385m je dovela do izmeštanja Kriveljske reke pomoću dva tunela: postojeći i drugi, koji će biti lociran na desnoj obali flotacijskog jalovišta. Zbog toga je, pre bilo kakve aktivnosti, neophodno snimiti mesto gde će trasa novog tunela biti locirana.

Ključne reči: t*rasa novog tunela, snimanje terena, ispitivanje trase tunela*

1. UVOD

Za odlaganje flotacijske jalovine, površinski kop “Veliki Krivelj” koristi područje dobijeno pregrađivanjem doline Kriveljske reke.

Na početku rudarskih radova, korišćena je površina u blizini flotacijskih objekata za odlagalište. Flotacijsko jalovište je prošireno 1990. godine tako što je uzet dodatni prostor u dolini Kriveljske reke, nizvodno od polja 1.

Flotacijsko jalovište je omeđeno branama 1A i 2A i projektovano je do kote K+375 m sa ukupnim kapacitetom od $94{,}3 \cdot 10^6$ [m^3].

Novo flotacijsko jalovište (polje 2) je omeđeno branama 2A i 3A i projektovano je do kote K+350 m sa ukupnim kapacitetom od $89{,}4 \cdot 10^6$ [m^3].

Uzimajući u obzir da su brane projektovane do kote K+350 m, uključujući i činjenicu da je moguće deponovati flotacijsku jalovinu samo do sredine 2008. godine, to je bilo neophodno obezbediti podizanje nivoa brana do kote K+385 m.

Zbog svega ovoga Kriveljska reka (slika 1) mora biti premeštena i to sa:

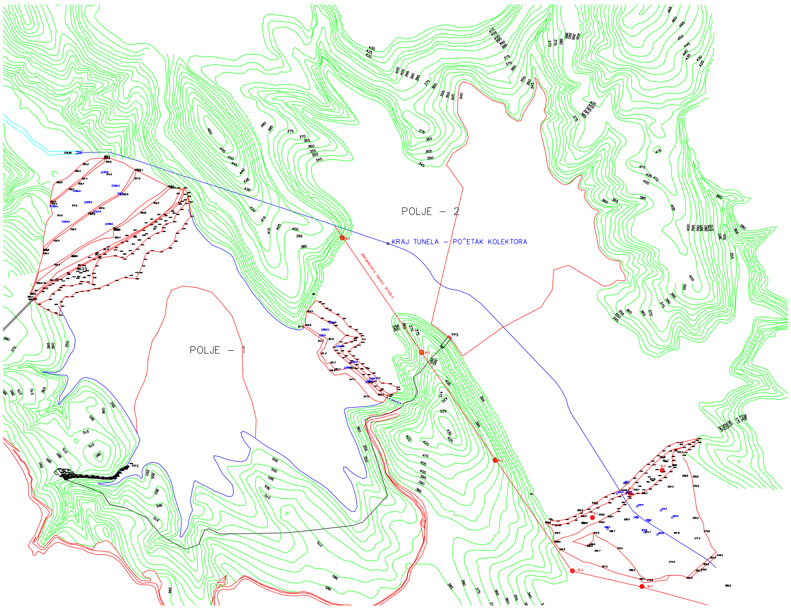
- postojećim tunelom (zona polja 1); D = 3,0 [m] and L = 1.414 [m],
- novo izgrađeni tunel; D = 3,0 [m] and L = 2.400 [m], duž desne obale flotacijskog jalovišta.

Izgradnja novog tunela na desnoj obali flotacijskog jalovišta učiniće mogućim povećanje nivoa polja 2, što će značajno povećati skladišni kapacitet (okvirno $83{,}3 \cdot 10^6$ [m^3]), slika 1.

---

[*] *Institut za rudarsvo i metalurgiju*

[**] *Ovaj rad je proistekao iz Projekta br. 33021 koji je finansiran sredstvima Ministarstva za prosvetu i nauku Republike Srbije*



Sl. 1. *Izgled terena i lokacija trase novog tunela Kriveljske reke*

2. MORFOLOŠKE KARAKTERISTIKE TRASE TUNELA

Morfolološki, teren je brdovit. Brdo Tilva Satuli se nalazi sa severozapadne strane početka trase tunela (kota terena 464).

Projektovana trasa tunela, od svoje kote 385, se pruža niz padinu Tilva Satuli prema jugoistoku, a zatim 7 do 10 metara ispod flotacijskog jalovišta (polje 2), koje je formirano u aluvionu Kriveljske reke u dužini od 450 m. Na dužini id 1.820 m menja pravac is a kote 335 skreće ka istoku.

3. HIDROLOŠKE KARAKTERISTIKE

Hidrološka analiza obuhvata slivno područje Kriveljske reke do njenog ušća u tunel, zatim reku Saraka to njenog ušća sa Kriveljskom rekom, Borsku reku do tunela Borska reka flotacijsko jalovište i, konačno, neimenovani potok koji se uliva u tunel Borske reke. Navedena slivna područja su prikazana u tabeli 1.

Tabela 1.

Reka	Profil	Površina F [km^2]	Dužina L [km]	Pad I_{ur} [‰]	Pad I_{max} [‰]
Saraka	Ulaz u Kriveljsku reku	16,38	9,7	43,3	48,5
Kriveljska reka	Ulaz u ušće	95,71	18,6	13,4	34,1
Neimenovani potok	Presek sa tunelom borske reke	2,3	3,15	59,7	69,8
Borska reka	Ulaz u tunel borske reke	12,6	6,6	39,4	62,9

F –poprečni presek sliva

L – dužina rečnog toka

I_{ur} – kontinualni rečni tok

I_{max} – maksimalni rečni tok.

Jalovište je sastavljeno od sedimenata i vulkanita iz perioda gornje krede, koji su hidrotermalno izmenjeni i koji se pojavljuju sporadično, i kvartalni sedimenata.

4. TEKTONSKA UZVIŠENJA PODRUČJA

Postoji veliki broj gravitacionih i okrenutih raseda (longitudinalnih, dijagonalnih i poprečnih). Dijagonalni i poprečni rasedi su mlađi i pretežno gravitacioni sa različitim preskocima i manje horizontalnim intervalima, i može se sa sigurnošću reći das u nastali nakon rude.

Dva osnovna pravca pružanja su SZ-JI i SZ-S.

5. INŽENJERSKO-GEOLOŠKA SVOJSTVA TERENA

5.1. Kompleks flotacjskog jalovišta

Flotacijski mulj se sastoji od prašinastog materijala sa 5 do 50% frakcija manjih od 0,006 [mm] i od oko 25% frakcija krupnijih od 0,2 [mm].

Ciklonizirani pesak je, po svom sastavu, sitnozrni pesak sa najkrupnijom frakcijom od 2 [mm], srednjezrnom frakcijom od 0,1 [mm] i oko 15% frakcije je sitnije od 0,07 [mm].

5.2. Kompleks površinski raspadnute stene

Kao is vi drugi kompleksi, i ovaj je takođe sastavljen od tri litološke jedinice, u zavisnosti od matične stene, i to su: raspadnuti andeziti različitih varijeteta, konglomerati i peščari (K^3_2), laporci, laporoviti krečnjaci, glinci, konglomerati i peskoviti krečnjaci (K^{2-3}_2).

Raspadnuta stena je smeštena ispode deluvijumskih slojeva, ređe na vrhu tla i predstavlja prelaz prema čvrstoj, nepromenjenoj steni.

5.3. Kompleks čvrste stene

Ovaj kompleks se sastoji od čvrstih, nepromenjenih stena, koje process dezintegracije nije obuhvatio. Ovde će biti opisane tri najzastupljenije celine:

Andeziti – Čvrsta stena, delimično mineralizovana, svetlo zelena do crveno braon boje. Pukotine su ispunjene glinom, i one dele stenu na blokove veličine oko 20 [cm]. Poroznost je ispucana, slabo razvijena i u osnovi vodonepropusna.

Konglomerati i peščari – To su čvrste stene, retko ispucale, a retke pukotine su ispunjene glinenim vezivom. Ove pukotine formiraju blokove veličine 50 [cm]. Poroznost je slabo razvijena. Stena je praktično vodonepropusna.

Peščari i laporci – Oni predstavljaju osnovne litološke članove, u kojima se mogu pojaviti: glinci, kongomerati, laporoviti krečnjaci i peskoviti krečnjaci. Stene su čvrste, blago prekrivene otvorenim pukotinama.

6. KONCEPCIJA I METODOLOGIJA ISTRAŽIVANJA

Cilj sledećeg koraka istraživanja je definisanje inženejrsko-geološkog odnosa i geomehaničkih karakteristika stena na trasi tunela. Neophodno je rešiti sledeće zadatke:

- utvrditi morfološke karakteristike terena,
- dati pregled geološkog sastava,
- opisati lito-genetski sastav litoloških članova i kompleksa (strukturne i teksturne karakteristike),
- definisati fizičko-mehaničke i deformacione karakteristike stena za svaki inženjersko-geološki sloj.

6.1. Terenski rad

Istražno bušenje je projektovano na 5 istražnih bušotina duž trase tunela i dve bušotine kroz flotacijsko jalovište (polje 2), za određivanje debljine flotacijskog jalovišta i takođe za određivanje kvaliteta stene, koja je između flotacijske jalovine i projektovane trase tunela.

Detaljno inženjersko-geološko kartiranje jezgra bušotina mora biti odrađeno radi definisanja litoloških članova, strukturnih i teksturnih karakteristika stena.

Kao deo inženjersko-geološkog kartiranja bušotina, svaki uzorak litoloških članova mora biti odabran i poslat u laboratoriju za dalja istraživanja.

Uzorkovanje za fizočko-mehanička i deformaciona ispitivanja se moraju obaviti za svaku promenu litoloških članova, ili na svakih 5 metara.

7. ZAKLJUČAK

Zbog neophodnosti povećanja kapaciteta flotacijskog jalovišta, Kriveljska reka mora biti izmeštena sa dva tunela. Zbog toga je, pre bilo kakve aktivnosti, neophodno osmatrati mesto gde će trasa novog tunela biti locirana.

LITERATURA

[1] R. Popović, M. Ljubojev, Elaborat o geološkim istraživanjima i fizičko-mehaničkim ispitivanjima stena trase tunela za izmeštanje ''Kriveljske reke'', Bor, 2007

[2] M. Ljubojev, Z. Stojanović, D. Mitić, D. Ignjatović, Mining-geological conditions influence for selection the best location of Krivelj's River tunnel, 41st International October Conference on Mining and Metallurgy, Kladovo, Serbia, strana 73-80, 2009.

[3] M. Ljubojev, Z. Stojanović, D. Mitić, L. Đ. Ignajtović, D. Ignjatović, Cross section form determination of new Krivelj's River tunnel, 41st International October Conference on Mining and Metallurgy, Kladovo, Serbia, strana 225-230, 2009.

[4] M. Ljubojev, M. Avdić, D. Ignjatović, L. Đurđevac Ignjatović, Influence from flotation tailings, field 2, on Krivelj River tunnel stability, Rudarski radovi, 2/2009, 2009., strana 21-28

[5] R. Popović, L. Đurđevac Ignjatović, D. Urošević, Consolidation and coefficient of permeability of flotation dumped material, Rudarski radovi 2/2008, 2008., strana 25-30

MINING AND METALLURGY INSTITUTE BOR YU ISSN: 1451-0162
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS UDK: 622

UDK: 551.49:622.26(045)=20

Milenko Ljubojev[*], *Dragan Ignjatović*[*], *Lidija Đurđevac Ignjatović*[*], *Vesna Ljubojev*[*]

PREPARATIONS FOR INVESTIGATION THE TUNNEL AXIS AND FIELD SURVEYING[**]

Abstract

Necessity of increasing the capacity of flotation tailing dump, and also increasing the top of the dams 1A, 2A and 3A to the level of K+385 m has brought to the relocation of the Krivelj River by two tunnels: the existing one and the other one that will be located on the right bank of the flotation tailing dump. Therefore, before any activity, it is necessary to survey the place where the new tunnel axis will be located.

Key words: *new tunnel axes, field surveying, tunnel axes investigation*

1. INTRODUCTION

For disposal of flotation tailing, the open pit mine "Veliki Krivelj" uses an area, got by partition of the Krivelj river valley.

In the beginning of the mining works, the area near flotation facilities was used for tailing dump. Flotation tailing dump was expanded in 1990 by taking an additional area of the Krivelj river valley, downstream from the Field 1.

Flotation tailing dump is bordered with dams 1A and 2A and designed to the level of K+375 m, and with total capacity of $94.3 \cdot 10^6$ [m^3].

New flotation tailing dump (Field 2) is bordered with dams 2A and 3A and designed to the level of K+350 m, and with total capacity of $89.4 \cdot 10^6$ [m^3].

Considering that the dams were designed to the level of K+350 m, including a fact that it is possible to deposit the flotation tailing dump only till the middle of 2008, it is necessary to ensure an increase of the dam level up to K+385 m.

Therefore, the Krivelj River (Figure 1) has to be relocated by:

- Existing tunnel (zone of the Field 1); D = 3.0 [m] and L = 1,414 [m],
- New constructed tunnel; D = 3.0 [m] and L = 2,400 [m], along the right bank of the flotation tailing dump.

Construction of a new tunnel on the right bank of the flotation tailing dump will make possible the level increase in

[*] *Mining and Metallurgy Institute Bor*
[**] *This paper is the result of the Project No. 33021, funded by the Ministry of Education and Science of the Republic of Serbia*

the Field 2, what will significantly increase the storage capacity (approximately $83.3 \cdot 10^6$ [m^3]), Figure 1.

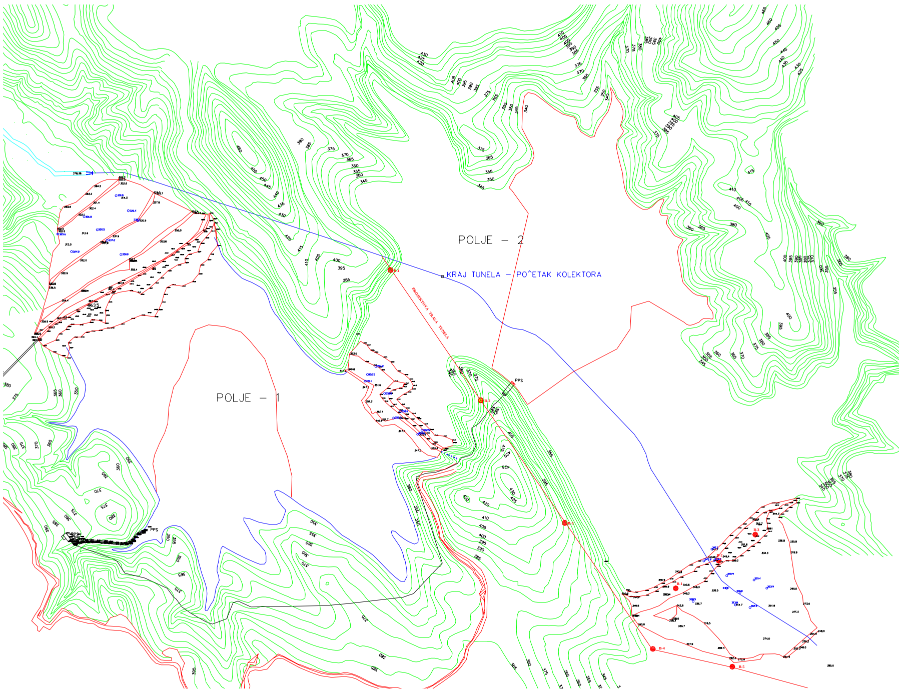


Figure 1. *View of the field and location of the new tunnel axis of the Krivelj River*

2. MORPHOLOGICAL CHARACTERISTICS OF THE TUNNEL AXIS

Morphologically, the field is hilly. The Tilva Satulli hill (pick elevation 464) is located on the north-west side from the beginning of tunnel axis

The designed tunnel axis, from its pick elevation 385, is outreached down the slope of Tilva Satuli to the south-east, and then 7 to 10 meters under the flotation tailing dump (Field 2), that was formed in the alluvium of the Krivelj River in length of 450 meters. In length of 1.820 meters, it changes strike and from pick elevation 335 it turns to the east.

3. HYDROLOGICAL CHARACTERISTICS

Hydrological analysis covers the catchment area of the Krivelj River to its mouth into the tunnel, then Saraka River to its mouth into the Krivelj River, Bor River up to tunnel the Bor River-flotation tailing dump and, finally, the unnamed stream that intakes into the tunnel of Bor River. The mentioned catchment areas are present in Table 1.

Table 1.

River	Profile	Square measure F [km^2]	Length L [km]	Downfall I_{ur} [‰]	Downfall I_{max} [‰]
Saraka	Intake into the Krivelj River	16.38	9.7	43.3	48.5
Krivelj River	Intake into the mouth	95.71	18.6	13.4	34.1
Unnamed stream	Cross section with the Bor River tunnel	2.3	3.15	59.7	69.8
Bor River	Entrance into the Bor river tunnel	12.6	6.6	39.4	62.9

F – cross section of the river basin
L – river flow length
I_{ur} – continuous river flow
I_{max} – maximum river flow.

The tailing dump consists of sediments and vulcanite from the Upper Cretaceous Period, which are hydrothermal altered, occurring sporadically there, and Quaternary sediments.

4. TECTONICAL ELEVATIONS OF THE AREA

There are numerous gravity and reverse faults (longitudinal, diagonal and cross faults). Diagonal and cross faults are younger and mostly gravity with various skips and less horizontal intervals, and it can be said for sure that they were formed after ore.

Two main directions of expanding are NW-SE and NW-N.

5. ENGINEERING-GEOLOGICAL PROPERTIES OF THE FIELD

Complex of the flotation tailing dump

Flotation sludge contains a dusty material with 5 to 50% fractions less than 0.006[mm] and about 25% fractions coarser than 0.2 [mm].

Cyclonic sand is, by its content, a fine grain sand with largest grain of 2 [mm], middle grain of about 0.1 [mm] and about 15% grains less than 0.07 [mm].

Complex of the surface disintegrated rock

Like all other complexes, this complex also consists of three lithologic units, depending on the basic rock, and those are: disintegrated andesite of different variety, conglomerates and sandstones (K32), marls, marl limestone, slates, conglomerates and sand limestone (K2-32).

Disintegrated rock is situated under diluvium layers, rarely on the top of the ground and it presents a transition to the solid, unchanged rock.

Complex of the solid rock

This complex consists of the solid, unchanged rocks, which were not included in the disintegration process. Here, three the most represented units will be described.

Andesites – Solid hard rock, partially mineralized, light-green to red-brown color. Cracks are filled with clay, and they are divided the rock on blocks, size about 20 [cm]. Porosity is ruptured, poorly developed, and basically water tightness.

Conglomerates and sandstones – Those are solid rocks, rarely cracked, and possible cracks are fulfilled with the clay binding agent. Those cracks forms the blocks, size 50 [cm]. Porosity is poorly developed. The rock is practically water tightness.

Sandstones and marls – They present the basic lithologic units, where the followings can appear: slates, conglomerates, marl limestone and sand limestone. The rocks are solid, slightly covered with open cracks.

6. CONCEPTION AND METHODOLOGY OF INVESTIGATION

The next step of investigation is aimed to defining the engineering – geological relations and geomechanical characteristics of rocks on the tunnel axis. It is necessary to solve the following - tasks:

- to determine the morphological characteristics of the ground,
- to give a review of geological composition,
- to describe a litho-genetic composition of lithologic units and complex (structural and textural characteristics),
- to define the physical – mechanical and deformable characteristics of rocks for each engineering – geological layer.

Field works

Prospecting drilling is designed with 5 prospecting drill holes along the tunnel axis and two drill holes through the flotation tailing dump (Field 2), to determine a thickness of the flotation tailing dump and also to determine the rock quality that is between the flotation tailings and designed tunnel axis.

Detailed engineering-geological mapping of drill hole cores has to be carried out to define the lithological units, structural and textural characteristics of the rock.

As a part of engineering-geological mapping of the drill holes, each lithologic unit sample has to be selected and sent to the laboratory for further testing.

Sampling for physical-mechanical and deformation testing have to be carried for each change of lithological unit, or at each 5 meters.

7. CONCLUSION

Due to the necessity of increasing the capacity of flotation tailing dump, the Krivelj River has to be relocated with two tunnels. Therefore, before any activity, it is necessary to observe the place where the new tunnel axis will be located.

REFERENCES

[1] R. Popovic, M. Ljubojev, Project of Geological and Physical-Mechanical Investigations of Rocks from the Tunnel Axis for Relocation the Krivelj River, Bor, 2007 (in Serbian)

[2] M. Ljubojev, Z. Stojanovic, D. Mitic, D. Ignjatovic, Mining-Geological Conditions Influence for Selection the best Location of the Krivelj River Tunnel, 41st International October Conference on Mining and Metallurgy, Kladovo, Serbia, pp. 73-80, 2009.

[3] M. Ljubojev, Z. Stojanovic, D. Mitic, L. Dj. Ignajtovic, D. Ignjatovic, Cross Section for Determination the new Krivelj River tunnel, 41st International October Conference on Mining and Metallurgy, Kladovo, Serbia, pp. 225-230, 2009.

[4] M. Ljubojev, M. Avdić, D. Ignjatović, L. Djurđevac Ignjatović, Influence of Flotation Tailings, Field 2, on the Stability of the Krivelj River Tunnel, Mining Engineering, 2/2009, 2009, pp. 21-28

[5] R. Popović, L. Djurdjevac Ignjatović, D. Urošević, Consolidation and Coefficient of Permeability of the Flotation Dumped Material, Mining Engineering, 2/2008, 2008, pp. 25-30

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:622.73(045)=861

Dragan Milanović, Zoran Marković**, Daniela Urosević*, Miroslav Ignjatović**

UNAPREĐENJE SISTEMA USITNJAVANJA RUDE U POSTROJENJU „VELIKI KRIVELJ"***

Izvod

Povećanje projektovanog kapaciteta trostepenog drobljenja i dvostepenog prosejavanja od 8,5 x 10^6 t na 10,6 x 10^6 t vlažne rude godišnje u pogonima flotacije Veliki Krivelj RTB-a Bor, sa g.g.k. gotovog proizvoda drobljenja od 100(%) -16 mm nižom od predhodne, projektovane g.g.k. od 100(%) -20 mm, je prema zahtevima trebalo obaviti sa raspoloživom opremom i po postojećoj tehnološkoj šemi pripreme mineralne sirovine. Detaljnom analizom rada i proizvoda drobljenja ovovg pogona registrovana su „uska grla" u postojećem procesu drobljenja i prosejavanja i u skladu s'tim predložena su adekvatna tehnička rešenja za postizanje definisanog cilja: „Povećanje kapaciteta prerade vlažne rude na Q = 10,6 x 10^6 t/god sa g.g.k. od 16 mm", uz što manja investiciona ulaganja sa postojećom opremom i po postojećoj tehnološkoj šemi prerade.

U ovom radu biće prikazan deo analize rada pomenutog pogona sa predlogom tehničkih rešenja koja mogu dovesti do ostvarenja postavljenih uslova i postizanja kpaciteta pogona drobljenja i prosejavanja rudnika bakra „Veliki Krivelj" od Q = 10,6 x 10^6 t vlažne rude godišnje sa g.g.k. od 100(%) - 16 mm.

Ključne reči: *Kapacitet, drobljenje, prosejavanje, proizvod*

1. UVOD

Ležište rude „Veliki Krivelj" nalazi se, vazdušnom linijom na oko 3 km severozapadno od Bora i na 0,5 km od najbližeg sela Krivelj, u slivu Kriveljske reke. U okviru ležišta „Veliki Krivelj" nalazi se površinski kop „Veliki Krivelj" u kome je eksploatacija počela 1982 god. U neposrednoj blizini površinskog kopa, izgrađena su drobilična postrojenja, flotacija i drugi prateći objekti, neophodni za eksploataciju i preradu, odnosno obogaćivanje rude flotacijskim postupkom. Ruda se transportuje od površinskog kopa do primarnog drobljenja kamionima, dok se jalovina transportuje kamionima i

* *Institut za rudarstvo i metalurguju Bor*
** *Univerzitet u Beogradu, Tehnički fakultet u Boru*
*** *Ovaj rad je proistekao iz Projekta TR 33023 koji je finansiran sredstvima Ministarstva za prosvetu i nauku Republike Srbije*

kombinovanim sistemom kamioni-transporteri sa trakom. Nakon primarnog drobljenja, ruda prolazi kroz prihvatni bunker, odakle se transportnim trakama transportuje do otvorenog sklada primarno izdrobljene rude. Sa sklada, ruda se zvezdastim dodavačima i sistemom transportnih traka šalje u postrojenje za sekundarno i tercijarno drobljenje sa prosejavanjem [4,3].

Rudnici bakra – Bor (u daljem tekstu RBB) su dogovorili sa Institutom za rudarstvo i metalurgiju Bor (IRM Bor) izradu tehničkog rešenja da bi se postigao kapacitet prerade rude ležišta „Veliki Krivelj" od $10{,}6\times10^6$ tona vlažne rude godišnje. Sa time u vezi, neophodno je izvršiti proveru kapaciteta i stanja postojeće opreme i dati rešenja za postizanje definisanog kapaciteta prerade sa definitivnim proizvodom drobljenja g.g.k. 16 mm. Uz napomenu, da u svim pokušajima da se u industrijskoj praksi ostvari definisani kapacitet od $10{,}6 \times 10^6$ rovne rude godišnje, nisu dobijeni zadovoljavajući rezultati. Uprkos činjenici, da je pri tim pokušajima za finoću gotovog proizvoda drobljenja zahtevano da ona iznosi g.g.k. 20 mm. To je u odnosu na sada zahtevanu finoću gotovog proizvoda drobljenja od g.g.k. 16 mm predstavljalo daleko lakšu mogućnost.

Bondov radni indeks rude Velikog Krivelja se kreće u rasponu od 12-14 kWh/t, pa za dalji rad usvajamo tvrdu rudu na prelazu između srednje tvrdih i tvrdih ruda sa vrednošću Bondovog Radnog indeksa od 14 kWh/t odnosno, 15,43 kWh/sht.

U procesu iznalaženja tehničkog rešenja razmotrene su sve mogućnosti da se sa postojećom tehnologijom prerade rude uz najminimalnije investicione zahvate u fazi sekundarnog i tercijalnog drobljenja. Primarno prosejavanje, sekudarno drobljenje, tercijalno drobljenje u zatvorenom ciklusu sa sekundarnim prosejavanjem odgovori zahtevima traženog kapaciteta prerade i tražene finoće gotovog proizvoda drobljenja, ($Q=10{,}6 \times 10^6$ t/god. i g.g.k. 16 mm) [5,6]. Dakle, kriveljska ruda u sistemu sekundarnog i tercijernog drobljenja i prosejavanja terbalo bi da se *prerađuje po postojećoj tehnološkoj šemi prerade sa postojećom opremom*, uz predviđena tehnička rešenja koja će zadovoljiti postavljene uslove u pogledu postizanja traženog kapaciteta prerade i finoće gotovog proizvoda drobljenja. Na slici 1. data je panorama „Velikiog Krivelja" sa objektima za preradu i obogaćivanje rude.

Sl. 1. *Panorama „Velikiog Krivelja" sa objektima za preradu i obogaćivanje rude*

POSTIZANJE KAPACITETA ($Q=10{,}6 \times 10^6$ t/god., g.g.k. 16 mm)

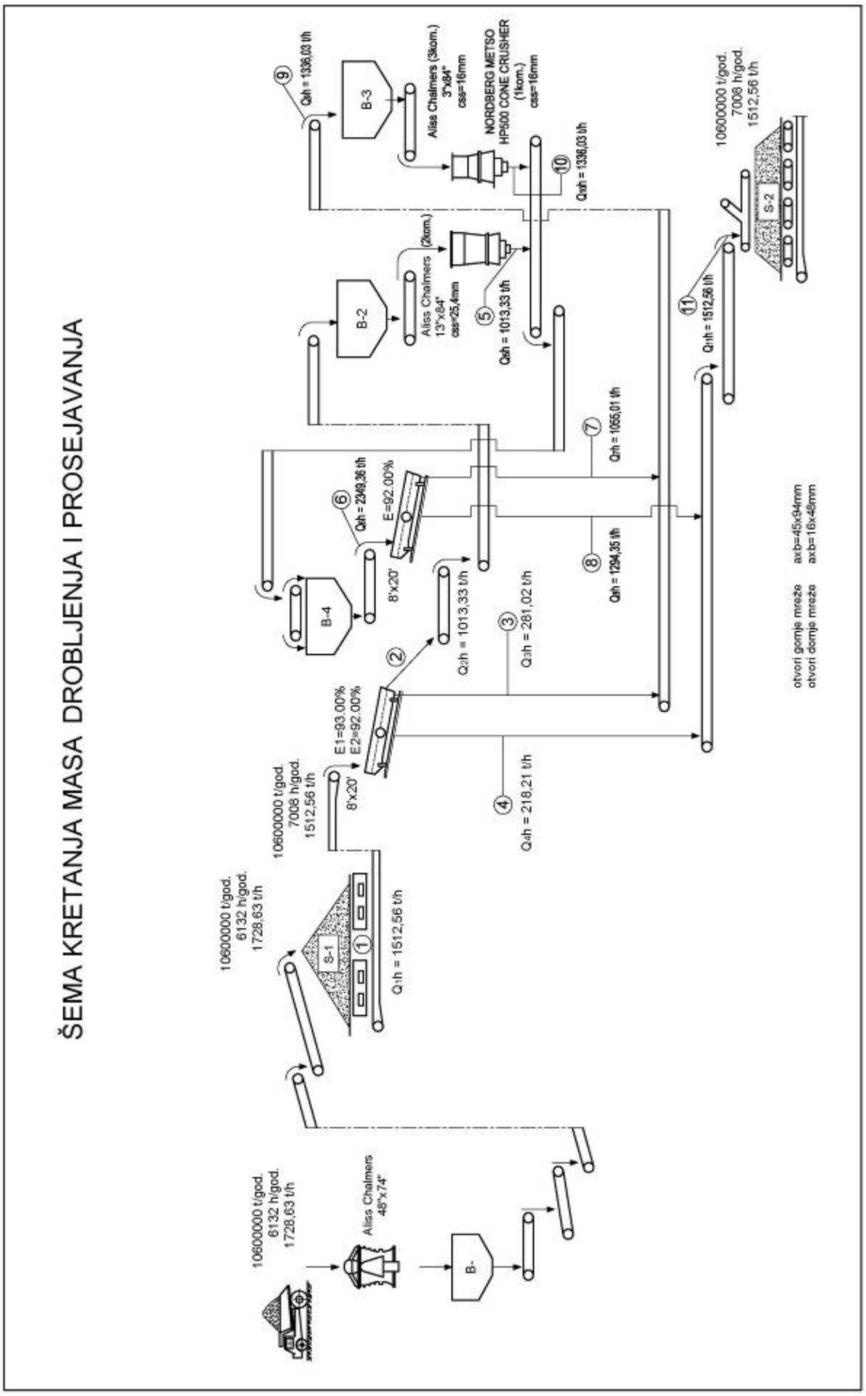
Preduslov, za ostvarenje definisanih ciljeva, ($Q=10{,}6 \times 10^6$ t/god. i g.g.k. 16 mm) je da sva raspoloživa oprema i objekti budu u ispravnom i funkcionalnom stanju, *revitalizovana oprema*, u meri u vremenskom iskorišćenju u kojem je to neophodno za dostizanje predviđenog kapaciteta i kvaliteta prerađene rude. To vremensko iskorišćenje opreme i objekata očekuje se *da bude povećano* u odnosu na predhodni režim rada kada se radilo sa nižim kapacitetom prerade (od 8,5 ili $10{,}6 \times 10^6$ t/god., i većom gornjom graničnom krupnoćom gotovog proizvoda drobljenja od g.g.k. 20 mm). Sa usvojenim koeficijentom vremenskog iskorišćenja postrojenja sekundarnog i tercijalnog drobljenja i prosejavanja od k=0,8 potrebni časovni kapacitet prerade rude će biti: Qh = 1512.56 t/h vlažne rude. Za isti, izvršiće se neophodna verifikacija postrojenja sekundarnog i tercijalnog drobljenja i prosejavanja, kao i prikaz šeme kretanja masa. Dakle, za ostvarenje većeg kapaciteta će biti potrebno pored ostalog, maksimalno korišćenje efektivnog vremenskog fonda rada agregata i povećanje dopuštenog maksimalnog opterećenja procesne opreme. Dalje se nećemo zadržavati na vremenskom iskorišćenju opreme i objekata ali moramo konstatovati da u projekciji konačnog tehničkog rešenja raspoloživi fond vremenskog iskorišćenja je upotrebljen kao jedan od bitnih činilaca tog tehničkog rešenja [5]. Na slici 2., dat je prikaz postojeće tehhnološke šeme sa opremom i kretanjem masa sistema sekundarnog i tercijernog drobljenja i prosejavanja rude u rudniku „Veliki Krivelj“, a za usvojeno tehničko rešenje.

Sa slike 2., vidi se, da je sistem sekundarnog i tercijernog drobljenja i prosejavanja tehnološki povezan sa sistemom primarnog drobljenja. Iz tih razloga sistem primarnog drobljenja treba da da proizvod koji će najbolje odgovarati sistemu sekundarnog i tercijernog drobljenja i prosejavanja u smislu postizanja traženih uslova. Pri tome se i ovde podrazumeva da je sva oprema u ispravnom i funkcionalnom stanju, *revitalizovana oprema*, kako bi mogla da zadovolji novoprojektovane trehnološke uslove [6]. A to su, da proizvod primarnog drobljenja koji se prema tehnološkoj šemi skladišti na otvorenom skladu S-1., bude najfiniji mogući proizvod. On kasnije u sistemu sekundarnog i tercijernog drobljenja i prosejavanja dolazi prvo na dvoetažno sito gde prosev druge sejne površine predstavlja gotov proizvod drobljenja. Pa je učešće tog proizvoda što poželjnije u ulaznoj rudi koja dolazi u sistem sekundarnog i tercijernog drobljenja i prosejavanja. Zbog navedenih čijenica neophodno je da se izvede takvo setovanje primarne drobilice koje će da sa jedne strane zadovolji gore navedene uslove, a sa druge strane da obezbedi zadovoljenje kapaciteta i pouzdanost u radu. Granulosastav proizvoda primarnog drobljenja preuzet je iz kataloga proizvođača i predstavljen u tabeli 1., a prema usvojenom izlaznom otvoru primarne drobilice „Allis Chalmers“ 8’’x74’’ u otvorenom položaju od OSS =139,7 mm, ($5^{1/2''}$) i za srednje tvrde rude.

Tabela 1. *Granulometrijski sastav izlaza iz primarne drobilice „Allis Chalmers“ 48x74’*

Klasa krupnoće	Izlaz OSS =139,7 mm, ($5^{1/2''}$)		
d(mm)	m(%)	R(%)	D(%)
-203,20+190,5	1	1	100
-190,5+177,80	1,5	2,5	99
-177,80+165,10	1,5	4	97,5
-165,10+152,40	2	6	96
-152,40+139,70	4	10	94
-139,70+127	5	15	90
-127+114,30	7	22	85
-114,30+101,60	5,5	27,5	78
-101,60+88,90	8,5	36	72,5
-88,90+76,20	8	44	64
-76,20+63,50	8,5	52,5	56
-63,50+50,80	7,5	60	47,5
-50,80+45	4,51	64,51	40
-45+38,10	4,49	69	35,49
-38,10+31,75	3,50	72,5	31
-31,75+25,40	5	77,5	27,5
-25,40+19,05	3,5	81	22,5
-19,05+16	2,14	83,14	19
-16+12,70	1,86	85	16,86
-12,70+6,35	4	89	15
6,35+0	11	100	11

Predstavljeni granulometrijski sastav, proizvoda primarnog drobljenja je ulaz u pogon sekundarnog i tercijalnog drobljenja i prosejavanja. Ulaz na primarno dvoetažno sito.



Sl. 2. *Šema kretanja masa u sekundarnom i tercijalnom drobljenju i prosejavanju*

Iz granulosastava se vidi, da ulazna ruda u pogon sekundarnog i tercijalnog drobljenja i prosejavanja ima krupnoću od g.g.k. 100% -203 mm, a učešće klase krupnoće - 20 mm iznosi cca α^{-20}= 20 %, dok je učešće klase krupnoće –16 mm α^{-16}= 16,86 %. Takođe se vidi, da je učešće krupnijih klasa krupnoće i to: –63,5 mm $\alpha^{-63,5}$= 47,5 %, dok je to za klasu krupnoće -45 mm α^{-45}= 35,49 %.

Za svu, a posebno za visoko opterećenu opremu, gde se u ovakvom sistemu drobljenja, prvenstveno očekuje da to budu tercijarne drobilice i sekundarna sita za prosejavanje gotovog proizvoda drobljenja, biće izvršena verifikacija radi utvrđivanja stepena njihove kapacitativne mogućnosti i opterećenosti.

Za sekundarno drobljenje u pogonu „Veliki Krivelj“ predviđene su dve (u predom periodu instalisane i potvrđene u radu) **revitalizovane** sekundarne drobilice tipa: Allis Chalmers Hydrocone EHD, veličine: 13’’ x 84’’, sa ecc: 2’’ (50,8 mm). Na slici 3., predstavljena je navedena drobilica koja je instalisana u zgradi sekundarnog drobljenja.

Za tercijerno drobljenje u pogonu „Veliki Krivelj“ predviđene su **tri** (u predhodnom periodu instalisane i potvrđene u radu) **revitalizovane** tercijerne drobilice tipa: Allis Chalmers Hydrocone EHD, veličine: 3’’ x 84’’, sa ecc: 2’’ (50,8 mm), i **jedna nova**: „metso minerals“ „*Nordberg*“ HP 500 Series Cone Crushers.

Na slici 4 i 5., predstavljene su respektivno navedene drobilice.

Sl. 3. *Sekundarne drobilice „Allis Chalmers“ HydroconeEHD,13’’x 84’’*

Sl. 4. *Tercijarne drobilice „Allis Chalmers" Hydrocone EHD, 3''x 84''*

Sl. 5. *Tercijerna drobilica „Nordberg" HP Series Cone Crushers, HP 500, „metso-minerals"*

U tabeli 2., prikazan je očekivani granulometrijski sastav sekundarno izdrobljene rude po Allis Chalmers-u, za usvojeni izlazni otvor sekundarne drobilice Hydrocone EHD, katalog broj B 223.025 E, veličine: 13x84'' pri CSS = 1''(25 mm) i ekcentru drobilice ecc= $1^{1/4}$''(32 mm) (maksimalno zatezanje drobilice), za srednje tvrde sirovine (Wi = 14 kWh/t; 15,43 kWh/sht). Iz koga se vidi, *da ruda ima g.g.k. preko 45 mm, odnosno 100% -50 mm*, a učešće klase krupnoće - 20 mm iznosi oko α^{-20}= 44 %, dok je učešće klase krupnoće cca α^{-16} = 36 %.

Neophodne podatke o granulometrijskom sastavu proizvoda tercijalnog drobljenja usvojićemo iz kataloga proizvođača „metso-minerals", a za novokupljenu drobilicu HP 500. Podrazumevajući da će nakon predviđene rekonstrukcije postojećih drobilica: „Allis Chalmers" Hydrocone EHD, 3''x 84'' sa ecc: 2'' (50,8 mm) one davati iste proizvode drobljanja kao i novokupljena drobilica. Pa prema tome i da će imati iste granulometrijske sastave. Kataloški očekivani granulometrijski sastav proizvoda „Nordberg", HP 500, „metso-minerals" drobilice, brošura broj N_O: 2051-04-07-CBL/Tampere-English, sa ekcentrom, ecc po standardu proizvođača za ovaj tip drobi-lica, pri datom zatezanju drobilice CSS = 16 mm za srednje tvrde sirovine (Wi = 14 kWh/t; 15,43 kWh/sht), dat je u tabeli. 3.

Na osnovu predhodnih analiza rada sistema sekundarnog i trecijernog drobljenja i prosejavanja rudnika „Veliki Krivelj", kada su u datom sistemu pored drugačijeg setovanja datih drobilica bile instalisane i drukčije mreže sita (ovde se to prvenstveno odnosi na veličinu otvora sejne površine sita, respektivno prema etaži sita, a x a=60x60 mm i a x b=20x40 mm) uvek je konstatovana preopterećenost tercijalnog drobljenja i sekundarnog prosejavanja, (sekundarnih sita sa smanjenom efikasnošću sejanja). [3,4]

Takođe, pri takvoj raspodeli masa sa ovim veličinama mreža sita, javlja se rasterećenost sekudarnog drobljenja izazivajući debalans masa u čitavom sistemu drobljenja i prosejavanja.

Tabela 2. *Kataloški granulometrijski sastav proizvoda drobljenja sekundarne drobilice „Allis Chalmers“ Hydrocone EHD, veličine:13'' x 84'', ecc=32 mm (11/4'')*

Klasa Krupnoće	Izlaz CSS =25,4 mm, (1'')		
d (mm)	m(%)	R(%)	D(%)
-50+40	8	8	100
-40+30	20	28	92
-30+25	12	40	72
-25+20	16	56	60
-20+16	8	64	44
-16+12	10	74	36
-12+10	4	78	26
-10+5	12	90	22
-5+0	10	100	10

Tabela 3. *Kataloški granulometrijski sastav proizvoda drobljenja tercijarne drobilice „Nordberg“, HP 500, „metso-minerals“*

Klasa Krupnoće d (mm)	Izlaz CSS =16 mm, (5/8'')		
	M (%)	R (%)	D (%)
-25+20	10	10	100
-20+16	12	22	90
-16+12	16	38	78
-12+10	9	47	62
-10+5	23	70	53
-5+0	30	100	30

Iz svega navedenog, a radi zadovoljenja datih uslova za postizanje traženog kapaciteta prerade sa traženom finoćom gotovog proizvoda drobljenja, Q=10,6 x 10^6 t/god. i g.g.k. 16 mm, pri izradi ovog tehničkog rešenja pripreme, uvešće se u odnosu na pređašnje, *izmenjeni tehnološko tehnički parametri prerade rude*. U tom smislu, napomenuto je da se pod time podrazumeva značajna rekonstrukcija tri tercijalnih drobilica tipa Allis Chalmers Hydrocone EHD, 3'' x 84'' sa ecc: 2'' (50,8 mm) (zame-na ekscentra drobilica i zamena postojećih čeličnih obloga t.j. uvođenje novog profila obloga drobilica i sistema za automatsku regulaciju IC-(Intelligent control) 7000) i nabavka jedne nove tercijerne drobilice tipa: „Nordberg“ HP Series Cone Crushers, HP 500, kompanije „metso-minerals“. Rekonstruisne postojeće drobilice Allis Chalmers EHD trebalo bi da daju slične ili iste granulometrijske sastave proizvoda drobljenja rude (kataloški granulo sastavi proizvoda drobljenja) kao i nova HP 500 drobilica. Zatim je neophodna i rekonstrukcija sita, koja podrazumeva promenu sejne površine sita tj. uvođenje sejne površine sita sa drugačijim otvorima odnosno, dimenzijama otvora mreže sita. (Umesto postojećih mreža sita, sa dimenzijama pravougaonih otvora a x a = 60 x 60 mm i a x b = 20 x 40 mm, uvode se dimenzije pravougaonih otvora mreže sita a x b=45 x 94 mm i a x b=16 x 48 mm, respektivno prema odgovarajućim etažama sita). Sve ovo, uz dato setovanje drobilica sa očekivanim proizvodima drobljenja istih (tab. 2 i 3.), kao i uz finiji ulaz u sistem drobljenja i prosejavanja, finiji proizvod primarne drobilice (tab. 1), i uz rekonstruisane granulometrijske sastave, koji su sračunati za pojedine neophodne proizvode, a koji zbog preobimnosti ovovga rada neće biti prikazani, omogućilo je da se dobije bilans masa koji je već predstavljen u ovom radu (sl. 2).

Na ovaj način, postiže se ravnomerniji balans kretanja masa u pogonu. Izvršena je preraspodela masa i dodatno se opterećuju sekundarne drobilice uz povećanje dopuštenog maksimalnog opterećenja tercijalnih drobilica. Deo mase koji je ranije odlazio na tercijerno drobljenje sada odlazi na sekundarno drobljenje. Time se omogućuje rad sekundarnog prosejavanja u granicama maksimalnih tehnoloških mogućnosti a radi ostvarenja datih potreba u ovom zatvorenom sistemu tercijalno drobljenje-sekundarno prosejavanje.

Sada se može predstaviti očekivani granulometrijski sastav definitivnog proizvoda drobljenja koji je dat u tabeli 4., iz kojih se vidi da očekivani definitivni proizvod drobljenja ispunjava zahtev u pogledu granulometrije.

Tabela 4. *Granulo sastav definitivnog proizvoda drobljenja i prosejavanja*

Klasa krupnoće	**Definitivni proizvod drobljenja i prosejavanja**		
d(mm)	**m(%)**	**R(%)**	**D(%)**
-16+12	21,12	21,12	100
-12+10	10,94	32,06	78,88
-10+5	30,06	62,12	67,94
-5+0	37,88	100	37,88

UTICAJ NA BILANS MASA

Sada raspolažemo podacima koji mogu kvalitativno i kvantitativno da predstave efekte usvojenog rešenja, za postizanje zadatih vrednosti kapaciteta i finoće proizvoda.

Tabela 5. *Uporedni prikaz sadržaja karakterističnih klasa krupnoće i kapaciteta na odgovaraujćim pozicijama u sistemu drobljenja i prosejavanja, a prema veličinama otvora sejnih površinama sita*

Granulosastav ulaza u sistem drobljenja, na primarno sito							
Sadržaj klase krupnoće α^{+45}mm			**α^{+45}mm 64,51(%)**		**Sadržaj klase krupnoće α^{+60}mm**		**α^{+60}mm 55,20(%)**
Proizvod		**Ulaz na prim.sito Q_1**	**Ulaz na II etažu prim.sita Q_2'**	**Odsev I etaže prim.sita $Q_2 = Q_5$**	**Odsev II etaže prim.sita Q_3**	**Prosev II etaže prim.sita Q_4**	**Proizvod sekundarnog drobljenja Q_5**
Učešće (t/h)	# 45 # 16 mm	1512,56	499,23	1013,3	281,02	218,21	γ_m (-203,2+ 45) mm 1013,33
Učešće (%)	# 45 # 16 mm	100	33	67	18,58	14,42	γ_m (-203,2+ 45) mm 67
Učešće (t/h)	# 60 # 20 mm	1512,56	630,19	882,3	371,32	258,87	γ_m (-203,2+ 60) mm 882,37
Učešće (%)	# 60 # 20 mm	100	41,66	58,34	24,55	17,11	γ_m (-203,2+ 60) mm 58,34

Proizvod		Ulaz na sek. sito $Q_6 = Q_{10} + Q_5$	Odsev sek.sita Q_7	Prosev sek.sita Q_8	Ulaz u terc.drob $Q_9 = Q_3 + Q_7$	Proizvod terc.drob $Q_{10} = Q_9$	Definitivni proizvod drobljenja $Q_{11} = Q_8 + Q_4$
Učešće (t/h)	# 45 # 16 mm	2349,36	1055,01	1294,35	1336,03	1336,03	1512,56
Učešće (%)	# 45 # 16 mm	155,33	69,75	85,58	88,33	88,33	100
Učešće (t/h)	# 60 # 20 mm	1965,11	711,42	1253,69	1082,74	1082,74	1512,56
Učešće (%)	# 60 # 20 mm	129,96	47,07	82,89	71,62	71,62	100

Ti podaci su predstavljeni u tabeli 5., i odnose se na prikaz obračunske klase krupnoće α^{+xx} mm(%), masene raspodele karakteristične klase krupnoće, γm (%) i kapaciteta sita i drobilica, Q_1-Q_{11} (t/h) za usvojeno rešenje tj. za usvojenu vrednost otvora sejnih površina primarnog i sekundarnog sita.

Ranije je konstatoano, da se postojeći debalans masa „usko grlo" masene raspodele u datoj tehnološkoj liniji može otkloniti promenom veličina sejnih površina primarnog sita.

Na taj način postiže se ravnomernija opterećenost svih drobiličnih agregata i bolje iskorišćenje raspoloživog kapaciteta tj. mogućnosti istih.

U prvom redu ove tabele, predstavljen je uporedni prikaz sadržaja karakteristične klase krupnoće u granulosastavu ulaza u sistem drobljenja, na primarno sito. Samo po osnovu granulometrijskog sastava vidi se evidentna razlika masenog sadržaja kada je u pitanju klasa krupnoće, (-g.g.k.+60) mm ili kalsa, (–g.g.k.+45) mm. Ovo se direktno prenosi na masenu raspodelu γm(%) = 1013,33 t/h odnosno, γm(%) = 882,37 t/h, odseva I etaže primarnog sita klasa krupnoće: (-203+60)mm i (-203+45) mm respektivno u predhodnom odnosno, u ovde usvojenom rešenju.

Konačno, u daljem prikazu ove tabele 5., a na osnovu podataka iz šeme kretanja masa, vidi se promena kapaciteta na sekundarnom drobljenju - kvantitativna promena u bilansu masa.

To je dovelo, uz promene veličine otvora donje prosevne površine primarnog dvoetažnog sita i sekundarnog jednoetažnog sita i doterivanje tercijalnog proizvoda drobljenja datim setovanjem drobilice do postizanja traženog kapaciteta sistema drobljenja od Q=10,6 x 10^6 t /god i potrebne finoće proizvoda od g.g.k. 100 % (-16) mm.

4. VERIFIKACIJA KAPACITETA

Usled postizanja dopuštenog maksimalnog opterećenja procesne opreme neophodno je za svu, a posebno za tu visoko opterećenu opremu, (gde se prema datoj šemi kretanja masa očekuje da to budu tercijarne drobilice i sekundarna sita za prosejavanje gotovog proizvoda drobljenja), izvrši verifikacija radi utvrđivanja stepena njihove kapacitativne mogućnosti i opterećenosti u ovakvom sistemu drobljenja i prema ovakvoj raspodeli kretanja masa. [1,2]

Provera kapaciteta sita je izvršena po metodi Mehanobr a na osnovu podataka koji su sadržani u šemi kretanja masa. Propusna moć sejnih površine sita je u granicama tehnoloških mogućnosti ostvarenja zadatih potreba. Iz tih razloga tehnološki proces prosejavanja se mora konstantno održavati na zadatom projektovanom nivou. U protivnom, može doći do poremećaja rada sita.

Dalja provera kapaciteta postojeće opreme u sekundarnom i tercijernom drobljenju pokazala je da, većina opreme uglavnom može zadovoljiti predviđene kapacitete i nove tehnološke uslove.

Ostvarenje konstantnog projektovanog kapaciteta od Q=1512,56 t/h, zavisiće isključivo od drobilica u radu tj. od njihovog parcijalno ostvarivog kapaciteta. Za propisane tehnološke uslove rada, pri kojima je predviđeno da sekundarne drobilice ove veličine, Allis-Chalmers Hydrocone crusher EHD 13“ x 84“ (330,20 x 2133,6 mm) budu podešene na CSS=25,4 mm (1"), imajući tada kapacitet (Prema katalogu proizvođača, firme Allis Chalmers, broj B 223025 E,) od *Qkat=540 t/h suve rude,* a da tercijerne drobilice Allis Chalmers HYDROCONE - EHD, 3’’ x 84’’ (76,2 x 2133,6 mm) budu podešene na CSS = 16 mm (5/8"), imajući pri tome ka-pacitet (Prema katalogu, Allis Chalmers, kataloški broj 17 B 5239), od *kat* = 471,64 t/h (520 sht/h) suve rude odnosno, da tercijerna drobilica *Nordberg* HP 500 CONE CRUSHERS kompanije „metso-minerals“ ima, (prema kataloškim poda-cima firme metso-minerals, brošura broj N_O: 2051-04-07-CBL/ Tampere-English,), kapacitet u granicama od minimalni $Qkat_{min}= 280\ t/h$ odnosno, maksimalni od $Qkat_{max} = 350\ t/h$ suve rude.

Dve sekundarne drobilice, prema šemi kretanja masa treba da savladaju časovni kapacitet sekundarnog drobljenja od $Qh_{sec} = 1013,33\ t/h$ odnosno, = 506,66 t/h vlažne rude tj. $Qh_{sec}=483,86\ t/h$ suve rude, po jednoj drobilici. Kako je potreban kapacitet sekundarnog drobljenja po jednoj drobilici, manji od kataloškog kapaciteta, $Qh_{sec}=483,86\ t/h < Qkat=540\ t/h$ suve rude, za predviđeno podešavanje CSS=25,4 mm (1") drobilice u potpunosti zadovoljavaju novonastale potrebe.

Tri tercijarne drobilice, Allis Chalmers HYDROCONE EHD, 3’’ x 84’’ i jedna nova *Nordberg* HP 500 CONE CRUSHERS kompanije „metso-minerals“ treba, prema istoj šemi kretanja masa, da savladaju časovni kapacitet tercijernog drobljanja od $Qh_{terc} = 1336,03\ t/h$, t.j. po jednoj tercijernoj drobilici: *Qh=334 t/h vlaže rude, odnosno Qh= 318,97 t/h suve rude.*

Tražena rekonstrukcija ovih drobilica, može dovesti do značanije promene kapaciteta ovih uređaja. Kapacitet svake konusne drobilice za srednje i sitno drobljenje izveden po osnovu teoreme „Guldena“ za zapreminu prstena radnog prostora drobilice. Nakon tražene rekonstrukcije naših drobilica možemo predpostaviti da će u kompaniji „metso-minerals“ pri rekonstrukciji zadržati stare kataloške kapacitete ovih drobilica. To se, obzirom na teoretske izraze za optimalni broj obrtaja ekscentrične čaure $\mathbf{n_0}$ i kapaciteta drobilice Q, može ostvariti preko određivanja karakterističnih ključnih vrednosti tehničkih parametara koji direktno utiču na kapacitet drobilice (promena profila zaštitnih obloga tj. zapremine prstena, radnog prostora drobilice, ekscentritet drobilica, broj obrtaja ekscentrične čaure itd.) a da se pri tome dobije potrebni kapacitet i određeni granulo-metrijski sastav proizvoda drobljenja.

Kako je potreban časovni kapacitet tercijernog drobljenja po jednoj drobilici manji od maksimalnog kataloškog kapaciteta $Qh = 318{,}97\ t/h < Qh_{katalo\check{s}i} = 471{,}64\ t/h$ suve rude ovog tipa drobilice. Zaključujemo da Allis Chalmers HYDROCONE EHD drobilice odgovaraju novonastalim potrebama drobljenja, tj. zadovoljavaju definisane kapacitete.

Isto se dešava i sa tercijernom drobilicom HP 500. Kao u predhodnom slučaju, potreban je isti časovni kapacitet tercijerne drobilice $Qh= 318{,}97\ t/h$ suve rude, i isti je takođe, manji od maksimalnog kataloškog kapaciteta ovog tipa drobilice, (Nordberg HP 500 CONE CRUSHER), $Qh = 318{,}97\ t/h < Qkat_{max} = 350\ t/h$ suve rude pa i ta drobilica odgovara novoprojektovanim uslovima rada.

ZAKLJUČAK

Da bi se postigao kapacitet prerade rude ležišta „Veliki Krivelj“ od $10{,}6 x 10^6$ tona vlažne rude godišnje sa g.g.k. 16 mm, kriveljska ruda u sistemu sekundarnog i tercijernog drobljenja i prosejavanja terba da se *prerađuje po postojećoj tehnološkoj šemi prerade sa postojećom opremom*, uz predviđeno tehnička rešenje koja će zadovoljiti postavljene uslove. Ono je ostvareno kroz nekoliko napadnih tačaka i to:

- Da se vremensko iskorišćenje rada opreme i agregata podigne na viši neophodni nivo.
- Da sva raspoloživa oprema i objekti budu u ispravnom i funkcionalnom stanju, *revitalizovana oprema.*
- Da proizvod primarnog drobljenja bude najfiniji mogući proizvod primarnog drobljenja.
- Da sva setovanja drobilica budu postavljena prema navodima u ovom radu kako bi se dobijali granulo-metrijski sastavi proizvoda drobljenja i prosejavanja kao ovde prikazani.
- Da se zameni jedna dotrajala tercijerna drobilica sa novom HP 500 „metso-minerals” drobilicom.
- Da se rekonstrukcija postojećih tercijernih drobilica Allis Chalmers HYDROCONE EHD, 3’’ x 84’’ izvede po navodima iz ovoga rada tako da one zadovolje u pogledu potrebnog kapaciteta i granulometrije proizvoda drobljenja.
- Da se uvode dimenzije pravougaonih otvora mreže sita a x b = 45 x 94 mm i a x b = 16 x 48 mm, respektivno prema odgovarajućim sitima i etažama sita.

Jedino pod takvim uslovima moguće je zadovoljiti postizanje kapaciteta definitivnog proizvoda drobljenja od $Q = 10{,}6 \times 10^6$ t godišnje sa g.g.k. 16 mm čiji je očekivani granulometrijski sastav definitivnog proizvoda drobljenja dat u tablici 4. Izvesna potvrda ovih navoda je izvršena verifikacija kapaciteta svih uređaja u sistmu drobljenja i prosejavanja rudnika „Veliki Krivelj“ na osnovu koje je konstatovano da u ovakvoj koncepciji prerade data oprema može kapacitativno da zadovolji ovakvim rešenjem novopostavljene uslove.

5. LITERATURA

[1] „Tehnološke osnove projektovanja postrojenja za pripremu mineralnih sirovina“, Rudarski Institut Beograd, 1999.

[2] N. Magdalinović, „Usitnjavanje i Klasiranje,“ Beograd, 1999.

[3] „DRP Rekonstrukcija flotacije Veliki Krivelj u cilju povećanja kapaciteta prerade na 10,6 x 106 t rovne rude godišnje“ – RI Beograd, 1995.

[4] „Dopunski rudarski projekat povećanja godišnje prerade rude u flotaciji Veliki Krivelj na 10,6 mil.t rovne rude.“ Tehnološki projekat – Institut za Bakar Bor,1997.
[5] „Predlog tehničko-tehnološkog rešenja za smanjenje g.g.k. proizvoda drobljenja rude V.Krivelj u cilju smanjenja troškova usitnjavanja i povećanja kapaciteta prerade rude“. Institut za bakar Bor - Rudarski Institut Beograd, januar 1993.
[6] „Studija opravdanosti investiranja u proizvodnji kocentrata bakra na površinskom kopu i flotaciji „Veliki Krivelj“ – RTB Bor,“ Beograd, 2010. Geo-in International Beograd.

MINING AND METALLURGY INSTITUTE BOR YU ISSN: 1451-0162
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS UDK: 622

UDK: 622.73(045)=20

Dragan Milanović, Zoran Marković**, Daniela Urosević*, Miroslav Ignjatović**

SYSTEM IMPROVEMENT OF ORE COMMINUTING IN VELIKI KRIVELJ PLANT***

Abstract

Increasing the design capacity of three-stage crushing and two-stage sieving from 8.5 x 10^6 t to 10.6 x 10^6 t of wet ore annually in the Flotation Plant Veliki Krivelj of RTB Bor with the upper size limit of final crushing product of 100 (%) - 16 mm lower than the previous one, the designed upper size limit of 100 (%) - 20 mm, according to investor demands should have to be done with the available equipment and existing technological scheme of mineral processing. By detailed analyzing the operation and crushing product of this plant, the bottlenecks in the existing processes of crushing and sieving were registered and, in accordance with them, the adequate solutions for achieving the defined objective were proposed: "Increase the capacity of wet ore processing to Q = 10.6 x 10^6 t/annually with the upper size limit of 16 mm", with as low as possible the investments, that is with the existing equipment and according to the existing technological scheme of processing.

This paper presents a part of operation analysis of the aforementioned plant with a proposal of technical solutions that can lead to the realization of set conditions and achieving the capacity of crushing and sieving plant in the copper mine Veliki Krivelj Q = 10.6 x 10^6 t of wet ore annually with the upper size limit of 100(%) – 16 mm.

Key words: *capacity, crushing, sieving, product.*

1. INTRODUCTION

The ore deposit Veliki Krivelj is about 3 km northwest of Bora and at 0.5 km from the nearest village Krivelj, by the air line, in the river basin of the Krivelj River. The open pit Veliki Krivelj is located within the deposit Veliki Krivelj where the exploitation began in 1982. In the immediate vicinity of the open pit, the crushing plant, flotation and other associated facilities were constructed, necessary for exploitation and processing, that is ore dressing by the flotation process. Ore is transported from the open pit to the

* *Mining and Metallurgy Institute Bor-dragan.milanovic@irmbor.co.rs*
** *University of Belgrade, Technical Faculty in Bor*
*** *This paper is the result of the Project No.33023, funded by the Ministry of Education and-Science of the Republic Serbia.*

primary crushing by trucks, while the waste is transported by trucks and combined system of trucks - belt conveyors. After primary crushing, ore passes through the reception bin, where it is transported by conveyor belts to the open yard of the primary crushed ore. From the open yard, the ore is sent by the star feeders and belt conveyor system to the plant for secondary and tertiary crushing with screening [4,3].

Copper Mines – Bor (hereinafter referred to RBB) agreed with the Mining and Metallurgy Institute Bor (MMI Bor) to develop a technical solution to achieve a capacity of ore processing from the ore deposit Veliki Krivelj of 10.6 x 10^6 t of wet ore per year. Regarding to this, it is necessary to check the capacity and condition of existing equipment and provide solutions for achieving the defined processing capacity with the final productof crushing g.g.k.16 mm. Noting that in all attempts to achieve in industrial practice the defined capacity of 10.6 x 10^6 t of run-of-mine ore per year, the satisfactory results were not obtained. Despite the fact, that in these attempts to finess of the final product crushing was requested that it is g.g.k. 20mm.

This is, compared to the present required fineness of the finished product from the crushing g.g.k. 16 mm, represented a far easier option.

The Bond work index of ore from Veliki Krivelj ranges from 12-14 kWh /t and, for further work, the hard ore is adopted at the junction between medium hard and hard ore with the value of the Bond work index of 14 kWh/t, that is kWh/sht.

In the process of finding technical solutions considered all options to the existing technology for ore processing with minimum investment procedures in the stage of secondary and tertiary crushing. (Primary screening, secondary crushing, tertiary crushing in a closed cycle with a secondary sieving) meet the needs of required processing capacity and required fineness of the finished product crushing, (Q = 10.6 x 10^6 t/year g.g.k and 16 mm) [5,6]. So, the Krivelj ore in the system of secondary and tertiary crushing and sieving would be processed by the existing technological scheme of processing with existing equipment, the technical solutions designed to meet the set requirements in terms of achieving the required processing capacity and crushing fineness of finished product. Figure 1 is a panorama of Veliki Krivelj with facilities for ore processing and enrichment.

Figure 1. *Panorama of Veliki Krivelj with facilities for ore processing and enrichment*

2. ACHIEVING THE CAPACITY (Q=10,6 x 10^6 t/year g.g.k. 16 mm)

A prerequisite for achievement the defined goals, (Q = 10.6 x 10^6 t / year g.g.k. and 16 mm) is that all available equipment and facilities are in good and functional condition, revitalized equipment, to the extent that the utilization of time in which the necessary to achieve the planned capacity and quality of processed ore. This time of use the equipment and facilities is expected to be increased from the previous mode when dealing with low processing capacity (from 8.5 or 10.6 x 10^6 t/year, and a higher upper limit of the finished product size from the crushing g.g.k. 20 mm). With the adopted coefficient of time utilization facilities of secondary and tertiary crushing and sieving of k = 0.8, the required horly capacit for ore processing will be: Qh = 1512.56 t/h wet ore. For the same, the verification of plant secondary and tertiary crushing and screening will be carried out, as well as a view of scheme of mass movement. Therefore, to achieve a higher capacity, among other things, the maximum use of the effective time of equipment, working conditions and increasing the allowable maximum loads of equipment will be required. Furthermore, we will not dwell on the time utilization of equipment and facilities, but it must be noted that the projection of the final technical solution the utilization of available fund time was used as one of the important factors of this technical solution [5]. Figure 2, gives an overview of the existing technological scheme with equipment and movement of a mass of system for secondary and tertiary ore crushing and sieving in the mine Veliki Krivelj for the adopted technical solution.

It is seen from Figure 2 that the system of secondary and tertiary crushing and sieving is technologically connected to the primary crushing system. For these reasons, the primary crushing system has to give a product that will best suit to the system of secondary and tertiary crushing and sieving in terms of achieving the required conditions. There, it means that all equipment is in good and functional condition, **the revitalized equipment**, in order to meet the new designed technological conditions [6]. And those are that the product of primary crushing, which is, according to the technological scheme, stored in the open line S-1, as the finest possible product. It, later in the system of secondary and tertiary crushing and sieving, comes first on two-level screen where where the sieving product of the second sieving presents the finished crushing product. So, the share of this product as desirable as possible in the inlet ore that coming into the system of secondary and tertiary crushing and sieving. Due to the given facts, it is necessary to carry out such setting of the primary crusher that will satisfy, on one side, the above conditions and, on the other side, to ensure the satisfaction of capacity and reliability in operation. The grain size distribution of the primary crushing product is taken from the catalogue of the manufacturer and presented in Table 1 according to the adopted outlet of the primary crusher Allis Chalmers 48''x74'' in an open position of **OSS = 139.7 mm ($5^{1/2"}$)** and for the medium hard ore.

Table 1. *Grain size distribution of the outlet from the primary crusher Allis Chalmers 48x74''*

Size range	Outlet OSS =139.7 mm. ($5^{1/2"}$)		
d(mm)	m(%)	R(%)	D(%)
-203.20+190.5	1	1	100
-190.5+177.80	1.5	2.5	99
-177.80+165.10	1.5	4	97.5
-165.10+152.40	2	6	96
-152.40+139.70	4	10	94
-139.70+127	5	15	90
-127+114.30	7	22	85
-114.30+101.60	5.5	27.5	78
-101.60+88.90	8.5	36	72.5
-88.90+76.20	8	44	64
-76.20+63.50	8.5	52.5	56
-63.50+50.80	7.5	60	47.5
-50.80+45	4.51	64.51	40
-45+38.10	4.49	69	35.49
-38.10+31.75	3.50	72.5	31
-31.75+25.40	5	77.5	27.5
-25.40+19.05	3.5	81	22.5
-19.05+16	2.14	83.14	19
-16+12.70	1.86	85	16.86
-12.70+6.35	4	89	15
6.35+0	11	100	11

The presented grain size distribution of the primary crushing product is inlet into to the secondary and tertiary crushing and sieving plant.

Inlet to the primary two-level sieve.

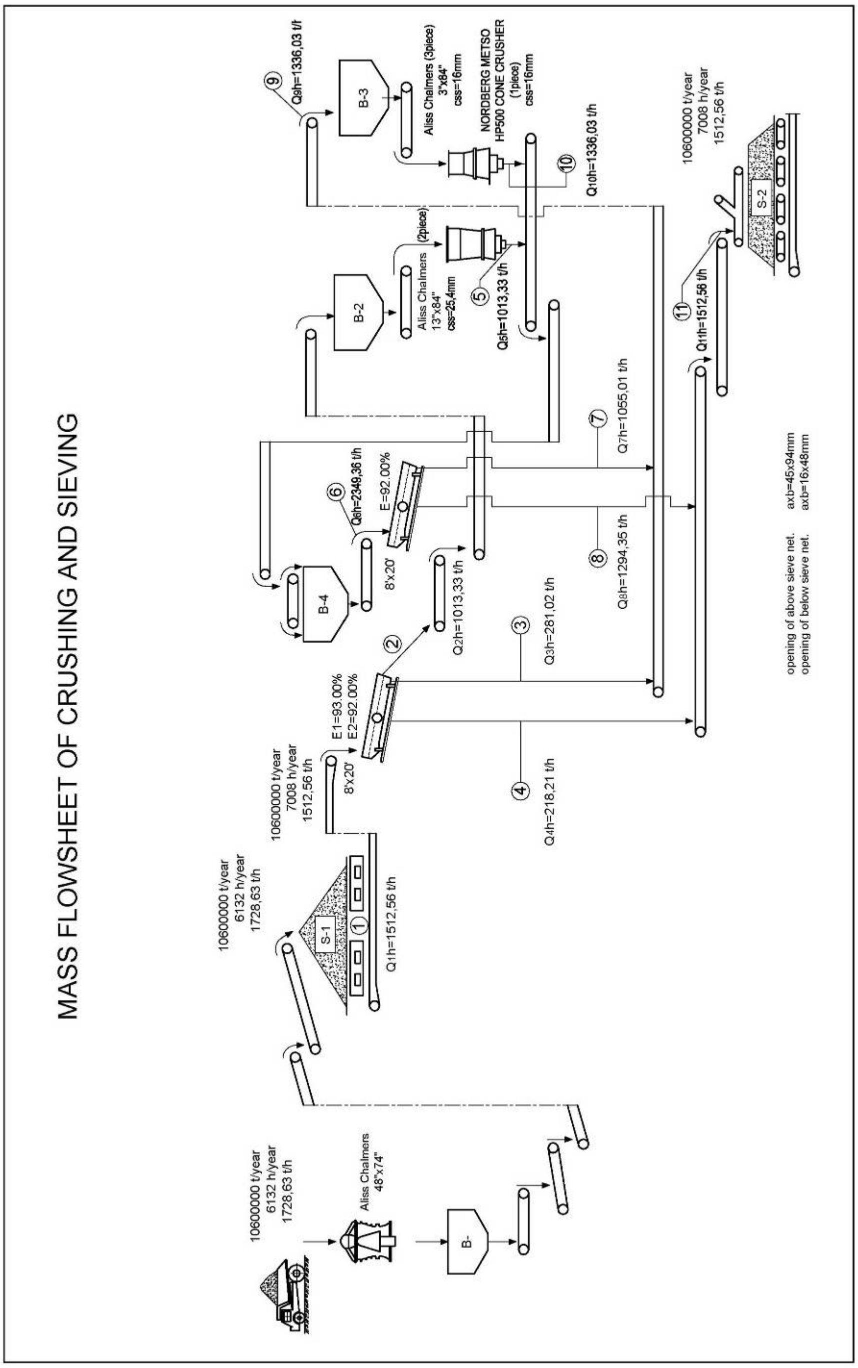


Figure 2. *Scheme of mass movement in the secondary and tertiary crushing and sieving*

It is seen from the grain size distribution that the input ore, in the plant of secondary and tertiary crushing and sieving, has a size class of g.g.k. 100% -203 mm, and a participation of size class of -20 mm is approximately α^{-20}= 20%, while the participation of size class of -16 mm is α^{-16}= 16.86 %. It is also seen that the participation of larger size classes is as follows: -63.5 mm $\alpha^{-63,5}$= 47.5 %, while itis for the -45 mm size class α^{-45}= 35.49 %.

For all and especially for highly loaded equipment, which is in such system of crushing, it is primarily expected to be the tertiary crushers and secondary screens for sieving the final product of crushing, and verification will be conducted to determine the extent of their capabilities and capacitive load.

For secondary crushing in the plant Veliki Krivelj, two (in the previous period, installed and validated in ooperation) **revitalized** secondary crusher type: Allis Chalmers Hydrocone EHD, size: 13’’ x 84’’ with ECC: 2’’ (50.8 mm) are provided. Figure 3 presents the above mentioned crusher which is installed in the building of the secondary crushing.

For tertiary crushing in the plant Veliki Krivelj, three (in the previous period, installed and validated in operation) **revitalized** tertiary crusher type: Allis Chalmers Hydrocone EHD, size: 13’’ x 84’’with ECC: 2’’ (50.8 mm) are provided and a new ”metso-minerals“ „*Nordberg*“ HP 500 Series Cone Crushers. Figures 4 and 5, respectively, present the above crushers.

Figure 3. *Secondary crushers „Allis Chalmers“ Hydrocone EHD, 13’’x 84’’*

Figure 4. *Tertiary crushers „Allis Chalmers" Hydrocone EHD, 3''x 84''*

Figure 5. *Tertiary crusher „Nordberg" HP Series Cone Crushers, HP 500, „metso-minerals"*

Table 2 presents the expected particle size distribution of secondary crushed ore per Allis Chalmers, for the adopted secondary crusher outlet Hydrocone EHD, catalogue number B 223 025 E, size: 13x84'' at CSS = 1'' (25 mm) and eccenter of crusher ECC = $1^{1/4}$'' (32 mm) (maximum tightening of a crusher), for the medium hard raw materials (Wi = 14 kWh / t, 15.43 kWh / sht). From it is seen that ore has g.g.k. over 45 mm, or 100%-50mm, and a size class participation - 20 mm is about α^{-20}= 44 %, while the participation of size class is approximately $\boldsymbol{\alpha^{-16}}$ **= 36%**.

Necessary information about the granulometric composition of the product of tertiary crushing are adopted from the catalogue of the „metso-minerals", and the newly purchased HP 400 crusher. Assuming that after the planned reconstruction of the existing crusher: „Allis Chalmers" Hydrocone EHD, 3''x 84'' sa ecc: 2'' (50.8 mm) they will give the same products as the newly purchased crusher. And therefore, they will also have the same particle size distribution. The expected particle size distribution of products from catalogues of the „Nordberg“, HP 500, „metso-minerals“ crusher, number of brochures No.: 2051-04-07-CBL/Tampere-English with eccenter, ecc according to the Standard of manufacturer for this type of crusher, at a given tensile of a crusher CSS = 16 mm for the medium hard raw materials (Wi = 14 kWh/t, 15.43 kWh/sht), is given in Table 3.

Based on previous analysis of opeartion the system of secondary and tertiary crushing and sieving of the mine Veliki Krivelj, when in a given system in the addition of different settings of crushers, different network of screens were installed (here is primarily related to the size of surface mesh, respectively to the sieve level, axa = 60x60 mm and axb = 20x40 mm), overloading was always detected of tertiary crushing and secondary screening, (secondary mesh sieve with reduced efficiency) [3,4]

Also, during such mass distribution with the mesh sizes, there is deloading of secondary crushing causing imbalance in the entire system of crushing and sieving.

Table 2. *Grain size distribution of the crushed product of the secondary crusher „Allis Chalmers“ Hydrocone EHD, size:13’’x84’’, ecc=32 mm (11/4’’) per catalogue*

Size class	Outlet CSS =25.4 mm, (1’’)		
d (mm)	m(%)	R(%)	D(%)
-50+40	8	8	100
-40+30	20	28	92
-30+25	12	40	72
-25+20	16	56	60
-20+16	8	64	44
-16+12	10	74	36
-12+10	4	78	26
-10+5	12	90	22
-5+0	10	100	10

Table 3. *Grain size distribution of the crushed product of the tertiary crusher „Nordberg“, HP 500, „metso-minerals“ per catalogue*

Size class d (mm)	Outlet CSS =16 mm, (5/8’’)		
	M (%)	R (%)	D (%)
-25+20	10	10	100
-20+16	12	22	90
-16+12	16	38	78
-12+10	9	47	62
-10+5	23	70	53
-5+0	30	100	30

From the foregoing, and to meet the given conditions to achieve the required processing capacity with the required fineness of the finished product crushing, Q = 10.6 x 10^6 t/year g.g.k. and 16 mm, in development of this technical solution preparation, it will be introduced in relation to the former, the changed technological and technical parameters of ore processing. In this sense, it is noted that under this implies a significant reconstruction of the three tertiary crushers type Allis Chalmers Hydrocone EHD, 3’’x84’’ sa ecc: 2’’ (50.8 mm) (replacement of eccenter and replacement of existing steel linings, that the introduction of a new profile and lining of crushers and IC system of automatic control (Intelligent Control) 7000) and acquisition of a new tertiary crushers type „Nordberg“ HP Series Cone Crushers, HP 500, company „metso-minerals“. Reconstructed existing crushers type Allis Chalmers EHD should give similar or the same particle size distribution, crushing ore product (grain size distribution of grushing product per catalogues) as well as the new HP 500 crusher. Then the necessary and reconstruction of screens, which implies a change in surface mesh screens that introduce mesh surface sieves with different openings, i.e., the dimensions of the hole mesh sieve. (Instead of the existing nets of screens, with dimensions of rectangular openings axa = 60 x 60 mm and iaxb=20x40 mm, introducing the dimensions of the rectangular hole mesh sieve axb = 45x94 mm and axb=16x48 mm, respectively, by adequate levels of the screen). All this, with a given setting with an expected crushing crushing of the same products (Tables 2 and 3), and the finer the entrance to the system of crushing and sieving, finer primary crusher product (table 1), and the reconstructed particle size distribution, which calculated for certain essential products that due to the overvolume of this paper will not be displayed, made it possible to obtain mass balance that has already been presented in this paper (Figure 2).

In this way, more even balance of movement the masses in the site is obtained. Redistribution of the mass was performed and further burden the secondary crusher to increase the allowed maximum allowed loading of tertiary crushers. A part of mass that previously went to the tertiary crushing, now goes to the secondary crushing. This allows the operation of secondary sieving in the limits of maximum technological capabilities to achieve the given needs in this closed system, tertiary crushing-secondary screening.

It can now be expected to present the final product particle size distribution of crushing, which is given in Table 4, which show that the expected final product meet the crushing demand in terms of granulometry.

Table 4. *Grain size distribution of the crushing and sieving final product*

Size class	Final product of crushing and sieving		
d(mm)	m(%)	R(%)	D(%)
-16+12	21.12	21.12	100
-12+10	10.94	32.06	78.88
-10+5	30.06	62.12	67.94
-5+0	37.88	100	37.88

3. THE EFFECT ON THE MASS BALANCE

We have now the data that can present qualitatively and quantitatively the effects of tadopted design to achieve the given values of capacity and product fineness.

Table 5. *Comparative review of the contents of typical class sizes and capacity at the suitable positions in the system of crushing and sieving and according to the mash size of sieving surfaces*

Grain size distribution of input into the crushing system on the primary sieve							
Grain size content α^{+45}mm			α^{+45}mm 64.51(%)		Grain size content α^{+60}mm		α^{+60}mm 55.20(%)
Product		Inlet on primary screen Q_1	Inlet on II level of primary screen Q_2'	Undersize of I level screen Q_2= Q_5	Undersize of II level screen Q_3	Oversize of II level screen Q_4	Product of secondary crushing Q_5
Partic. (t/h)	# 45 # 16 mm	512.56	499.23	1013.33	281.02	218.21	γ_m (-203.2+ 45) mm 1013.33
Prtic. (%)	# 45 # 16 mm	100	33	67	18.58	14.42	γ_m (-203.2+ 45) mm 67

Partic. (t/h)	# 60 # 20 mm	1512.56	630.19	882.37	371.32	258.87	γ_m (-203.2+ 60) mm 882.37
Partic.(%)	# 60 # 20 mm	100	41.66	58.34	24.55	17.11	γ_m (-203.2+ 60) mm 58.34
Product		**Inlet on sec.screen Q_6= Q_{10}+ Q_5**	**Oversize sec.screen Q_7**	**Undersize sec.screen Q_8**	**Inlet tert.crushing Q_9= Q_3+ Q_7**	**Proizvod terc.drob Q_{10}= Q_9**	**Final crushing product Q_{11}= Q_8+ Q_4**
Partic.(t/h)	# 45 # 16 mm	2349.36	1055.01	1294.35	1336.03	1336.03	1512.56
Partic. (%)	# 45 # 16 mm	155.33	69.75	85.58	88.33	88.33	100
Partic.(t/h)	# 60 # 20 mm	1965.11	711.42	1253.69	1082.74	1082.74	1512.56
Partic. (%)	# 60 # 20 mm	129.96	47.07	82.89	71.62	71.62	100

Those data are presented in Table 5 and related to the review of account size class α^{+xx} (%) of mass distribution of the characteristic size class γm (%) and capacity of screens and crushers. Q_1-Q_{11} (t/h) for the adopted solution, that is the adopted value of sieving areas openings of the primary and secondary screens.

Earlier, it was stated that the existing mass imbalance, the „bottleneck“ of the mass distribution in a given technological line, can be eliminated By the change of size the screening areas of the primary screen.

In this way, the balanced loading of all crushing aggregates is achieved and better use of the available capacity, i.e, the possibility of the same.

The first line of this Table prsents a comparative view of content the specific size class in the grain size distribution of the input in the crushing system on the primary screen. Only based on the grain size distribution, an evident difference is seen between the mass content when it is a size class (-g.g.k. +60) mm or size class (–g.g.k.+45) mm. This is directly transferred to the mass distribution γm (%) = 1013.33 t/h respectively γm (%) = 882.37 t/h of sieve oversize from the first stage of the primary screen size class: (-203+60) mm (- 203+45) mm respectively in the previous, i.e. the adopted solution here.

Finally, in further view of thisTable 5., and based on data from the mass movement scheme, itis seen the change in the secondary crushing capacity - quantitative change in the mass balance.

This led, with changing the mesh size of the lower surface screening area of the primary two-stage screen and the secondary one-stage scree, and adjustment the and tertiary crushing product by the given setting of crusher to achieve the required capacity of the crushing system of Q = 10.6 x 10^6 t/year and the required product fineness of g.g.k. 100 % (-16) mm.

4. VERIFICATION OF THE CAPACITY

Due to achieve the permitted maximum load process equipment, it is necessary for all, and especially for the highly loaded equipment (where according to the mass movement scgeme it is expected to be the tertiary crushers and secondary screens for sieving the final product of crushing), to perform a verification in order to determine a degree of their capabilities and loads in a such system of crushing and according to such distribution of mass movements [1,2].

Checking the capacity of screens was carried out according to the Mehanobr method and based on data contained in the scheme of mass movement. Throughput capacity of surface mesh is in the range of technological possibilities for achieving the given requirements. For these reasons. the technological process of screening must be constantly maintained at the given designed levels. Otherwise, there may be disturbances of screens.

Further checking the capacity of existing equipment in the secondary and tertiary crushing showed that most of the equipment can meet the most anticipated new technological capabilities and requirements in general. The realization of constant design capacity of Q=1512.56 t/h, will depend exclusively on the crushers in operation, that is their partially achievable capacity. For specified technological conditions, in which it is provided that the secondary crushers of this size, Allis-Chalmers Hydrocone crusher **EHD** 13“ x 84“ (330.20 x 2133.6 mm) are set to CSS=25.4 mm (1"), then having the capacity (according to the catalogue of manufacturer company Allis Chalmers number B 223 025 E) of Qkat = 540 t/h dry ore. and the tertiary crusher Allis **Chalmers HYDROCONE - EHD.** 3’’x84’’ (76.2x2133.6 mm) are set to CSS=16 mm (5/8"), having a capacity during this (according to the catalogue Allis Chalmers. No. 17 B 5239) of Qkat = 471.64 t/h (520 sht/h) dry ore, that is tertiary crusher Nordberg HP 500 CONE CRUSHERS company Metso-Minerals has (according to the catalogues data No. NO: 2051-04-07-CBL/Tampere-English) capacity within the limits of minimum $\boldsymbol{Qkat_{min}= 280}$ ***t/h***, that is the maximum of $\boldsymbol{Qkat_{max} = 350}$ ***t/h*** dry ore.

Two secondary crushers, according to the scheme mass movement, have to overcome the hourly capacity of secondary crushing of $\boldsymbol{Qh_{sec} = 1013.33}$ ***t/h***, i.e./2=506.66 t/h wet ore, that is $\boldsymbol{Qh_{sec}=483.86}$ ***t/h*** dry ore per one crusher. As the required capacity of the secondary crushing per one crusher is less than the catalogue capacity $\boldsymbol{Qh_{sec}=483.86}$ ***t/h*** < ***Qkat=540 t/h*** dry ore, it completely meets the newly arisen needs for the planned adjustment CSS=25.4 mm (1") of crusher.

Three tertiary crushers **Allis Chalmers HYDROCONE EHD.** 3’’x84’’ and a new **Nordberg HP 500 CONE CRUSHERS company „metso-minerals“** should, according to the same scheme of mass movement to cope with hourly capacity of the tertiary crushing $\boldsymbol{Qh_{terc} = 1336.03}$ ***t/h,*** i.e. per one tertiary crusher: ***Qh=334 t/h*** wet ore and ***Qh= 318.97 t/h*** dry ore.

The required reconstruction of these crushers can lead to an important change of capacity of these devices. Capacity of each cone crusher for medium and fine crushing was derived based on the theorem of Gulden for the ring volume of working space of crusher. After required reconstruction of our crushers, it can be assumed that the company „metso-minerals“ will retain, in the reconstruction, the old catalogue capacities of these crushers. That is, regarded to the theoretical expressions for optimum r. p. m. of eccentric bushings $\mathbf{n_0}$ and crusher capacity Q, can be-achieved by determination the characteristic values of key technical parameters that directly affect the capacity of crusher (profile change of protective linings, that is the ring volume, Working space of crusher, eccentric of crushers, r.p.m. of eccentric bushing, etc.), and that, during this, the required capacity and the certain grain size distribution of crushing product are obtained.

As the required hour capacity of tertiary crushing per one crusher is less than maximum catalogue capacity ***Qh = 318.97 t/h***<$\boldsymbol{Qh_{catalogues}}$***=471.64 t/h*** dry ore crusher of this type of crusher, we conclude that Allis Chalmers HYDROCONE EHD crushers fit the newly appeared needs of crushing, i.e. meeting the defined capacities.

The same happens with the tertiary crusher HP 500 th. As in the previous case. the same hour capacity of tertiary crushers is needed ***Qh= 318.97 t/h*** dry ore, and the same is also less than the maximum catalogue capacity of this type of crusher (Nordberg HP 500 CONE CRUSHER). ***Qh=318.97 t/h*** < $\boldsymbol{Qkat_{max}}$ ***= 350 t/h*** dry ore and this crusher suits the newly designed operating conditions.

5. CONCLUSION

To achieve the capacity of ore processing from the deposit Veliki Krivelj of $10.6x10^6$ tonnes of wet ore per year with g.g.k. 16 mm, the Krivelj ore in the system of secondary and tertiary crushing and sieving should be processed per the existing technological flowsheet with the existing equipment with the planned technical solution that will meet the set requirements.

This technical solution was realized through several offensive points as follows:

- To increase the time utilization of operation the equipment and aggregates at a higher required level.
- That all the available equipment and facilities are in good and functional condition, revitalized equipment.
- That the primary crushing product is the finest possible product of primary crushing.
- That all the settings of crusher are set according to the given statements in this paper to get the particle size distribution of crushing and sieving product as shown here.
- To replace the worn tertiary crusher with a new HP 500 „metso-minerals" crusher.
- That the reconstruction of the existing tertiary Allis Chalmers HYDROCONE EHD. 3''x84'' is carried out according to the statements in this paper so that they meet the required capacity and granulometry of crushing product.
- To introduce the sizes of rectangular hole of sieve mesh **axb=45x94 mm and axb=16x48 mm**, respectively, by the adequate sieves and levels of screens.

REFERENCES

[1] Technological Basis in Designing the Plant for Mineral Processing, Mining Institute, Belgrade, 1999 (in Serbian)

[2] N. Magdalinović, Comminuting and Classification Belgrade, 1999 (in Serbian)

[3] AMP Reconstruction of the Veliki Krivelj Flotation Plant to the aim of Increase the Processing Capacity to 10.6 x 106 t per Year of Run-of-Mine Ore – Mining Institute, Belgrade, 1995 (in Serbian)

[4] AMP of Increase the Annual Ore Processing in the in the Veliki Krivelj Flotation Plant to 10.6 million t of Run-of-Mine Ore, Technological Project – Copper Institute Bor,1997 (in Serbian)

[5] Proposal of Technical – Technological Solution for Decrease the g.g.k. of the Ore Crushing Product from the Veliki Krivelj to the aim of Decrease the Costs of Comminuting and Increase the Capacity of Ore processing, Copper Institute Bor – Mining Institute Belgrade, January 1993, (in Serbian)

[6] Feasibility Study on Investments into the Copper Concentrate Production at the Open Pit and Flotation Plant Veliki Krivelj, RTB Bor, Belgrade, March 2010, Geo-in International Belgrade (in Serbian)

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:622.261.2(045)=861

Duško Đukanović, Miodrag Denić,** Dušan Dragojević,**

BRZINA IZRADE PODZEMNIH PROSTORIJA, KAO USLOV UVOĐENJA MEHANIZOVANE IZRADE PODZEMNIH PROSTORIJA U RUDNICIMA JP PEU RESAVICA

Izvod

U rudnicima JP PEU Resavica, podzemne prostorije se trenutno izrađuju bušačko-minerskim radovima. Postojeći način izrade, podzemnih prostorija, ne zadovoljava po pitanju brzine i troškova izrade. Ostvarene brzine izrade podzemnih prostorija, primenom bušačko-minerskih radova su veoma male. U rudnicima JP PEU, prevashodno u rudnicima „Lubnica“, „Soko“ i „Rembas“, projektovano je otvaranje novih otkopnih polja i jama, pri čemu je ukupna dužina projektovanih prostorija oko 26.000 m. U koliko bi se projektovane prostorije radile samo bušačko-minerskim radovima, došlo bi do značajnog zakašnjenja u otvaranju novih proizvodnih kapaciteta, a samim tim i do ne ispunjavanja zadatog cilja-povećanja proizvodnje uglja.

Iz ovog razloga razmatra se mogućnost uvođenja mehanizovane izrade podzemnih prostorija, u okviru ovog rada razmatrana je brzina izrade podzemnih prostorija, kao jedan od uslova uvođenja mehanizovane izrade podzemnih prostorija u rudnicima JP PEU Resavica.

Ključne reči: *brzina izrade podzemne prostorije, rudnik, mašina, ugalj.*

UVOD

Podzemne prostorije u rudnicima JP PEU, se trenutno izrađuju bušačko-minerskim radovima.

Klasičan način izrade podzemnih prostorija ne omogućava zadovoljavajući učinak, jer se tehnološke operacije bušenja, miniranja i utovara izvode ručno i zahtevaju nedopustivo velike utroške vremena. S obzirom na ovu činjenicu, kao i na ranije velike zaostatke na otvaranju i pripremi novih podzemnih proizvodnih kapaciteta, postojeći način njihove pripreme vodi stagnaciji razrade jama i ne omogućava povećanje proizvodnje uglja.

U cilju povećanja proizvodnje uglja i otvaranja novih jama i otkopnih polja, neophodno je razmotriti primenu produktivnijeg načina izrade podzemnih prostorija, tj. primenu mašina za izradu podzemnih prostorija.

Čak i bez detaljne analize se može konstatovati da bi uvođenje mehanizovane izrade, značajno unapredilo postupak izrade podzemnih prostorija u rudnicima JP PEU, pogotovo u smislu povećanja brzine izrade i unapređenja sigurnosti i bezbednosti radnika.

* *JP za PEU, Biro za projektovanje i razvoj Beograd*
** *JP PEU Resavica, Sektor za investicije, projektovanje i tehnološki razvoj*

U okviru ovog rada određena je brzina izrade podzemnih prostorija, pri kojoj su troškovi izrade isti kod obe tehnologije, tj. određena je minimalna brzina izrade podzemne prostorije, koja se mora ostvariti kod primene mehanizovane izrade.

ODREĐIVANJE MINIMALNE BRZINE IZRADE PODZEMNIH PROSTORIJA

U ovom radu razmatrana je mogućnost primene mašina za izradu podzemnih prostorija u rudnicima Lubnica, Soko, Rembas (jama Strmosten i jama Ravna Reka IV blok). Za svaki rudnik, odabrane su karakteristične radne sredine, oblici i površine poprečnog preseka prostorija (tabela 1.).

Tabela 1. *Karakteristične radne sredine, oblici i površine poprečnog preseka prostorija*

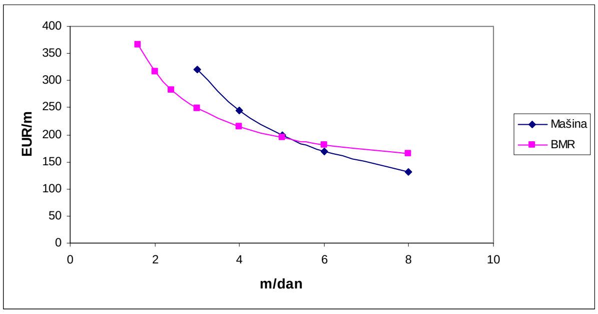
Rudnik	Radna sredina	Čvrstoća na pritisak (MPa)	Oblik prostorije	Svetli profil (m^2)	Iskopni profil (m^2)
Lubnica	Jalovina	≈10	Lučni	10,6	12
Soko	Jalovina	≈52	Lučni	14,5	16,5
	Ugalj	≈22	Kružni	9,6	11,4
Rembas Strmosten	Jalovina	≈40	Kružni	9,6	11,4
Rembas R. Reka IV	Jalovina	≈66	Kružni	9,6	11,4
	Ugalj	≈8	Kružni	9,6	11,4

U cilju određivanja minimalne brzine izrade, tj. brzine izrade podzemne prostorije pri kojoj su troškovi izrade isti kod obe tehnologije, ustanovljena je određena metodologija proračuna troškova izrade. Proračunom troškova izrade obuhvaćeni su samo direktni troškovi izrade podzemne prostorije, a koji se sastoje iz sledećih vrsta troškova: troškovi radne snage, troškova normativnog materijala, troškova energije i troškova sredstava za rad (opreme).

Kako cilj ovog rada nije da se odrede ukupni troškovi izrade prostorije, već da se dodredi minimalna brzina izrade podzemnih prostorija mašina u našim rudnicima, kod proračuna direktnih troškova izrade prostorija, nisu proračunavati oni troškovi koji su zajednički i kod tehnologije izrade prostorija bušačko-minerskim radovima. To se prvenstveno misli na troškove podgradnog materijala, provetravanja i drugih normativnih troškova koji su zajednički za obe tehnologije.

U cilju poređenja tehnologija izrade podzemnih prostorija bušačko-minerskim radom i mašinama, a na osnovu ustanovljene metodologije proračuna troškova izrade, određeni su troškovi izrade u zavisnosti od brzine izrade, za odabrane anlizirane radne sredine. Kod određivanja troškova izrade mašinama uzeta je nabavna vrednost mašine od 700.000 EUR.

Na osnovu izvršenih proračuna troškovi izrade u zavisnosti od brzine izrade, za svaku analiziranu radnu sredinu, sačinjen je dijagram (slika 1.) zavisnosti troškova i brzine izrade podzemnih prostorija, sa kojeg je očitana vrednost brzine izrade podzemne prostorije pri kojoj su troškovi izrade isti kod obe tehnologije (tabela 2.). Na slici broj 1. dat je dijagram zavisnost troškova i brzine izrade podzemne prostorije tehnologijom bušačko-minerskog rada i mašinom u radnoj sredini 4.

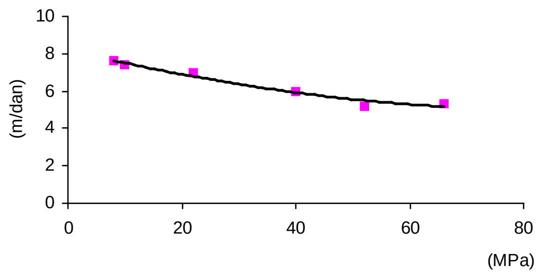


Sl. 1. *Zavisnost troškova i brzine izrade podzemne prostorije tehnologijom BMR i mašinom u radnoj sredini 4*

Tabela 2. *Vrednost brzine izrade podzemne prostorije pri kojoj su troškovi izrade isti kod obe tehnologije*

Rudnik	Radna sredina	Čvrstoća na pritisak (MPa)	Brzina izrade (m/dan)
Lubnica	Jalovina	≈10	7,4
Soko	Jalovina	≈52	5,2
	Ugalj	≈22	7,0
Rembas - Strmosten	Jalovina	≈40	6,0
Rembas - R. Reka IV	Jalovina	≈66	5,3
	Ugalj	≈8	7,6

Na osnovu navedenih podataka o potrebnoj brzini izrade podzemnih prostorija (tabela 2.), sačinjen je dijagram zavisnosti čvrstoće radne sredine i brzine izrade podzemne prostorije za analizirane rudnike (slika 2.), kao i analitička zavisnost (obrazac 1.).



Sl. 2. *Grafički prikaz zavisnosti čvrstoće radne sredine i brzine izrade podzemne prostorije*

Zavisnost čvrstoće radne sredine i brzine izrade podzemne prostorije, data je i obrascem:

$$V = 0{,}0004\sigma^2 - 0{,}0731\sigma + 8{,}1897 \quad \left(R^2 = 0{,}9687\right) \qquad (1)$$

Sa dijagrama prikazanog na slici broj 1., se vidi da se sa povećanjem čvrstoće stenske mase, smanjuje i minimalna brzina izrade podzemne prostorije, a obrazac (1) nam omogućava da na osnovu čvrstoće radne sredine, odredimo minimalnu brzinu (m/dan), sa velikom preciznošću.

ZAKLJUČAK

I ako je opšete poznato da se primenom mehanizovane izrade postižu veće brzine izrade, u ovom radu smo odredili brzinu izrade koja se mora ostvariti kod primene mehanizovane izrade podzemnih prostorija u uslovima rudnika JP PEU. Na osnovu izvršene analize može se reći da je potrebna minimalna brzina izrade prostorija mašinom 6,85 m/dan.

LITERATURA

[1] Vidanović N, Đukanović D., Dragosavljević Z.; Inovation of technology of construction of underground mininig workings by use drilling and blasting methods of work, Technics Technologies Education Management, TTEM, Vol. 5, No.4, pp. 861–866; 2010.

[2] Đukanović D., Đukanović D. Savić Lj.; Analiza ostvarenih tehničkih parametara pri izradi podzemnih prostorija u rudnicima uglja u Srbiji, Podzemni radovi 13, str. 17-22, RGF, Beograd, 2004.

[3] Đukanović D.; Model optimizacije tehno-ekonomskih pokazatelja pri izradi podzemnih prostorija u rudnicima uglja Srbije, Doktorska disertacija, RGF, Beograd, 2005.

[4] Đukanović D.; Tehnologija izrade jamskih prostorija kombinovanim mašinama sa osvrtom na mogućnost primene u rudnicima uglja Srbije. Savez energetičara, Beograd, 143 strane; 2005.

[5] Đukanović D., Đukanović D.; Analiza zavisnosti ostvarenih troškova i brzine izrade podzemnih prostorija u rudnicima uglja u Srbiji, Rudarski radovi 01/2005, str. 51-55, Komitet za podzemnu eksploataciju mineralnih sirovina, Bor, 2005.

[6] Đukanović D.; Istraživanje strukture troškova pri izradi podzemnih prostorija u rudnicima uglja u Srbiji, (Investigation of the costs structure in development the underground rooms in the coal mines in Serbia), Rudarski radovi 02/2005, Komitet za podzemnu eksploataciju mineralnih sirovina, Bor, str. 81-88. 2005.

[7] Đukanović D.; Possibilities for application of contemporary technologies production of shafts premises in coal mines in Serbia, 39th International October conference on mining and metallurgy, 07 - 10. October, Sokobanja, University of Belgrade, Tehnical Faculty Bor and Copper institute Bor, Serbia, str. 39-43; 2007.

[8] Đukanović D., Denić M., Dragojević D.; Modernizacion of tehnological process of the construction of shaft premises in coal mines of Serbia, Internacional Mining Forum 2009 – Deep Mining Challenges – I Polish-Serbian forum, 18-21. February, Krakow, str. 69-72; 2009.

[9] Đukanović D.; Studija opravdanosti mehanizovane izrade rudarskih prostorija u rudnicima JP PEU; Biro za projektovanje i razvoj, Beograd, 75 strana, 2010.

[10] Milisavljević V., Đukanović D.; Present situation and perspective of roadways development in underground coal mines in Serbia, Technicka Diagnostika, str. 253-257, TU, Ostrava, 2005.

[11] Tokalić R., Trajković S., Đukanović D.; Analiza postignutih učinaka na izradi podzemnih prostorija u rudnicima uglja u Srbiji, Podzemni radovi 11, str. 33-38, RGF, Beograd, 2002.

[12] Trajković S., Đukanović D., Lutovac S.; analysis of technical parameters impact at advance rate of roadway development in Serbian coal mines, 2nd Balkan Mining Congress, (BALKANMINE 2007), 10. do 13. 09., Beograd, Academy of Engineering Sciences of Serbia and Faculty of Mining and Geology University Belgrade, 143-148; 2007.

MINING AND METALLURGY INSTITUTE BOR YU ISSN: 1451-0162
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS UDK: 622

UDK: 622.261.2(045)=20

Duško Đukanović, * *Miodrag Denić,* ** *Dušan Dragojević,* *

DRIVAGE RATE OF UNDERGROUND ROOMS, AS A CONDITION OF INTRODUCTION THE MECHANIZED DRIVAGE OF UNDERGROUND ROOMS IN THE JP PEU RESAVICA MINES

Abstract

In the JP PEU Resavica Mines, the underground rooms are currently driven by the drilling-blasting works. The current way of drivage the underground rooms does not meet the rate and costs of drivage. The realized drivage rates by the use of drilling-blasting works are very small. In the JP PEU Mines, primarily in the mines "Lubnica", "Soko" and "Rembas", opening of new mining fields and pits was designed, where the total length of designed rooms is about 26,000 m. If the designed rooms would be driven by the drilling-blasting works, a significant delay in opening the new production capacities will be, and therefore the designed increase of coal production would not be met. Due to this reason, there is possibility for introduction the mechanized drivage of underground rooms. This work gives a consideration on drivage rate of underground rooms as one of the conditions for introduction the mechanized drivage of underground rooms in the JP PEU Resavica Mines.

Key words: *drivage rate, underground room, mine, machine, coal*

INTRODUCTION

Underground rooms in the JP PEU Mines are currently driven on by the drilling-blasting works. The classical way of drivage does not provide the satisfactory effect because the technological operations of drilling, blasting and loading are performed manually and require unacceptably large time consumptions. Regarding to this fact and previously large backlogs in the opening and preparation of the new underground production capacities, the existing method of their preparation leads to a stagnation of development the pits and prevents the increase of coal production.

In order to increase the coal production and opening of new pits and mining fields, it is necessary to consider the use of more productive way of drivage the underground rooms, i.e. the use of machines for drivage the underground rooms.

Even without a detailed analysis, it can be concluded that the introduction of mechanized drivage would significantly improve the drivage process in the JP PEU Mines, especially in terms of increasing the drivage rate and improvement of security and safety of workers.

The drivage rate of underground rooms was determined within this work with the same costs in both technologies, i.e. minimum drivage rate was determined that has to be realized by the use of mechanized drivage.

* *PC for Underground Exploitation Resavica, Bureau for Design and Development, Belgrade*
** *PC for Underground Exploitation Resavica, Sector for Investments, Designing and Technological Development*

DETERMINING THE MINIMUM DRIVAGE RATE OF UNDERGROUND ROOMS

The use of machines for drivage of underground rooms was considered in this work in the mines Lubnica, Soko, Rembas (Strmosten pit and Ravna Reka IV block pit). The characteristics of working environment, forms and cross-sectional areas of rooms were selected for each mine (Table 1).

Table 1. *Typical working environment, forms and cross-sectional areas of rooms*

Mine	Working environment	Compressive strength (MPa)	Room shape	Bright cross-section (m^2)	Excavation profile (m^2)
Lubnica	Waste rock	≈10	Arched	10.6	12
Soko	Waste rock	≈52	Arched	14.5	16.5
	Coal	≈22	Circular	9.6	11.4
Rembas Strmosten	Waste rock	≈40	Circular	9.6	11.4
Rembas R. Reka IV	Waste rock	≈66	Circular	9.6	11.4
	Coal	≈8	Circular	9.6	11.4

In order to determine the minimum drivage rate, i.e. the rate of development the underground room with the same costs in both technologies, a specific methodology for calculation the production costs was established. The production costs include only direct production costs of drivage and which consist of the following types of costs: labor costs, costs of normative material, energy costs and equipment costs.

Since the objective of this study is not to determine the overall costs for drivage, but to determine minimum drivage rate of underground rooms using the machines in our mines, and in calculation the direct costs of drivage, those costs that are common were not calculated also for technology of drivage using the drilling-blasting operations. This primarily refers to the costs of supporting material, ventilation and other normative costs that are not common to both technologies.

In order to compare the technologies of drivage the underground rooms using the drilling-blasting operations and machines, and based on the established methodology of calculation the costs of production, the costs of production were determined depending on the rate of development for selected analyzed working environment. In determining the costs of machines, the purchase price of the machine of 700,000 EUR was taken.

Based on the realized calculation of the production costs depending on the rate of development, a diagram was made (Figure 1) for each analyzed working environment depending on the costs and drivage rate, from which the rate of development was reads where the costs of drivage are the same in both technologies (Table 2). Figure 1 gives a diagram of dependence the costs and drivage rate of underground rooms using the technology of drilling-blasting and machine in the working environment 4.

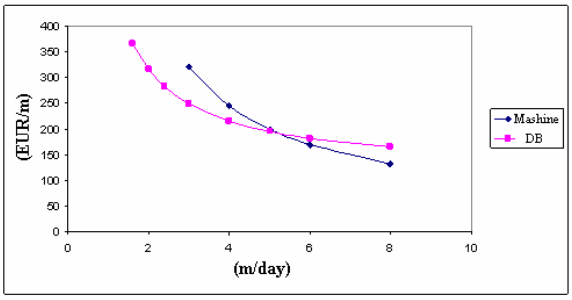


Figure 1. *Dependence of costs and drivage rate using the technology of drilling - blasting and machine in the working environment 4*

Table 2. *Value of drivage rate of underground space with the same production costs for both technologies*

Mine	Working environment	Compressive strength (MPa)	Speed development (m/day)
Lubnica	Waste rock	≈10	7.4
Soko	Waste rock	≈52	5.2
	Coal	≈22	7.0
Rembas Strmosten	Waste rock	≈40	6.0
Rembas R. Reka IV	Waste rock	≈66	5.3
	Coal	≈8	7.6

Based on data on the required drivage rate of underground rooms (Table 2), a diagram of dependence the strength of working environment and drivage rate was made for the analyzed of mines (Figure 2), as well as the analytical dependence (Form 1).

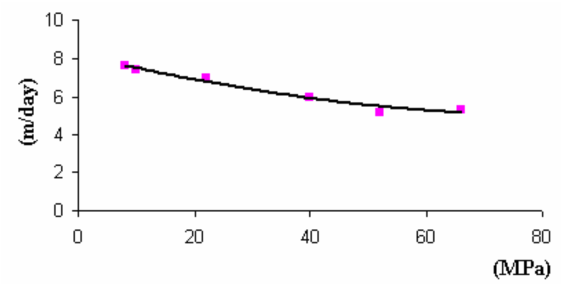


Figure 2. *Graphical presentation of dependence the strength of working environment and drivage rate*

Dependence the strength of working environment and drivage rate is given by the following form:

$$V = 0.0004\sigma^2 - 0.0731\,\sigma + 8.1897 \quad (R^2 = 0.9687) \quad (1)$$

It is seen from diagram in Figure 1 that with increasing the strength of rock mass, minimum drivage rate is decreased, and the form (1) enables to determine minimum rate (m/day) with high precision based on the strength of working environment.

CONCLUSION

Even it is commonly known that the use of mechanized production results into higher drivage rates, this paper gives a determination of drivage rate that has to be achieved using the mechanized production of underground rooms in the conditions of the JP PEU Mine. Based on the realized analysis, it can be said that the required minimum drivage rate using the machine is 6.85 m/day.

REFERENCES

[1] Vidanović N, Đukanović D., Dragosavljević Z.; Inovation of technology of construction of underground mininig workings by use drilling and blasting methods of work, Technics Technologies Education Management, TTEM, Vol. 5, No.4, pp. 861–866; 2010.

[2] Đukanović D., Đukanović D. Savić Lj.; Analiza ostvarenih tehničkih parametara pri izradi podzemnih prostorija u rudnicima uglja u Srbiji, Podzemni radovi 13, str. 17-22, RGF, Beograd, 2004.

[3] Đukanović D.; Model optimizacije tehno-ekonomskih pokazatelja pri izradi podzemnih prostorija u rudnicima uglja Srbije, Doktorska disertacija, RGF, Beograd, 2005.

[4] Đukanović D.; Tehnologija izrade jamskih prostorija kombinovanim mašinama sa osvrtom na mogućnost primene u rudnicima uglja Srbije. Savez energetičara, Beograd, 143 strane; 2005.

[5] Đukanović D., Đukanović D.; Analiza zavisnosti ostvarenih troškova i brzine izrade podzemnih prostorija u rudnicima uglja u Srbiji, Rudarski radovi 01/2005, str. 51-55, Komitet za podzemnu eksploataciju mineralnih sirovina, Bor, 2005.

[6] Đukanović D.; Istraživanje strukture troškova pri izradi podzemnih prostorija u rudnicima uglja u Srbiji, (Investigation of the costs structure in development the underground rooms in the coal mines in Serbia), Rudarski radovi 02/2005, Komitet za podzemnu eksploataciju mineralnih sirovina, Bor, str. 81-88. 2005.

[7] Đukanović D.; Possibilities for application of contemporary technologies production of shafts premises in coal mines in Serbia, 39th International October conference on mining and metallurgy, 07 - 10. October, Sokobanja, University of Belgrade, Tehnical Faculty Bor and Copper institute Bor, Serbia, str. 39-43; 2007.

[8] Đukanović D., Denić M., Dragojević D.; Modernizacion of tehnological process of the construction of shaft premises in coal mines of Serbia, Internacional Mining Forum 2009 – Deep Mining Challenges – I Polish-Serbian forum, 18-21. February, Krakow, str. 69-72; 2009.

[9] Đukanović D.; Studija opravdanosti mehanizovane izrade rudarskih prostorija u rudnicima JP PEU; Biro za projektovanje i razvoj, Beograd, 75 strana, 2010.

[10] Milisavljević V., Đukanović D.; Present situation and perspective of roadways development in underground coal mines in Serbia, Technicka Diagno-stika, str. 253-257, TU, Ostrava, 2005.

[11] Tokalić R., Trajković S., Đukanović D.; Analiza postignutih učinaka na izradi podzemnih prostorija u rudnicima uglja u Srbiji, Podzemni radovi 11, str. 33-38, RGF, Beograd, 2002.

[12] Trajković S., Đukanović D., Lutovac S.; analysis of technical parameters impact at advance rate of roadway development in Serbian coal mines, 2nd Balkan Mining Congress, (BALKANMINE 2007), 10. do 13. 09., Beograd, Academy of Engineering Sciences of Serbia and Faculty of Mining and Geology University Belgrade, 143-148; 2007.

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:551.49:622.79:504.06(045)=861

Slađana Krstić, Goran Marinković**, Vesna Ljubojev**

TUNEL ZA IZMEŠTANJE KRIVELJSKE REKE – TRAJNO REŠENJE RIZIKA MOGUĆIH KRITIČNIH ASPEKATA***

Izvod

Jalovišta su jedan od glavnih problema stanja životne sredine u okviru RTB kompleksa Bor. Kao prioriteti po urgentnosti za rešavanje problema stanja životne sredine, u vrhu se nalazi izrada tunela za izmeštanje kriveljske reke, čime se trajno rešava rizik od mogućih kritičnih aspekata kolektora koji se nalazi ispod jalovišta Velikog Krivelja. Ovim radom sagledane činjenice ukazuju potrebu pri izradi tunela za izmeštanje kriveljske reke čime se eliminiše rizik ekološke katastrofe. Prikazane su geotehničke, petrološke i hidrogeološke karakteristike stena rudnika Veliki Krivelj. Tunelom se trajno rešava jedan od rizika životne sredine reke Timok i šire, jer Timok pripada slivnom području Dunava i Crnog mora.

Ključne reči: *Jalovišta, hidrogeološke karakteristike stena, problema stanja životne sredine, tunel.*

UVOD

Borski rudarski kompleks (RTB) se nalazi uglavnom na pretežno brežuljkastom i brdovitom području sa nadmorskom visinom od 400-600 m. Bor je smešten u dolini istoimene reke na nadmorskoj visini od 360 m.

Severozapadno od Bora u brdovitom predelu formirano je razvođe Kriveljskog potoka. Reke Cerovo na istoku i Valja Mare na jugozapadu, spajanjem, na udaljenosti od oko 2 km jugoistočno od rudnika Cerovo i isto toliko od sela Mali Krivelj daju Kriveljsku reku, koja teče u svom prirodnom basenu do otvorene jame Veliki Krivelj.

Zbog rudarskih radova koji su se odvijali tokom poslednjeg stoleća, morfologija se znatno izmenila u odnosu na prvobitno stanje. Područje Bora ima usmeren pravac podzemnih vodotokova na reku Timok (slika 1), koja pripada slivnom području Dunava i Crnog mora.

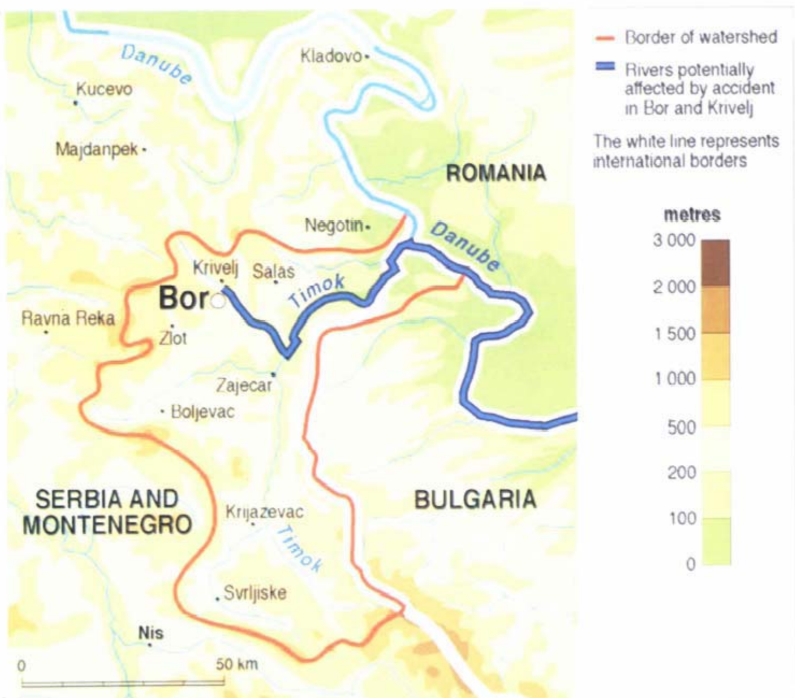
Hidrološka situacija u slivnom području reke Timok je složena zbog mnogih mesta

* *Institut za rudarstvo i metalurgiju Bor*
** *Geološki institut Srbije, Beograd*
*** *Ovaj rad je proistekao iz Projekta 33021, koji je finansiran srestvima Ministarstva prosvete i nauke Republike Srbije.*

gde se izliva otpadna voda iz RTB kompleksa zajedno sa sanitarnom otpadnom vodom iz grada Bora i više sela. Ceo RTB kompleks utiče na vodotokove, jer osim taloženja rastvorenih čvrstih materija kod rudnika Cerovo, u metalurškom kompleksu se ne obavlja prečišćavanje otpadnih voda. Kriveljski potok južno od rudnika i jalovišta Veliki Krivelj je kiselast i sadrži povišeni nivo rastvorenih čvrstih materija, gvožđa, bakra i cinka.

Borska reka i Kriveljska reka su krajnje destinacije otoka i otpadnih voda od procesa flotacije koji se obavlja u Velikom Krivelju i voda iz topionice i rafinerije i neprečišćenih gradskih otpadnih voda. Zbog toga je krajnje zagadjena i degradirana površinska voda (pH, rastvorene čvrste materije, bakar i gvožđe).

Rudarska aktivnost je proteklih godina veoma uticale na prirodni tok Borske reke, koja je prvobitno tekla sa severozapada na jugoistok i dalje do Bora, i koja je skrenuta cevovodom sagrađenim severno od borske otvorene jame, sada utiče u devijaciju Kriveljske reke ispred podzemnog kolektora koji je postavljen ispod jalovišta Veliki Krivelj. Južno od jalovišta “RTH”, prirodni rečni basen ne prima nikakvu rečnu vodu, ali voda odvodi taj tok u Kriveljsku reku jugoistočno od mesta Slatina. Tako je tok Kriveljske reke izmenjen u odnosu na prvobitni tok i to kod otvorene jame Velikog Krivelja gde sada oivičava jamu, i kod jalovišta Veliki Krivelj gde je skrenut u u podzemni kolektor koji prolazi ispod istočnog jalovišta.



Sl. 1. *Basen reke Timok*

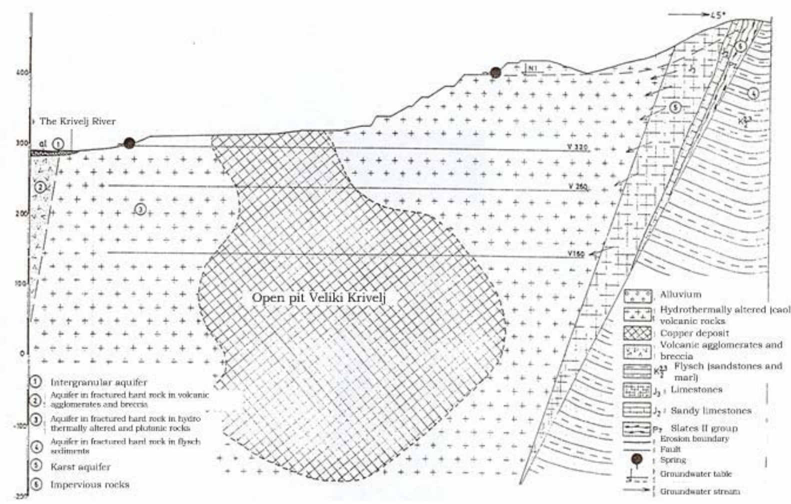
Zagađenost Borske reke se jasno vidi između Bora i Slatine i rečne obale imaju naslage jalovine od prethodnih incidenata na borskom jalovištu. Borska voda je još uvek kisela i sadrži povišeni nivo rastvorenih čvrstih materija i koncentracije bakra i na udaljenosti od 10 km od metalurškog kompleksa.

Kvalitet zemljišta je ispitivan u Srbiji i smatra se da je rudarenjem u Boru (metalična i nemetalična eksploatacija) uništeno zemljište površine 1.110 ha, što je glavno učešće u toj raspodeli na nivou Srbije. Procenjena na oko 60,6% od ukupnog poljoprivrednog zemljišta. Kao glavni uzroci uništavanja zemljišta su rudarstvo i metalurgija, rudni kopovi, deponije za odlaganje jalovine i flotacijske jalovine.

Eksploatacija (metala i nemetala) je degradirala poljoprivrednu, obradivu zemlju u Boru, Slatini, Oštrelju, Krivelju, Bučju i Donjoj beloj reci. Ispuštanje otpadnih voda iz postrojenja za flotaciju i jalovišta su degradirali zemlju u industrijskoj zoni katastarskih opština Slatina, Rgotina, Vražogrnci i mnogim selima u dolini reke Veliki Timok.

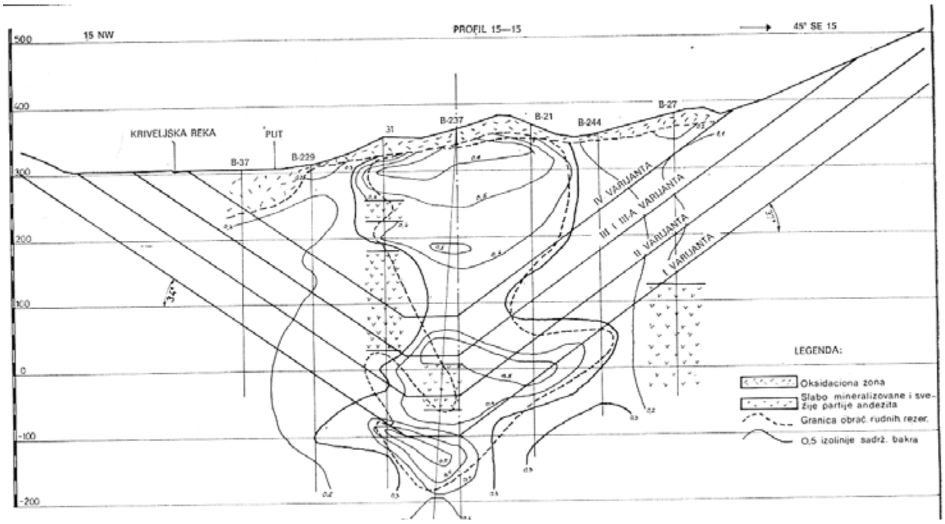
HIDROGEOLOGIJA KRIVELJSKE REKE

Vodopropustljivost područja Velikog Krivelja je uslovljena karakteristikama hornblenda andezitskih aglomerata i konglomerata koji imaju malu propustljivost. Hidrogeološke karakteristike područja ležišta Veliki Krivelj (V. Dragišić) detaljno su istraživane (slika 2). Na slici je prikazan izvor vode što ukazuje na prvobitni nivo vode i priliv podzemnih voda iz karstnog vodonosnog sloja do sloja koji je prisutan u naprslim stenama koje sadrže naslage bakra. Ova cirkulacija podzemne vode se povećavala formiranjem otvorene jame koja je eliminisala škriljce (desni gornji deo slike) koji su delili vulkanske i flišne sedimentne vodonosne slojeve. Podzemna voda se sada gravitacijom vodi od peščara flišnog sedimenta (vodonosni sloj 4) do naprsle tvrde stene (vodonosni sloj 3).



Sl. 2. *Poprečni presek hidrogeološkog profila ležišta Veliki Krivelj i prvobitnu cirkulaciju podzemne vode u vulkanskim i sedimentnim stenama (Izvor: Ibid)*

Geološki poprečni presek (slika 3) od nadmorske visine 350 do -200 m prikazuje položaj istražnih bušotina i konture ležišta (isprekidana linija) pri obračunu rezervi rude (procenjenih na 440 Mt u konturnoj liniji sa 0.43 % bakra u rudi). Kose linije (I do IV varijanta) predstavljaju četiri različite kosine i dna otvorenog kopa u raznim fazama rudarskih radova. Na ovom poprečnom preseku uočava se da četvrtom varijantom napredovanja rudarskih radova, Kriveljska reke teče duž ivice kopa. U oblasti kopa Velikog Krivelja, rudarskim aktivnostima dolazi do izbijanja na površinu peščara (čime se dobija mala propustljivost) i hidrotermalno izmenjenih vulkanskih stena koje su nepropustljive i u kojima je smešten ispucali vodonosni sloj. Sa raspoloživim informacijama ne može se proceniti opasnost od ovog područja.



Sl. 3. *Geološki poprečni presek i IV projektovane varijante dna otvorenog kopa Veliki Krivelj*

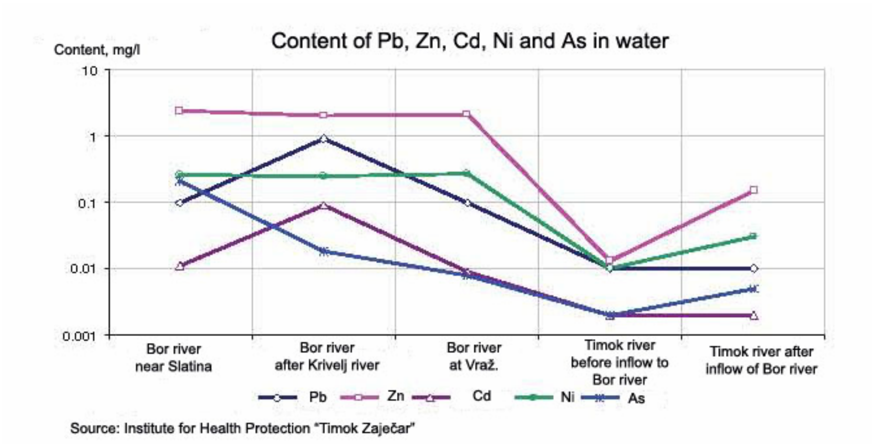
ZAGAĐENJE ZEMLJIŠTA I VODA

Ekološko stanje u opštini Bor prema podacima iz 2005. godine za sadržaj bakra je iznad dozvoljenih granica Republike Srbije u Oštrelju, Slatini i Bučju (odnosno 125 mg/kg, 135 mg/kg i 120 mg/kg). Sadržaj arsena, blizu maksimalne dozvoljene koncentracije od 25 mg/kg je pronađen u Krivelju, Slatini i Metovnici. Kiselost zemljišta se javlja kao zajednički problem na celokupnoj ispitanoj površini. Vrednost pH < 5 izmerena je u Boru i Brestovcu, dok je na drugim lokacijama pH vrednost ispod 6.

Borska i Kriveljska reka predstavljaju otvoreni kolektor za otpadnu vodu, kompletno su degradirane i nemogu se klasifikovati prema zakonskim propisima. Posle uliva Borske reke u Kriveljsku reku nastaje Bela reka koja se uliva u Veliki Timok (slika 1). Ispitivanje rečnog sedimenta je radjena u okviru projekta UNEP-a. Sedimenti u Borskoj reci pre njenog spajanja sa Kriveljskom rekom (uzorak ID 10-33); Sedimenti u Kriveljskoj reci, na mestu spajanja sa Borskom rekom (uzorak ID 10-34) i sedimenti u borskoj reci posle spajanja sa Kriveljskom rekom (uzorak ID 10-33).

Reke koje se nalaze nizvodno od RTB Bor i ulivaju se u Borsku reku su zagadjene i njihov dotok utiče na kvalitet Dunava. U njihovim plavnim zonama se talože sedimenti jalovine od flotacije. To predstavlja međudržavni problem zagađenja životne sredine.

U drugoj polovini dvadesetog veka, jalovina od flotacije se izlivala u Borsku reku i tako oštetila najmanje 2.500 ha plavne zone Borske reke i Velikog Timoka (slika 4).



Sl. 4. *Sadržaj metala iz reka (Izvor: LEAP)*

Na svim lokacijama gde su se uzimali uzorci nađena je velika količina rastvorenih materija. U Borskoj reci, uzvodno i nizvodno od uliva u Kriveljsku reku i pre spajanja sa Timokom, u Kriveljskoj reci, uzvodno od uliva Borske reke i u Timoku, nizvodno od uliva Borske reke koncentracije gvožđa i bakra su visoke u poredjenju sa limitima klase III.

Borska, Kriveljska i Bela reka imaju velike koncentracije nikla na svim lokacijama gde su uzimani uzorci. Cink je prisutan u velikim koncentracijama i u Borskoj i Beloj reci.

Kriveljska i Bela reka se karakterišu veoma malim pH vrednostima (<5). Lokacija uzimanja uzorka u Beloj reci pokazuje visoke koncentracije olova i kadmijuma.

Isto tako je očigledno da se ulivanjem Borske reke u Kriveljsku reku smanjuju pH vrednosti a povećava BPK5, HPK, rastvorene materije, gvožđe, amonijak, TOC ukupni ugljovodonici, bakar, cink, nikl i mineralna ulja.

Ulivanje Bele reke u Timok smanjuje pH vrednosti i povećava BPK5, HPK, rastvorene materije, gvoždje, ukupne ugljovodonike, bakar, cink, nikl i arsen.

ZAKLJUČAK

Rezultati svih ispitivanja su jasni, vidi se uticaj Bele reke na kvalitet vode Timoka, odnosno, kvalitet vode u Timoku se naglo smanjuje posle ulivanja Bele reke (slika 4). Borska reka je 2002. godine od svog izvora do naselja Bor klasifikovana kao vodotok II kategorije. Nizvodno od naselja Bor do spajanja sa Kriveljskom rekom – kao klasa IV. Kriveljska reka je van kategorija, dok Bela reka ima klasu IV.

Timok je od naselja Zaječar do spajanja sa Belom rekom kategorisan klasom IIb. Odatle, pa do spajanja sa Dunavom, njegov tok je klasifikovan kao III kategorija.

U okviru rudarskog kompleksa RTB postoje tri jalovišta za odlaganje jalovine, sa ukupnom rekultivisanom površinom oko 30 ha. Jalovište Veliki Krivelj (Polje 1 i Polje 2), sa tri nasipa (nasip 1A, 2A i 3A) je najkritičnije zbog stanja kolektora i mogućeg ekološkog incidenta.

LITERATURA

[1] Analiza koncentracije metala u Borskoj reci, Zavod za zaštitu zdravlja Timok, Zaječar, 2005.

[2] Izveštaj o ekološkom stanju u opštini Bor za period I-XII 2004 do I-VI 2005, Odsek za društvene i industrijske aktivnosti, Opština Bor, 2005.

[3] Plan monitoringa životne sredine (EMP) za RTB Bor kompleks, tokom i posle privatizacionog procesa. Izvor: Opština Bor, 2005.

[4] Rezultati projekta “Zemljište i podzemne vode” teški metali (Centar za poljoprivredna istraživanja, 1997)

[5] Rezultati projekta “Zemljište i podzemne vode” rečni nanosi, UNEP, 2002.

[6] Assessment of Existing Environmental Monitoring Capacities in Bor, UNEP, 2002.

[7] Rezultati projekta “Plan praćenja stanja životne sredine“ Izvor: Opština Bor, 2005.

[8] S. Krstic, M. Ljubojev, V. Ljubojev, M. Bugarin, Developloment of a new tunnel under the flotation barren Veliki Krivelj, Proceedings of the 10-th International Multidisciplinary Scientific Geo-Conference (SGEM 2010), 21-26 june 2010, Albiena Bulgaria, pp. 227-231.

[9] S. Krstić, M. Ljubojev, V. Ljubojev, Petrological Characteristics Rock on Tunnel for Relocation Krivelj`s River, Proceedings of the 15 th Congress of geologists of Serbia with International participation (Extended abstract) pp.40., Belgrade, 26-29. 05. 2010 .ISBN 978-86-86053-08-4

[10] S. Krstić, M. Ljubojev, M. Bugarin, Litological types of rocks along the new tunnel route for the relocation of the Kriveljska River, 4th Croatian Geological Congress with international participation, Šibenik 14-15.10.2010, Abstracts Book, pp 167.

[11] S. Krstic, M. Ljubojev, V. Ljubojev, M. Bugarin, Massif rock and effect glivage on stability tunnel of Krivelj river inBor, Proceedings of the 9-th International Multidisciplinary Scientific Geo-Conference (SGEM 2009), 14-19.06 2009, Albiena Bulgaria, pp 65-71.

[12] M. Ljubojev, D. Ignjatović, V. Ljubojev, L. Đ. Ignjatović, S. Krstić, Determination of rock quality index during tunneling, Proceedings of the 42nd International October Conference on Mining and Metallurgy, 10.-13.10.2010, pp. 138-142.

[13] M. Ljubojev, D. Ignjatović, V. Ljubojev, L. Đ. Ignjatović, S. Krstić, Determination of rock quality index during tunneling, Proceedings of the 42nd International October Conference on Mining and Metallurgy, 10.-13.10. 2010, pp. 138-142

MINING AND METALLURGY INSTITUTE BOR YU ISSN: 1451-0162
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS UDK: 622

UDK: 551.49:622.79:504.06(045)=20

Sladjana Krstić, Goran Marinković**, Vesna Ljubojev**

TUNNEL FOR RELOCATION THE RIVER KRIVELJ PERMANENT RISK SOLUTION OF POSIBLE CRITICAL ASPECTS***

Abstract

Tailing dumps are one of the major environmental problems within the complex of RTB Bor. As the urgent priority of solving the environmental problems, a construction of tunnel for relocation the river Krivelj is at the top, what is a permanent solution of possible critical aspects of collector which is located below the tailing dump Veliki Krivelj. This paper analyzed the facts indicated a need for construction the tunnel for relocation the Krivelj river eliminating the risk of ecological disaster. Geotechnical, hydrogeological and petrological characteristics of rocks of the Veliki Krivel mine are reviewed in this paper. The tunnel is a permanent solution to environmental risk of Timok and larger, because it belongs to the catchment area of Danube and Black Sea.

Key words: *tailing dump, hydrogeological characteristics of rocks, environmental problems, tunnel*

1. INTRODUCTION

The Bor mining complex (RTB) is located mainly in the predominantly hilly and mountainous area at altitude of 400-600 m. Bor is situated in a valley of the same named river at altitude of 360 m.

Northwest of Bora, in the hilly area, a watershed of the Krivelj stream is formed. The rivers Cerovo, in the east, and Valja Mare, in the southwest, connecting at a distance of about 2 km southeast of the mine Cerovo and the same distance from the village Mali Krivelj, giving the Krivelj river, which flows in its natural basin to the open pit Veliki Krivelj.

Due to the mining activities, developed during the last century, the morphology is significantly changed compared to the original state. The area of Bor has focused direction of groundwater to Timok (Figure 1), which belongs to the catchment area of Danube and Black Sea.

Hydrological situation in the catchment area of Timok is complex due to many places where wastewater discharge

* *Mining and Metallurgy Institute Bor*
** *Geology Institute of Serbia, Belgrade*
*** *This paper is derived from the Project 33021, funded by the Ministry of Education and Science of the Republic Serbia.*

from RTB complex along with sanitary waste water from the town of Bor and many villages. The whole RTB complex affects the watercourses, because apart from deposition the dissolved solids in the mine Cerovo, there is no wastewater treatment in the metallurgical complex. The Krivelj stream, in the south of mine and tailing dump Veliki Krivelj is acidic and contains increased level of dissolved solids, iron, copper and zinc.

The Bor river and Krivelj rivers are the final destinations of waste water discharge from the flotation process, which takes place in Veliki Krivelj, and water from the Smelter and Refineries and untreated municipal wastewater. Therefore, the surface water is highly polluted and degraded (pH, dissolved solids, copper and iron).

Mining activity in recent years, the most affected the natural flow of the Bor river, which originally flowed from the northwest to the southeast and away to Bor, and that was diverted by the pipeline, built in the north of the Bor Open Pit, and now inflows a deviation of the Krivelj river in front of the collector, installed below the tailing dump Veliki Krivelj. In the south of the tailings dump "RTH", a natural river basin does not receive any river water, but water drains that flow into the Krivelj river in the southeast of the village Slatina. Thus, the flow of the Krivelj river was changed from the original course and to the Open Pit of Veliki Krivelj which now line the pit, and at the tailing dump Veliki Krivelj, it was diverted into underground collector that runs below the east tailing dump.

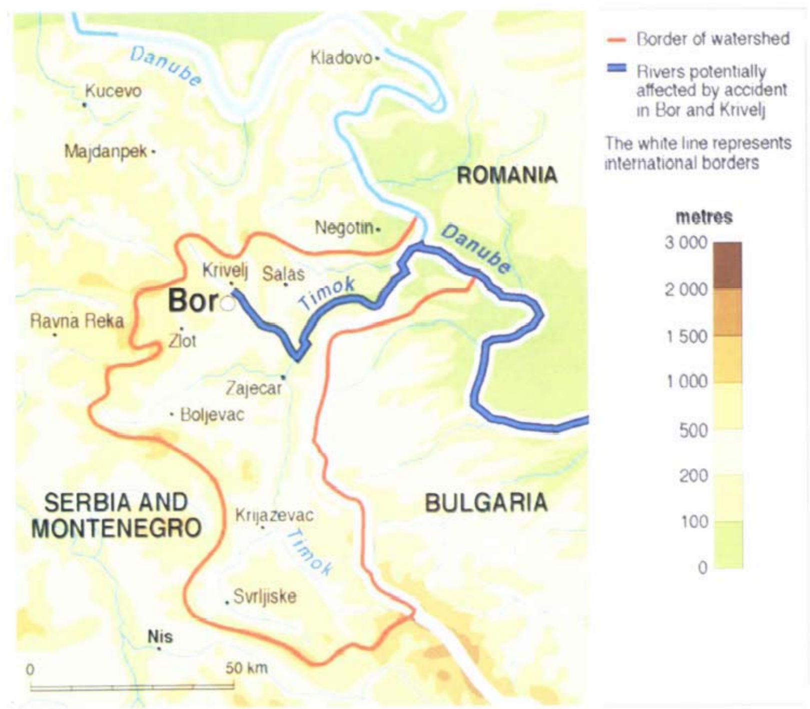


Figure 1. *The river basin of Timok*

Pollution of the Bor river is clearly visible between Bor and Slatina and the river banks have deposits of tailings from the previous incidents at the Bor tailing dump. The Bor water is still acidic and contains the increased level of dissolved solids and copper concentrations copper and at distance of 10 km from the metallurgical complex.

Soil quality was investigated in Serbia and it is considered that mining in Bor (metallic, nonmetallic exploitation) destroyed the land area of 1,110 ha, what is the major part in this distribution at the level of Serbia. It was estimated as about 60.6% of total agricultural land. The main causes of land destruction are mining and metallurgy, open pits, landfills for disposal of waste and flotation tailings.

Exploitation (metal and nonmetal) has degraded the agricultural cultivable land in Bor, Slatina, Oštrelj, Krivelj, Bučje and Donja Bela Reka. Discharge of waste water from the flotation plants and tailing dumps have degraded the land in the industrial area of the cadastral municipalities Slatina, Rgotina, Vražogrnac and many villages in the valley of the Veliki Timok.

2. HYDROGEOLOGY OF THE KRIVELJ RIVER

Water permeability in the area of Veliki Krivelj is conditioned by the characteristics of hornblende andesite agglomerates and conglomerates that have little permeability. Hydrogeological characteristics of the area of Mali Krivelj deposit (V. Dragišić, 1987) were investigated in detail (Figure 2). Figure 2 shows the water source what indicates the initial water level and inflow of groundwater from the karst aquifer to the present layer that in the cracked rocks that contain copper deposits. This circulation of groundwater was increased forming an open pit that eliminated shale (right upper part of Figue 2) sharing the volcanic and flysch sedimentary aquifers. Underground water is now taken by the gravity to the sandstone of flysh sediment (aquifer 4) to the cracked solid rock (aquifer 3).

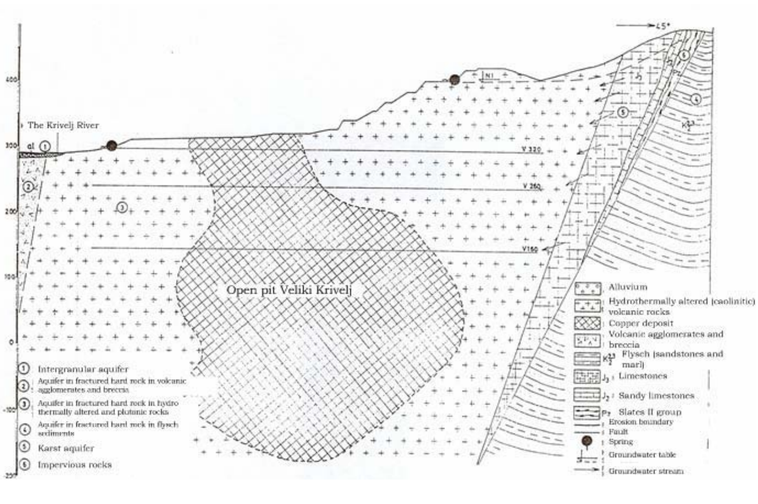


Figure 2. *Hydrogeological cross-section profile of the Veliki Krivelj deposit and initial circulation of groundwater in volcanic and sedimentary rocks (Source: Ibid)*

Geological cross section (Figure 3) at the altitude from 350 to 200 m shows the location of exploration drill holes and deposit contours (dashed line) in the calculation of ore reserves (estimated at 440 Mt in the contour line with 0.43% copper in ore). The oblique lines (I-IV version) represent four different slopes and bottom of the open pit at various stages of mining operations. It is observed in this cross section that the Krivelj river flows along the edge of open pit by the fourth version of progress the mining operations. In the area of the Veliki Krivelj mine,

The sandstone breaks out to the surface by the mining activities (resulting in a low permeability) and hydrothermally altered volcanic rocks that are impermeable and where the cracked aquifer is located. The risk of this area cannot be evaluated by the available information.

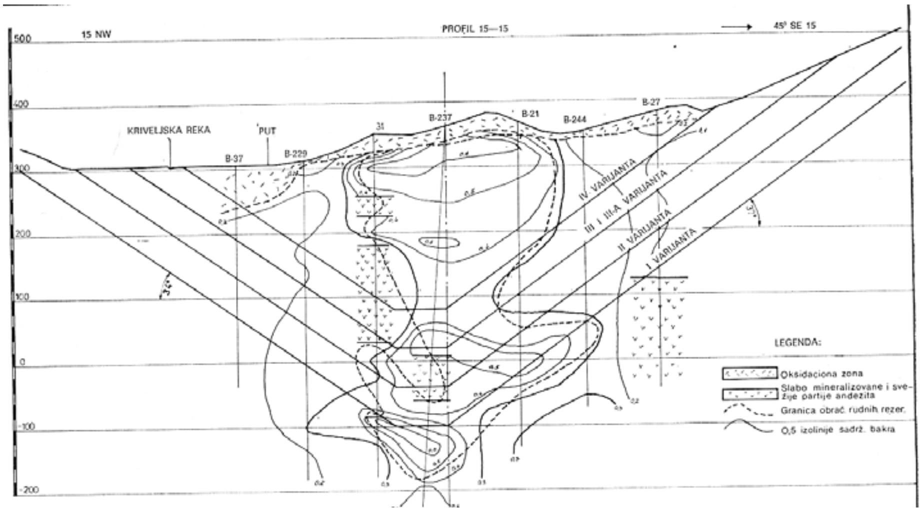


Figure 3. *Geological cross-section and IV designed versions of the Veliki Krivelj open pit bottom*

3. POLLUTION OF SOIL AND WATER

The ecological situation for copper content in the Bor municipality, according to the data from 2005, is over allowable limits of the Republic of Serbia in Oštrelj, Slatina and Bučje (i.e.125 mg/kg, 135 mg kg and 120 mg/kg). The content of arsenic, near maximum allowable concentration of 25 mg/kg was found in Krivelj, Slatina and Metovnica. Soil acidity occurs as a common problem on the entire investigated surface. pH <5 was measured in Bor and Brestovac, while in other locations pH value was below 6.

The Bor and Krivelj rivers, presenting an open collector for waste water, are completely degraded and can be classified according to legal regulations. After the inflow of the Bor river into the Krivelj river, the Bela River is formed that flows into the Veliki Timok (Figure 1). Investigation of the river sediments was done within the UNEP project. The sediments in the Bor river before its connection with the Krivelj river (sample ID 10-33); the sediments in the Krivelj river, at the connection point with the Bor river (sample ID 10-34) and sediments in the Bor river after its connection with the Krivelj river (sample ID 10-33).

The rivers, located downstream of RTB Bor and flowed into the Bor river, are polluted and their flow affects the quality of Danube. In their flood zones, the sediments from flotation tailings are deposited. This is an international problem of environmental pollution.

In the second half of the twentieth century, the tailings from the flotation was discharged into the Bor river and also damaged at least 2,500 ha of flood zone of the Bor river and Veliki Timok (Figure 4).

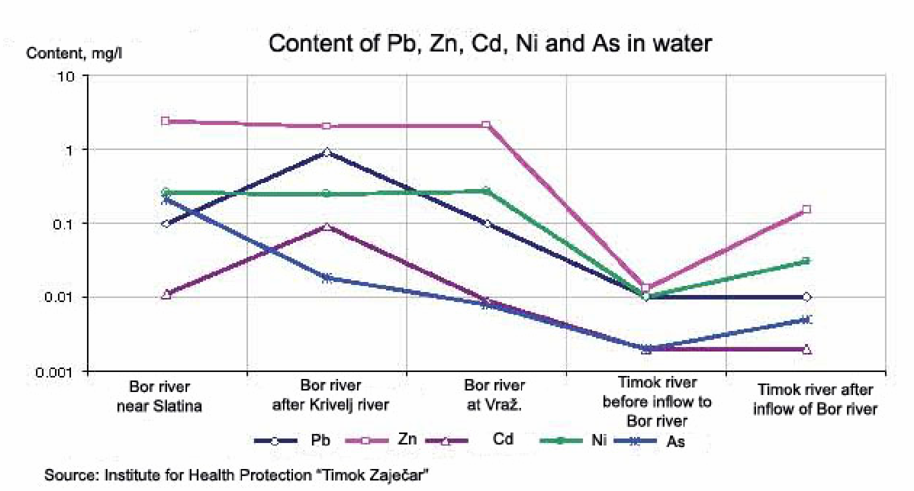


Figure 4. Metal contents in the rivers (Source: LEAP)

At all locations, where the samples were taken, a large quantity of dissolved solids was found. In the Bor river, upstream and downstream of the river flows into the Krivelj river and before connecting with Timok, in the Krivelj river, upstream from the inflow of the Bor river in Timok, downstream from the inflow of the Bor river, copper and iron concentrations are high compared with the limits of the III class.

The Bor, Krivelj and Bela River have high concentrations of nickel at all locations where samples were taken. Zinc is present in high concentrations in the Bor and the Bela River.

The Krivelj and Bela River are characterized by very low pH values (<5). Sampling location in the Bela River shows high concentrations of lead and cadmium.

Also, it is obvious that the inflow of the Bor river into the Krivelj river reduces pH values and increases BPK5, HPK, dissolved matters, iron, ammonia, TOC total hydrocarbons, copper, zinc, nickel and mineral oils.

Inflow of the Bela River into Timok decreases the pH values and increases BPK5, HPK, dissolved matters, iron, total hydrocarbons, copper, zinc, nickel and arsenic.

5. CONCLUSION

The results of all tests are clear; the effect of the Bela River to the water quality of Timok, i.e., the water quality in Timok rapidly decreases after inflow of the Bela River (Figure 4). The Bor river in 2002, from its source to the Bor settlement was classified as the II category watercourse; downstream from the Bor settlement to the connection with the Krivelj river as the IV class. The Krivelj river is outside the category, while the Bela River isthe IV class.

Timok is, from the Zaječar settlement to the connection with the Bela River, categorized as the IIb class. By there, to the connection with Danube, its course is classified as the III category.

Within the mining complex RTB, there are three tailing dump for disposal of tailings, with the total re-cultivated area about 30 ha. The tailing dump Veliki Krivelj (Field 1 and Field 2), with three dams (dams 1A, 2A and 3A) is the most critical due the collector state and possible environmental incident.

REFERENCES

[1] Analysis of Metal Concentrations in the Bor River, Institution for Health Protection Timok, Zaječar (in Serbian)

[2] Report on Environmental Condition in the Bor Municipality for the Period I-XII 2004 to I-VI 2005, 2005, Department of Social and Industrial Activities, Municipality of Bor (in Serbian)

[3] Environmental Monitoring Plan (EMP) for RTB Bor Complex During and After the Privatization Process, 2005, Source: Municipality of Bor (in Serbian)

[4] Project Results “Soil and Groundwater” Heavy Metals (Center for Agricultural Research, 1997), (in Serbian)

[5] Project Results “Soil and Groundwater” River Sediments, 2002, UNEP (in Serbian)

[6] Assessment of Existing Environmental Monitoring Capacities in Bor, September 2002, UNEP

[7] Project Results “Plan for Environmental Monitoring “ 2005, Source: Municipality of Bor (in Serbian)

[8] S. Krstić, M. Ljubojev, V. Ljubojev, M. Bugarin, Development of a New Tunnel under the Flotation Tailing Dump Veliki Krivelj, Proceedings, 10th International Multidisciplinary Scientific Geo-Conference (SGEM 2010), 21-26 June 2010, Albiena Bulgaria, pp. 227-231

[9] S. Krstić, M. Ljubojev, V. Ljubojev, Petrological Characteristics of Rocks on Tunnel for Relocation the Krivelj River, Proceedings , 15th Congress of Geologists of Serbia with International Participation (Extended Abstract) pp.40, Belgrade, 26-29 May 2010, ISBN 978-86-86053-08-4

[10] Sladjana Krstić, Milenko Ljubojev, M. Bugarin, Lithological Types of Rocks along the New Tunnel Route for Relocation of the Krivelj River, 4th Croatian Geological Congress with International Participation, Šibenik 14-15 October 2010, Abstracts Book, pp 167

[11] S. Krstić, M. Ljubojev, V. Ljubojev, M. Bugarin, Massif Rock and Effect of Clivage on the Tunnel Stability of the Krivelj River in Bor, Proceedings, 9th International Multidisciplinary Scientific Geo-Conference (SGEM2009), 14-19 June 2009, Albiena Bulgaria, pp 65-71

[12] M. Ljubojev, D. Ignjatović, V. Ljubojev, L. Djurdjevac - Ignjatović, S. Krstić, Determination of Rock Quality Index during Tunneling, Proceedings, 42nd International October Conference on Mining and Metallurgy, 10-13 October 2010, pp. 138-142

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:553.94:622.333:622.79(045)=861

Duško Đukanović, Milan Popović**, Dragomir Zečević***

EKONOMSKI EFEKTI UPOTREBE JALOVINE IZ SEPARACIJE UGLJA IBARSKIH RUDNIKA

Izvod

U Ibarskim rudnima kamenog uglja Baljevac, do 2005. godine čišćenje rovnog uglja obavljano je u teško tekućinskoj separaciji na principu „pliva–tone". Jalovina iz separacije je odlagana na lokaciji u neposrednoj blizini separacije. Tokom oksidacionih procesa separacijska jalovina se promenila, rezultat promene je novi materijal – rudnička šljaka, crvenkaste boje. Korišćenje rudničke šljake, nastale oksidacionim procesima za putnu privredu i proizvodnju građevinskih elemenata je način da se uz ostvarivanje proizvodnje ukloni jalovina i izvrši sanacija lokacije odlagališta jalovine i ostvare određeni ekonomski efekti.

Ključne reči: *separacija, separacijska jalovina, rudnička šljaka, građevinski elementi, troškovi proizvodnje.*

1. UVOD

Ibarski rudnici kamenog uglja bave se eksploatacijom uglja više osamdeset godina u ležištima Jarando, Ušće, Tadenje-Progorelica. Eksploatacija uglja u ležištu Ušće je zbog iscrpljenosti rezervi završena. Otkopani rovni ugalj u ležištima Jarando i Tadenje-Progorelica se oplemenjuje u jedinstvenoj separaciji u Piskanji, do koje se doprema vazdušnom žičarom.

U periodu od 1960. godine do 1963. godine izgrađena separacija uglja u Piskanji-Baljevac.

Prilikom izgradnje separacije izgrađeno je jalovište. Separaciska jalovina se sastoji od sitnih i krupnih klasa stena koje se javljaju u produktivnoj seriji ležišta Jarando i Tadenje – Progorelica i sraslace koji imaju sagorljivih materija. Na separacijskom jalovištu javljaju se oksidacioni procesi, tokom oksidacionih procesa jalovina se promeni, rezultat promene je novi materijal – rudnička šljaka, crvenkaste boje.

Godine 1974. izgrađena je fabrika za proizvodnju građevinskih elemenata od rudničke šljake, koja je radila do 2002. godine. Od 2005. godine se ne vrši čišćenje uglja po principu "pliva – tone", niti se odlaže jalovina na pomenutoj lokaciki. Planirano je da se izvrši sanacija postojećeg jalovišta, što se može učiniti ponovnim pokretanjem proizvodnje građevinskih elemanata i prodajom šljake putnoj privredi.

2. OSOBINE RUDNIČKE ŠLJAKE

Jalovine iz prališta uglja mogu da predstavljaju vrlo pogodne agregate za proizvodnju šupljih betonskih blokova pod uslovom da se radi o prirodno paljenim

* *JP za PEU, Biro za projektovanje i razvoj Beograd*
** *JP PEU Resavica, Ibarski rudnici kamenog uglja – Baljevac*

glinama čiji hemijski sastav i zapreminske težine odgovaraju uslovima postavljenim odnosnim standardima, odnosno propisima. Do procesa "samopaljenja" ovih otpadnih materijala dolazi zbog onečišćenja osnovne glinene mase znatnim količinama uglja koji nije uklonjen pri procesu separisanja, odnosno pranja uglja. Inače, do samog procesa sagorevanja primesa uglja, odnosno pečenja cele glinene mase dolazi zbog procesa spontanog sagorevanja sumpornih jedinjenja prisutnih u glinenoj masi. Ovaj, prirodnim procesom dobijeni agregat, rudnička šljaka, ima izgled i boju pečene gline, odlikuje se relativno visokim zapreminskim težinama (800-1000 t/m^3) i često puta vrlo visokim sadržajem sulfata. Zbog ovako visokog sadržaja rastvorljivih sulfata došlo je i do osporavanja mogućnosti iskorišćenja ovakvih šljaka kao agregata za proizvodnju betona, odnosno betonskih prefabrikata. Međutim, izvesna fizička ispitivanja nekoliko domaćih šljaka su pokazala da su ove šljake i pored visokog sadržaja sulfata bile "stalne zapremine", a što je bilo dokazano testom-kuvanjem kolačića na kojima nisu registrovanje pojave radijalnih pukotina, vitoperenja i svih ostalih pojava koje diskredituju ovu rudničku šljaku u pogledu stalnosti zapremine.

Hemijska analiza rudničke šljake sa deponije u Piskanji	
- Vlaga na 110^{0}C	0,34 %
- Gubitak žarenjem	1,35 %
- SiO_2	49,00 %
- Al_2O_3	11,69 %
- TiO_2	0,37 %
- Fe_2O_3	14,34 %
- MnO	0,10 %
- CaO	9,33 %
- MgO	2,22 %
- Na_2O	0,66 %
- K_2O	2,17 %
- SO_3	8,19 %
U K U P N O :	**99,76 %**

Ispitivanja hemijskog sastava rudničke šljake nastale spontanim sagorevanjem jalovine sa odlagališta uglja u Piskanji pokazala su da ispitivana šljaka predstavlja alumo-silikatnu materiju čiju osnovnu komponentu predstavlja nerastvorni SiO_2 pri čemu se šljaka odlikuje visokim sadržajem sulfata (oko 8 %). Ispitivanjem prirode sulfata sadržanih u ovoj šljaki utvrdjeno je da sadržaj sulfata rastvornih u vodi iznosi 2,97 % pri čemu je difraktometrijskom analizom utvrđeno da se sulfati nalaze uglavnom u obliku minerala anhidrita.

3. REZERVE RUDNIČKE ŠLJAKE

Ukupne rezerve rudničke šljake, koja se može koristiti za proizvodnju građevinskih elemenata, agregata za putnu privredu ili druge namene iznose oko 320.000 t.

4. LOKACIJA ODLAGALIŠTA SEPARACIJSKE JALOVINE

Jalovište separacije locirano je sa desne strane reke Ibar. Teren na kome je locirano jalovište je takoreći ravno, visinske razlike su male. Širi teren je nagnut prema reci Ibar koja drenira čitav teren, pa na jalovište nema izraženih uticaja površinskih i podzemnih voda. Sa istočne strane i severozapadne strane nalazi se Piskanjski potok, koji je regulisan do napuštanja obale kruga fabrike građevinskih elemenata. Prema reci Ibar urađen je nasip visine 3 m', širine 11 m', na kome je urađen obodni put širine 4 m', a zatim nasip visine od 4,6 m' do 7,1 m', širine 11 m'. Ovaj nasip štiti obalu reke i omogućava oticanje površinskih voda. Sa druge strane jalovišta prema taložnicima sitnih klasa uglja jalovište je zaštićeno nasipom visine 6,0 m' a širine 9 m'. Između jalovišta i taložnika sitnih klasa urađen je do fabrike građevinskih elemenata, put širine 4 m'.

5. PROIZVODNJA GRAĐEVINSKIH BLOKOVA OD RUDNIČKE ŠLJAKE

Vreme za koje separacijska jalovina usled oksidacionih procesa dobije oblik rudničke šljake, koja se može koristiti u proizvodnji građevinskih elemenata je 6 do 8 meseci. Rudnička šljaka, utova-rivačem RD

180 se utovara u kamione i sa odlagališta odvozi na drobljenje. Transportna dužina je do 70 m. Drobljenje šljake (1.) se vrši drobilicom sa čekićima UG 3B, kapaciteta 25 t/h. Nakon drobljenja obavlja se klasiranje na klase - 12 mm +8 mm, -8 mm+4 mm i - 4 mm.

Klase se odvajaju u posebne drvene boksove. Klasirana šljaka (2.) iz boksova, skreperom se tovari na transportere sa gumenom trakom i transportuje do postrojenja (3.) za mešanje cementa, šljake i vode. Kapacitet automatskog posreojenja za mešanje smese je 24 m^3/h. Cement se nalazi u silosima (4.) i automatski se dozira u postrojenje. Kapacitet silosa je 200 t cementa. Pripremljena smesa se iz postrojenja istače u kontejner samohodnog nosača smese (5.) na kome su kalupi i odvozi do piste (6.) na kojoj odlažu blokovi (7.) na sušenje.

Kalupi su izmenljivi tako da se mogu, po obliku i dimenzijama praviti različiti tipovi građevinskih blokova. Sušenje blokova traje 21 dan uz povremeno kvašenje. Po celoj pisti za odlaganje blokova razveden je sistem (8.) za snabdvanje vodom.

U smesi od koje se izrađuju blokovi klasirana šljaka ima sledeći odnos:

- šljaka krupnoće - 4 mm, 40 %
- šljaka krupnoće +8 mm - 4 mm, 30 %
- šljaka krupnoće -12 mm + 8 mm, 25 %

Na 1 t šljake dodaje se 0,2 t cementa.

Klase -4 mm se koriste i za druge namene: za potrebe putne privrede, izradu sportskih terena i dr. Proizvodnja građevinskih blokova od rudničke šljake je sezonskog karaktera, odvija se u periodu proleće – jesen. Klase -4mm se mogu pripremati i plasirati na tržište tokom cele godine.

Sl. 1. *Postrojenje za proizvodnjugrađevinskih elemenata*

6. EKONOMSKA ANALIZA

I ako se proizvodnja građevinskih blokova, ne vrši od 2002. godine, postrojenje za proizvodnju građevinskih blokova je i dalje u funkcionalnom stanju, tako da nisu potrebne dodatne investicije za ponovno pokretanje proizvodnje. Na osnovu ranijih iskustava i sadšnjih saznanja, postoji tržište na kome se mogu plasirati ovi proizvodi.

Proizvodnja građevinskih elemenata je sezonska, organizuje se šest meseci u toku godine u periodu april - septembar. Za proizvodnju bi bili angažovani radnici koji su već zaposleni u rudniku. Proizvodnja građevinskih elemenata bi bila organizovana pet radna dana u sedmici u samo u prvoj smeni. Poteban broj neposrednih izvršilaca za proizvodnju građevinskih elemenata dat je u tabeli broj 1. U tabeli broj 2. dat je mogući kapacitet proizvodnje građevinskih blokova. Od ukupnih rezervi šljake, koje iznose 320.000 t, količina od 200.000 t će biti plasirana za putnu privredu, a 120.000

za proizvodnju građevinskih elemenata. Za proizvodnju 1.404.000 komada građevinskih elemenata potrebno je 19.500 t šljake. Vek eksploatacije za rad u jednoj smeni iznosi 6,15 godina. U tabeli broj 3. dati su troškovi proizvodnje po jedinici proizvoda.

Tabela 1. *Poteban broj neposrednih izvršilaca za proizvodnju građevinskih elemenata*

Naziv	Broj izvršilaca
- Rukovaoc drobilice	1
- Pomoćni radnik na drobilici	1
- Rukovaoc skrepera	1
- Rukovaoc postrojenja za mešanje -miksera	1
- Rukovaoc samohodnog nosača smese	1
- Rukovaoc viljuškara	1
- Radnik na negi blokova	1
- UKUPNO	**8**

Tabela 2. *Kapacitet proizvodnje*

- smenska proizvodnja	7.800 kom.
- nedeljna proizvodnja	39.000 kom.
- mesečna proizvodnja	234.000 kom.
- godišnja prizvodnja	1.404.000 kom.

Tabela 3. *Troškovi proizvodnje po jedinici proizvoda*

1.	Cement	20,41	din./kom.
2.	Nafta	0,13	din./kom.
3.	Ulja i maziva	0,02	din./kom.
4.	Elektr. energija	0,59	din./kom.
5.	Troš. održavanja	1,22	din./kom.
6.	Bruto zarade	5,96	din./kom.
7.	Amortizacija	0,44	din./kom.
8.	Osiguranje	0,20	din./kom.
9.	Ostali troškovi	0,03	din./kom.
10.	Proizvodna cena koštanja:	29,00	din./kom.
11.	Troš. PDV 18%	5,40	din./kom.
12.	Cena košt. sa PDV	34,40	din./kom.

Građevinski elementi od rudničke šljake se na tržištu mogu plasirati po ceni od 36,00 din./kom.

Uvažavajući rezerve rudničke šljake, kapacitet proizvodnje i troškove proizvodnje jedinice proizvoda, prodajnu cenu proizvoda, određeni su ekonomski efekti ponovnog pokretanja proizvodnje blokva, a koji iznose: 1.404.000 kom. × 1,6 din./kom. = 2.246.600 din. ili 22.000 EUR.

ZAKLJUČAK

Analiza ekonomskih efekata proizvodnje građevinskih blokova, dala je pozitivne rezultate. Rudnik bi obezbedi dodatni prihod, a na proizvodnji bi mogao da uposli invalide rada ili radnike koji su u određenom periodu kategorisani kao radnici nesposobni za rad u jami. Pokretanjem proizvodnje uklonila bi se jalovina, i istovremeno bi se izvršila i sanacija lokacije odlagališta jalovine.

LITERATURA

[1] Projektna dokumentacija JPPEU-Biro za projektovanje i razvoj Beograd.

[2] Izveštaj o ispitivanju kvaliteta zgure sa deponije Ibarskih rudnika u Baljevcu na Ibru (Jugoslovenski građevinski centar Beograd, 1975. godine.)

[3] Đukanović D., Đukić B., Sanković Ć., 2005: Ukrupnjavanje sitnih asortimana uglja u cilju povećanja finansijske efikasnosti rudnika uglja sa podzemnom eksploatacijom u republici Srbiji, Energija 2/2005, Savez energetičara, Beograd, 244-245;

[4] Đukanović D., Ivković M.,2005: Uticaj podzemne eksploatacije mrkog uglja u RMU „Jasenovac"-Krepoljin na životnu sredinu, Naučno-stručni skup Ekološka istina EcoIst-05.

[5] Maksimović M., Pačkovski G., Jovanović M., Nikolić K. 2009: Ekonomska (vrednosna) ocena tehnogenog ležišta „Depo šljake 1", Rudarski radovi 2/2009, Bor, 45-52.

MINING AND METALLURGY INSTITUTE BOR YU ISSN: 1451-0162
COMMITTEE OF UNDERGROUND EXPLOITATION OF THE MINERAL DEPOSITS UDK: 622

UDK: 553.94:622.333:622.72(045)=20

Duško Đukanović, Milan Popović**, Dragomir Zečević***

ECONOMIC EFFECTS OF THE WASTE USE FROM COAL SEPARATION IN THE IBAR MINES

Abstract

In the Ibar Coal Mines Baljevac by 2005 the cleaning OF run of mine coal was performed in A difficult liquid separation by the "floats-sinks" principle. Tailings from separation were dumped on a site close to separation. During the oxidation processes, the separation tailings were changed; the result of change is new material - mining slag of reddish color. Use of mining slag, resulted from oxidation processes for road industry and production of construction elements is a way to eliminate the tailings with achievement of production and make the rehabilitation of tailing waste dump site and to achieve the economic effects.
Key words: *separation, separation tailings, mine slag, construction elements, the cost of production.*

INTRODUCTION

The Ibar Coal Mines have dealt with the coal exploitation over eighty years in the deposits of Jarand, Usce, Tadenje-Progorelica. Coal mining in the deposit Usce was completed due to the depleted coal reserves. Excavated run of mine coal in the deposits Jarand and Tadenje-Progorelica is valorized in the unique separation in Piskanja, where it is delivered by air lift.

In the period from 1960 to 196 the coal separation was built in Piskanja – Baljevac. During the construction of separation, the tailing dump was built. Separation tailings consist of small and large classes of rocks that occur in the productive series of deposits Jarando, Tadenje - Progorelica and mesogen with combustible materials. On the separation tailing dump, the oxidative processes occur during which the tailings are changed, and the result of change is a new material - mining slag of reddish color.

In 1974, a factory for production of construction elements of mine slag was built, which worked until 2002. Since 2005 there is no coal cleaning by the "floats-sinks" principle or the waste is not deposited at the aforementioned location. It was planned to carry out the rehabilitation of existing tailing dump, what can be done by restarting the production of construction elements and sale of slag to the road industry.

* *PC for Underground Exploitation Resavica, Bureau for Design and Development, Belgrade*
** *The Ibar Mines of Stone Coal - Baljevac*

CHARACTERISTICS OF MINE SLAG

Tailings from the coal cradle may represent very suitable aggregates for production of hollow concrete blocks provided that it is natural burnt clay with chemical composition and gravity that correspond to the set conditions of relevant standards or regulations.

Process of "self-ignition" of these waste materials occurs due to the contamination of primary clay mass by large quantities of coal which was not removed during the process of separation, that is coal washing. Otherwise, the combustion process of coal impurities or burning the whole clay mass is due to the process of spontaneous combustion of sulfur compounds in the clay mass.

This obtained aggregate by the natural process, mining slag, has look and color of baked clay, and it is characterized with the relatively high gravity (800-1000 t/m³) and often very high sulfate content.

Due to its high content of soluble sulfate there was a dispute of possible recovery of such slag as aggregate for concrete production or prefabricated concrete. However, the certain physical tests of several domestic slag showed that the slag even with a high sulfate content have the "constant volume", which has been proven by the test-cooking of cakes with no registered appearance of radial cracks, warping, and all other phenomena that discredit this mine slag in terms of volume constancy.

Chemical analysis of slag from the mine dump in Piskanja

- Moisture at 110°C	0.34 %
- Ignition loss	1.35 %
- SiO_2	49.00 %
- Al_2O_3	11.69 %
- TiO_2	0.37 %
- Fe_2O_3	14.34 %
- MnO	0.10 %
- CaO	9.33 %
- MgO	2.22 %
- Na_2O	0.66 %
- K_2O	2.17 %
- SO_3	8.19 %
T O T A L:	99.76 %

Analyses of chemical composition the mining slag formed by spontaneous combustion of coal from the coal dump in Piskanja have showed that the analyzed slag presents an alumino-silicate matter whose main component is insoluble SiO_2, where the slag is characterized by high content of sulfate (8%). Testing the nature of sulfate, contained in this slag, has determined that the content of soluble sulfate in water is 2.97% as the use of difractometrical analysis determined that the sulfates are usually in the form of mineral anhydrite.

MINE RESERVES OF SLAG

Total reserves of mine slag, which can be used for production of construction elements, aggregates for the road industry or other purposes amounts to about 320,000 t.

LOCATION OF THE DUMP OF SEPARATION TAILINGS

Dump of separation is located on the right side of the river Ibar. The terrain on which it is located is almost is flat, the height differences are small. Larger terrain is tilted towards the river Ibar, which drains the entire terrain, and tailing dump has no pronounced influence of surface and ground water. On its east side and north-west side there is the Piskanja stream, which is regulated to the coast leaving the factory of construction elements. An embankment, height 3 m', width 11 m', was built towards the Ibar river, where circumferential road was constructed, width 4 m', and then the embankment, height of 4.6 m' to 7.1 m', width 11 m'. This embankment protects the river side and allows run-off of the surface water. On the other side of tailing dump, towards the settlers of small coal classes, the tailing dump is protected by an embankment, height 6.0 m' and width o 9 m'. Between the tailing dump and settler of small classes, the road, width 4 m', was constructed to the factory of construction elements.

PRODUCTION OF CONSTRUCTION BLOCKS OF MINE SLAG

Time for which the separation tailings, due to the oxidation processes, takes the form of mine slag, which can be used in the production of construction elements, is 6 to 8 months. Mine slag is loaded into trucks using the RD 180 loader and transported from the dump to the crushing. Transport length is up to 70 m. Slag crushing (1) is carried out by the shredder hammers UG 3B, capacity 25 t/h. After crushing, the sizing is done to the classes -12 mm + 8 mm, -8 mm+4 mm and - 4 mm.

Classes are separated into special wooden boxes. Classified slag (2) from boxes, is loaded by the scraper on belt conveyors and transported to the plant (3) for mixing of cement, slag and water. Capacity of automatic plant for mixing is 24 m^3/h. Cement is situated in silos (4) and dosed automatically into plant. Silo capacity is 200 tons of cement. The prepared mixture from the plant is unloaded into container of the self-driven mixture carrier (5) on which it is molded and transported to the runway (6) where the blocks are delayed (7) for drying. Moulds are changeable such various types of construction blocks can be made by form and size. Drying of blocks is 21 days with occasional wetting. All over the catwalk for storage of blocks, the system for water supply (8) is distributed.

The mixtures for production of blocks, the classified slag has the following ratio:

- size class of slag -4 mm, 40%
- size class of slag +8 mm - 4 mm, 30%
- size class of slag -12 mm + 8 mm, 25%

0.2 t of cement is added to 1 t of slag

Classes of -4 mm are also used for other purposes: for the needs of road industry, construction of sports facilities and others. Production of construction blocks of mine slag is seasonal and it takes place during spring - autumn. Classes of -4 mm could be prepared and sent to the market throughout the year.

Figure 1. *Plant for production the construction elements*

ECONOMIC ANALYSIS

Although the production of construction blocks has not done since 2002, the plant for production of construction blocks is still in operational condition, so the additional investments are not need to restart the production. Based on the previous experiences and current knowledge, there is a market where these products can be placed.

Production of construction elements is seasonal; it is organized six months during the year from April-September. The workers, already employed in the mine, would be engaged for production. Production of construction elements would be organized five working days of the week in just the first shift. The required number of direct employees for production of construction elements is given in Table 1. Table 2 gives the potential production capacity of construction blocks. Out of the total reserves of slag, which amount to 320,000 tons, the amount of 200,000 tons will be placed for road industry, and 120,000 for production of construction elements. For the production of 1,404,000 pieces of construction elements, 19,500 tons of slag is required. Exploitation time for work in one shift is 6.15 years. Table 3 gives the production costs per unit of product.

Table 1. *Required number of direct employees for the production of construction elements*

Name	Number of employees
- Operator of crusher	1
- Support worker of crusher	1
- Operator of scraper	1
- Operator of mixing plant - mixer	1
- Operator of self-driven compound carrier	1
- Operator of forklift	1
- Worker on block care	1
- TOTAL	8

Table 2. *Production capacity*

- shift production	7,800 pieces
- weekly production	39,000 pieces
- monthly production	234,000 pieces
- annual production	1,404,000 pieces

Table 3. *Production costs per unit*

1.	Cement	20.41	din./pcs.
2.	Petroleum	0.13	din./pcs.
3.	Oils and lubricants	0.02	din./pcs.
4.	Electricity	0,59	din./pcs.
5.	Maintenance costs	1.22	din./pcs.
6.	Gross profit	5.96	din./pcs.
7.	Depreciation	0.44	din./pcs.
8.	Insurance	0.20	din./pcs.
9.	Other charges	0.03	din./pcs.
10.	Production price of costs	29.00	din./pcs.
11.	Costs of VAT 18%	5.40	din./pcs.
12.	Costs without VAT	34.40	din./pcs.

Construction elements of mine slag can be sold on the market at price of 36.00 din./pcs.

Recognizing the mining slag reserves, the production capacity and unit production costs of products and the selling price of products, the economic effects of re-starting the production of blocks were determined and they are the following:

1,404,000 pcs. × 1.6 din./pcs. =
= 2,246,600 din. or 22,000 EUR.

CONCLUSION

Analysis of the economic effects of production the construction blocks, has given the positive results. The mine would provide the additional income, and the production could employ disabled workers or workers who were ranked in the certain period as employees unable to work in the pit. Launching of production would remove the tailings, and also the remediation of the tailing dump would be performed.

REFERENCES

[1] Project Documentation of the JP PEU-Bureau for Design and Development, Belgrade (in Serbian)

[2] Report on Testing the Quality of slag from the Dump of the Ibar Mines in Baljevac on Ibar, Yugoslav Civil Center, Belgrade, 1975 (in Serbian)

[3] Djukanović D., Djukić B., Sanković Ć., 2005: Sizing the Small Size Assortments of Coal in Order to Increase the Financial Effects of the Coal Mines with the Underground Mining in the Republic of Serbia, Energy 2/2005, Association of Energetic, Belgrade, pp.244-245 (in Serbian)

[4] Djukanović D., Ivković M.,2005: Effect of Underground Mining of Brown in the Brown Coal Mine "jasenova" - Krepoljin on the Environmen, Scientific and Research Conference Ecological Truth EcoIst-05 (in Serbian)

[5] Maksimović M., Pačkovski G., Jovanović M., Nikolić K. 2009: Economic Evaluation of Technogenetic Deposit "Depot of Slag 1", Mining Engineering 2/2009, Bor, pp.45-52 (in Serbian)

INSTITUT ZA RUDARSTVO I METALURGIJU BOR YU ISSN: 1451-0162
KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

UDK:330.1:622(045)=861

Mile Bugarin, Gordana Slavković*, Zoran Stojanović**

UTVRĐIVANJE CENE KOŠTANJA U EKONOMSKOJ ANALIZI RUDARSKOG PROJEKTA

Izvod

U radu se predstavlja značaj utvrđivanja cene koštanja prilikom ekonomskog ocenjivanja projekata u rudarstvu. Na bazi tehničkih podloga određuju se svi tzv. inputi u projektovanoj rudarskoj proizvodnji: investicije, utvrđeni normativi materijala i energenata, zatim određene stope amortizacije, kamate na kredite, potrebna radna snaga, održavanje opreme, razni transakcioni troškovi i zakonske obaveze. Ekonomskom evulacijom definiše se troškovna strana poslovnih bilansa, što je predstavljeno na primeru rudarskog projektovanja za ležište Čoka Marin.

Ključne reči: *proizvodna cena, projekat, rudarstvo, ocenjivanje*

1. UVOD

Projektovane investicije, utvrđeni normativi materijal i energenati u tehničkom delu, zatim određene stope amortizacije, kamate na kredite, potrebna radna snaga, održavanje opreme, razni transakcioni troškovi i zakonske obaveze, odredili su većinu inputa za definisanje kalkulativnih troškova produkcije metala u ležištu Ćoka Marin. Ukupni troškovi mogu se podeliti na varijabilne, relativno fiksne i fiksne troškove u funkciji godišnje proizvodnje.

2. POLAZNI PARAMETRI ZA OBRAČUNA TROŠKOVA

Obračun troškova za rudarski deo i deo prerade-PMS izvršen je na bazi podloga tehničko-tehnološkog dela.

Svi obračuni su u američkim dolarima: USD

- Obračun troškova normativnog materijala je na osnovu projektovanih (planiranih) fizičkih utrošaka po jedinici i cena koje su sagledane na bazi ostvarenja i prognoze.

* *Institut za rudarstvo i metalurgiju Bor*

Tabela 2.1. *God.obračun troškova normativnog materjala za otkopavanje*

					20000 t
Naziv	**Jed.normt.**	**Ukupno**	**Cena**	**U 000**	**Po t rude**
Eksploziv praškasti	**0.46444**	**9288.8**	**1.2**	**11.15**	**0.5575**
elektro upaljači	**0.32418**	**6483.6**	**0.85**	**5.51**	**0.2755**
kabla za miniranje	**0.00888**	**177.6**	**0.1**	**0.02**	**0.001**
monoblok burgije	**0.00042**	**8.4**	**43.5**	**0.37**	**0.0185**
monoblok burg. l=1,6m	**0.00094**	**18.8**	**68.7**	**1.29**	**0.0645**
bušaće šipke	**0.0002**	**4**	**94.5**	**0.38**	**0.019**
nastavne šipke	**0.00021**	**4.2**	**105**	**0.44**	**0.022**
početni segment	**0.0002**	**4**	**94**	**0.38**	**0.019**
krune za busen. 26,5	**0.00074**	**14.8**	**26.5**	**0.39**	**0.0195**
industrijska voda	**0.00919**	**183.8**	**0.14**	**0.03**	**0.0015**
ulje i mazivo	**0.14889**	**2977.8**	**2.8**	**8.34**	**0.417**
plast. cevi za vodu	**0.00349**	**69.8**	**2.91**	**0.2**	**0.01**
creva vazduh i voda	**0.00599**	**119.8**	**2.67**	**0.32**	**0.016**
plastične cevi vetr.	**0.00517**	**103.4**	**72**	**7.44**	**0.372**
čelič. anker 1,6 m	**0.00706**	**141.2**	**7.31**	**1.03**	**0.0515**
čelič. anker 0,6 m	**0.00706**	**141.2**	**5.6**	**0.79**	**0.0395**
čelična mreža	**0.01049**	**209.8**	**3.8**	**0.8**	**0.04**
komprimirani vazduh	**73.35**	**1467000**	**0.004**	**5.87**	**0.2935**
obla jamska građa	**0.00004**	**0.8**	**53**	**0.04**	**0.002**
rezana jamska građa	**0.00003**	**0.6**	**220**	**0.13**	**0.0065**
gume za utovarivač	**0.00079**	**15.8**	**2000**	**31.6**	**1.58**
ulje i maziv. sp. tran	**0.3**	**6000**	**2.8**	**16.8**	**0.84**
gume sp. tran	**0.000625**	**12.5**	**1000**	**12.5**	**0.625**
ulje i mazivo utovar	**0.123**	**2460**	**2.8**	**6.89**	**0.3445**
gume utovar	**0.000366**	**7.32**	**2000**	**14.64**	**0.732**
el. energija	**12.46**	**249200**	**0.06**	**14.95**	**0.7475**
dizel gorivo	**1.49**	**29800**	**1.3**	**38.74**	**1.937**
gorivo dizel utovar.	**1.19**	**23800**	**1.3**	**30.94**	**1.547**
gorivo dizel sp.tran	**3.75**	**75000**	**1.3**	**97.5**	**4.875**
Ukupno				**309.48**	**15.474**

Tabela 2.2. *God.obračun troškova normativnog materjala za PMS*

					19.500 t
	Jed.normt.	**Ukupno**	**Cena**	**U 000**	**Po t rude**
2 101 ulja i maziva	**0.02**	**390**	**3.18**	**1.24**	**0.064**
2 103 čelične obloge	**0.005**	**97.5**	**4.18**	**0.41**	**0.021**
2 104 čelične obloge	**0.072**	**1404**	**4.18**	**5.87**	**0.301**
2 105 šipke	**0.46**	**8970**	**1.16**	**10.41**	**0.534**
2 106 kugle	**0.8**	**15600**	**1.18**	**18.41**	**0.944**
2 107 filter platno	**0.0005**	**9.75**	**7.45**	**0.07**	**0.004**
2 108 D250	**0.04**	**780**	**5.46**	**4.26**	**0.218**
2 109 KAX	**0.125**	**2437.5**	**3.42**	**8.34**	**0.428**
2 110 3418A	**0.125**	**2437.5**	**5.5**	**13.41**	**0.688**
2 111 KREČ	**10**	**195000**	**0.08**	**15.6**	**0.800**
2 112 obloge gumene	**0.04**	**780**	**5.6**	**4.37**	**0.224**
2 113 sveza indust. v	**2.3**	**44850**	**2.18**	**97.77**	**5.014**
2 201 el. energija	**25**	**487500**	**0.06**	**29.25**	**1.500**
Ukupno				**209.41**	**10.739**

- Troškovi održavanja su računati u iznosu od 5% u odnosu na nabavnu vrednost sredstava.
- Amortizacija osnovnih sredstava je utvrđena po važećim zakonskim propisima za nova ulaganja.

Tabela. 2.3. *Obračun troškova amortizacije*

NAZIV	GOD.	NAB.VRED.	STOPA AMORT.	AMORTIZ.	ISPRAVKA VRED.	KRAJNJA RED.
1 oprema RUD.	1	1,170	12.50	146	1,170	0
		1,170		146	1,170	0
2 POSTOJECA PMS	1	216	12.50	27	216	0
1 gr.obj.pr./otk	1	131	2.50	3	33	98
1 gr.rad.pr.i ot	1	35	2.50	1	9	26
1 gr.rad.pr.i ot	1	81	2.50	2	20	61
1 gr.rad.pr.i ot	6	50	2.50	1	6	44
1 gr.rad.pr.i ot	7	11	2.50	0	1	10
1 gr.rad.pr.i ot	7	45	2.50	1	4	41

1 gr.rad.pr.i ot	7	36	2.50	1	4	32
1 gr.rad.pr.i ot	8	45	2.50	1	3	42
1 gr.rad.pr.i ot	9	22	2.50	1	1	21
1 izrada podz.pr	1	598	2.50	15	149	449
1 pr.rad.pr.i ot	2	18	2.50	0	4	14
1 pr.rad.pr.i ot	2	40	2.50	1	9	31
1 pr.rad.pr.i ot	2	40	2.50	1	9	31
1 pr.rad.pr.i ot	3	51	2.50	1	10	41
1 pr.rad.pr.i ot	4	29	2.50	1	5	24
1 pr.rad.pr.i ot	4	65	2.50	2	11	54
1 pr.rad.pr.i ot	4	21	2.50	1	4	17
1 pr.rad.pr.i ot	5	50	2.50	1	7	43
		1,584		61	505	1,079
1 istrazni radov	1	1,850	20.00	370	1,850	0
1 ekologija	1	150	20.00	30	150	0
1 projektovanje	1	50	20.00	10	50	0
		2,050		410	2,050	0
U K U P N O:		4,804		617	3,727	1,077

- Bruto zarade radnika su obračunate za projektovani broj radnika (predvidjen u tehničkom delu) u visini od 1000 US$ mesečno po radniku za sedam meseci godisnje.

Tabela. 2.4. *Obračun troškova radne snage*

Opis	Broj radnika	Mesečna bruto zarada u USD	Ukupno u hiljd. godišnje (7 mes.)
1.Rudnik	28	1.000	196
2.PMS	10	1.000	70
Ukupno	**38**	**1.000**	**266**

- Naknada za korišćenje mineralnih sirovina obračunata je prema zakonskoj regulativi (3% na neto prihod topionice).

Tabela 2.5. *Obračun naknade za korišćenje mineralne sirovine u 000*

GODINE	PRIHOD	NAKNADA
1	2844	85,32
2	2844	85,32
3	2963	88,89
4	2963	88,89
5	2963	88,89
6	2963	88,89
7	2963	88,89
8	2963	88,89
9	2963	88,89
10	2963	88,89

- Troškovi prevoza koncentrata do topionice u Boru iznose 4 USD po t koncentrata-godišnje iznose 43.680 USD.
- Ostali materijalni i nematerijalni troškovi su procenjeni u odnosu na ukupan prihod.

Tabela 2.6. *Obračun ostalih troškova u 10.god.*

	OPIS		U 000
GODINA: 10			
1	1 NAKNADA		44.00
2	1 NAKNADA		44.00
1	2 OSTALI TROSKOVI		180.00
2	2 OSTALI TROSKOVI		100.00
2	5 prevoz koncentrata		43.68
1	6 rekultivacija	2.79	
1	101 NEMATERJALNI TROSK		100.00
2	101 NEMATERJALNI TROSK		86.00
2	101 NEMATERJALNI TROSK		0.00
Subtotal **			
			600.47

- Porez na dobit je utvrđen po stopi od 10%.
- Investiciona ulaganja u osnovna sredstva iznose: 6.104.000 USD i finansiraju se iz kredita pod sledećim uslovima:

- Iznos ukupnog kredita iznosi 4.200.000 USD
- rok vraćanja kredita je 5 godine
- kamatna stopa je 12% godišnje,
- otplata kredita: jednaki godišnji anuiteti

Tabela 2.7. *Ulaganja i izvori finansiranja*

	u 000 USD	%
1.Ukupno Osnovna sredstva	5304	0.87
2. Obrtna sredstva	800	13.11
Ukupne investic.	6104	100.00
IZVORI		
1.Sopstvena sred.	1904	31.19
2.Krediti	4200	68.81
UKUPNI IZVORI	6104	100.00

Tabela 2.8. *Plan otplate kredita*

Uslovi: Rok: 5 Kam.: 12.000% UC.: 0% Grac: 0

	Anuitet	kamata	otplata	dug
1	1165	504	661	4200
2	1165	425	740	3539
3	1165	336	829	2798
4	1165	236	929	1969
5	1165	125	1040	1040
Tot:	5825	1626	4199	
Pros:	1165	325	840	
Ukupno				
	5825	1626	4199	

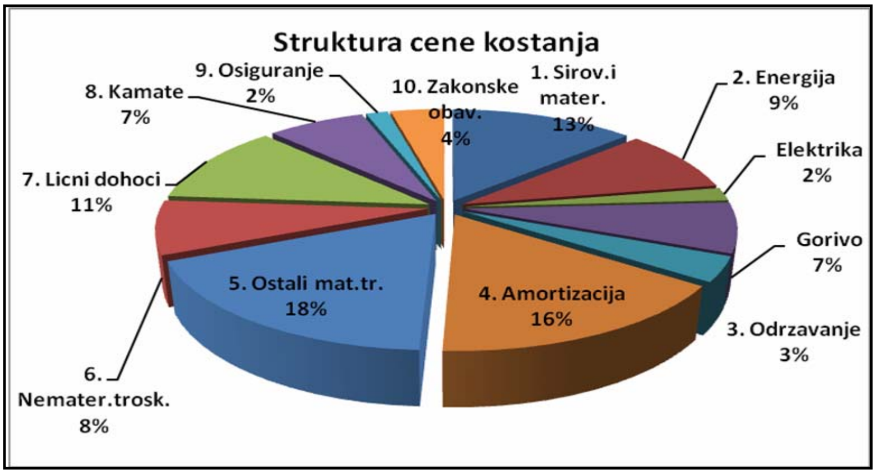
3. OBRAČUN CENE KOŠTANJA

Na osnovu sagledanih svih potrebnih inputa, formirana je cena koštanja: ukupna i tzv. fazna .

Puna cena koštanja po toni rude iznosi 106,4 USD/t rude.

Tabela 3.1. *Cena koštanja u 000*

GODINE	1	2	3	4	5	6	7	8	9	10	UKU-PNO	PRO-SEK	PO T/RUDE
1. Sirov.i mater.	308	308	308	308	308	308	308	308	308	308	3075	308	15.4
2. Energija	211	211	211	211	211	211	211	211	211	211	2114	211	10.55
Elektrika	44	44	44	44	44	44	44	44	44	44	442	44	2.2
Gorivo	167	167	167	167	167	167	167	167	167	167	1672	167	8.35
3. Odrzavanje	95	95	95	95	95	95	95	95	36	36	833	83	4.15
4. Amortizacija	604	607	608	611	612	204	206	207	34	34	3727	373	18.65
5. Ostali mat.tr.	418	416	418	418	418	418	418	418	418	420	4178	418	20.9
6. Nemater.trosk.	180	180	180	180	180	180	180	180	180	180	1800	180	9
7. Licni dohoci	266	266	266	266	266	266	266	266	266	266	2660	266	13.3
8. Kamate	504	425	336	236	125						1626	163	8.15
9. Osiguranje	42	42	42	42	42	42	42	42	15	15	368	37	1.85
I. TROS. POSLOVANJA-cena kostanja	*2629*	*2549*	*2464*	*2367*	*2257*	*1723*	*1726*	*1727*	*1468*	*1471*	*20380*	*2038*	*101.9*
10. Zakon. obav.	21	29	50	60	71	124	124	124	150	149	901	90	4,5
II. PUNA CENA KOST.	2650	2579	2515	2427	2328	1847	1849	1850	1617	1620	21282	2127	106,4



Sl. 1. *Struktura cene koštanja za Čoka Marin*

Slika 1 pokazuje procentualno učešće pojedinih troškova u ukupnom iznosu troškova.

Tabela 3.1.a. *Cena koštanja –faza otkopavanja u 000*

GODINE	1	2	3	4	5	6	7	8	9	10	UKU-PNO	PRO-SEK	PO T
1. Sirov.i mater.	127	127	127	127	127	127	127	127	127	127	1274	127	6.35
2. Energija	182	182	182	182	182	182	182	182	182	182	1821	182	9.10
Elektrika	15	15	15	15	15	15	15	15	15	15	150	15	0.75
Gorivo	167	167	167	167	167	167	167	167	167	167	1672	167	8.35
3. Odrzavanje	95	95	95	95	95	95	95	95	36	36	833	83	4.15
4. Amortizacija	577	580	581	584	585	177	179	180	34	34	3511	351	17.55
5. Ostali mat.tr.	230	226	227	227	227	227	227	227	227	230	2274	227	11.35
6. Nema-ter.trosk.	100	100	100	100	100	100	100	100	100	100	1000	100	5.00
7. Licni dohoci	196	196	196	196	196	196	196	196	196	196	1960	196	9.80
8. Kamate	504	425	336	236	125						1626	163	8.15
9. Osiguranje	38	38	38	38	38	38	38	38	15	15	333	33	1.65
I. TROS. POSLOVANJA	2049	1969	1882	1786	1676	1142	1144	1145	918	921	14632	1463	73.15

Cena koštanja po toni rude za fazu otkopavanja iznosi 73,15 USD .

Tabela 3.1.b. *Cena koštanja –faza PMS u 000*

GODINE	1	2	3	4	5	6	7	8	9	10	UKUPNO	PROSEK	PO T
Sirov.i mater.	180	180	180	180	180	180	180	180	180	180	1802	180	9.23
2. Energija	29	29	29	29	29	29	29	29	29	29	293	29	1.49
Elektrika	29	29	29	29	29	29	29	29	29	29	293	29	1.49
3. Odrzavanje													
4. Amortizacija	27	27	27	27	27	27	27	27			216	22	1.13
5. Ostali mat.tr.	189	190	191	191	191	191	191	191	191	191	1904	190	9.74
6. Nemater. trosk.	80	80	80	80	80	80	80	80	80	80	800	80	4.10
7. Licni dohoci	70	70	70	70	70	70	70	70	70	70	700	70	3.59
8. Kamate													
9. Osiguranje	4	4	4	4	4	4	4	4	4	4	35	4	0.21
I. TROS. POSLOVANJA	579	580	581	581	581	581	581	581	550	550	5748	575	**29.49**

Cena koštanja po toni rude za fazu prerade iznosi 29,49 USD .

4. ZAKLJUČAK

Ekonomskom evulacijom definiše se troškovna strana poslovnih bilansa , što je predstavljeno na primeru rudarskog projektovanja za ležište Čoka Marin. Inputi za definisanje kalkulativnih troškova produkcije metala u ležištu odredjeni su na bazi: projektovanih investicija, utvrđenih normativih materijala i energenata u tehničkom delu, zatim određene stope amortizacije, kamate na kredite, potrebne radna snaga, održavanja, ostalih potrebnih troškova i zakonskih obaveza. Utvrđena je ukupna cena koštana i fazna cena koštanja.

LITERATURA:

[1] B. Cavender, Mineral Production Costs - Analyses and Management, SME, 1999.

[2] N. Dondur, Ekonomska analiza projekata, Mašinski fakultet, Beograd 2002.

[3] G. Mankju, Principi ekonomije, Ekonomski fakultet Beograd, 2005.

[4] M. Bugarin, G. Slavković "Tehno-ekonomska ocena " Institut za bakar, Bor, 2006.

[5] T. Kuronen: Capital Budgeting In A Capital-Intensive Industry, Helsinki University Of Technology, Mat-2.108 Independent research projects in applied mathematics, 2007.

[6] M. Bugarin, G. Slavković, M. Maksimović, Vrednovanje korisne sirovine Čoka Marina, Rudarski radovi, br. 2-2010, str. 17-23

UDK:330.1:622(045)=20

Mile Bugarin, Gordana Slavković*, Zoran Stojanović**

DETERMINATION OF COST PRICE IN THE ECONOMIC ANALYSIS OF MINING PROJECT

Abstract

This paper presents the importance of determination the cost price in the economic evaluation of mining projects. Based on the technical backgrounds, all so-called inputs in designed mining production are determined: investments, set standards of materials and energy, then the specific depreciation rate, loan interests, the required manpower, equipment maintenance, various transaction costs and legal obligations. Economic evaluation defines the cost side of the balance sheets, what was present in the example of mining design the deposit Čoka Marin.

Key words: *cost price, project, mining, evaluation*

1. INTRODUCTION

The designed investments, set standards of materials and energy in the technical part, then the specific rates of depreciation, loan interests, required manpower, equipment maintenance, various transaction costs and legal obligations, have determined the majority of inputs for defining the calculation costs of metal production in the deposit Čoka Marin. Total costs can be divided into variable, relatively fixed and fixed costs as a function of annual production.

2. INPUT PARAMETERS FOR COSTING

Costing for the mining part and part of mineral was done based on the backgrounds of technical – technological part.

All calculations are in U.S. dollars: USD

- Costing of normative materials is on the basis of designed (planned) hysical consumptions per unit and prices that are analyzed based on realization and forecast.

* *Mining and Metallurgy Institute Bor*

Table 2.1. *Costing of annual normative material for mining*

Name	Unit normative	Total	Price	In 000	20,000t t /ore
Powdered explosive	0.46444	9288.8	1.2	11.15	0.5575
Electric igniters	0.32418	6483.6	0.85	5.51	0.2755
Cable for blasting	0.00888	177.6	0.1	0.02	0.001
Monoblock drills	0.00042	8.4	43.5	0.37	0.0185
Monoblock drill l = 1.6 m	0.00094	18.8	68.7	1.29	0.0645
Drill rods	0.0002	4	94.5	0.38	0.019
Drill rod extension	0.00021	4.2	105	0.44	0.022
Initial segment	0.0002	4	94	0.38	0.019
Drill bits 26.5	0.00074	14.8	26.5	0.39	0.0195
Industry water	0.00919	183.8	0.14	0.03	0.0015
Oil and lubricant	0.14889	2977.8	2.8	8.34	0.417
Plastic pipes for water	0.00349	69.8	2.91	0.2	0.01
Air and water hoses	0.00599	119.8	2.67	0.32	0.016
Plastic pipes for ventilation	0.00517	103.4	72	7.44	0.372
Steel anchor 1.6 m	0.00706	141.2	7.31	1.03	0.0515
Steel anchor 0.6m	0.00706	141.2	5.6	0.79	0.0395
Steel grid	0.01049	209.8	3.8	0.8	0.04
Compressed air	73.35	1467000	0.004	5.87	0.2935
Round pit timber	0.00004	0.8	53	0.04	0.002
Cut pit timber	0.00003	0.6	220	0.13	0.0065
Tires for loader	0.00079	15.8	2000	31.6	1.58
Oil and lubricant for conveyor	0.3	6000	2.8	16.8	0.84
Tires for conveyor	0.000625	12.5	1000	12.5	0.625
Oil and lubricant for loader	0.123	2460	2.8	6.89	0.3445
Tires for loader	0.000366	7.32	2000	14.64	0.732
Electric energy	12.46	249200	0.06	14.95	0.7475
Diesel	1.49	29800	1.3	38.74	1.937
Diesel for loader	1.19	23800	1.3	30.94	1.547
Diesel for conveyor	3.75	75000	1.3	97.5	4.875
TOTAL				**309.48**	**15.474**

Table 2.2. *Costing of annual normative material for mineral processing*

Name	Unit normative	Total	Price	In 000	19,500 t t/ore
2 101 Oil and lubricant	0.02	390	3.18	1.24	0.064
2 103 Steel linings	0.005	97.5	4.18	0.41	0.021
2 104 Steel linings	0.072	1404	4.18	5.87	0.301
2 105 Rods	0.46	8970	1.16	10.41	0.534
2 106 Balls	0.8	15600	1.18	18.41	0.944
2 107 Filter cloth	0.0005	9.75	7.45	0.07	0.004
2 108 D250	0.04	780	5.46	4.26	0.218
2 109 KAX	0.125	2437.5	3.42	8.34	0.428
2 110 3418A	0.125	2437.5	5.5	13.41	0.688
2 111 Lime	10	195000	0.08	15.6	0.800
2 112 Rubber linings	0.04	780	5.6	4.37	0.224
2 113 Fresh industry water	2.3	44850	2.18	97.77	5.014
2201 Electrical energy	25	487500	0.06	29.25	1.500
TOTAL				**209.41**	**10.739**

- Maintenance costs were calculated in the amount of 5% compared to the purchase value of assets.

- Depreciation of fixed assets was determined by the applicable legislations for the new investments.

Table 2.3 *Costing of depreciation*

name	year	p.value	rate .	depreciation	Correction of value	Final value
1 Mining equiment	1	1,170	12.50	146	1,170	0
		1,170		146	1,170	0
2 Mineral processing existing equipment	1	216	12.50	27	216	0
1 mining facilities	1	131	2.50	3	33	98
1 construct.works	1	35	2.50	1	9	26
1 construct.works	1	81	2.50	2	20	61
1 construct.works	6	50	2.50	1	6	44
1 construct.works	7	11	2.50	0	1	10
1 construct.works	7	45	2.50	1	4	41
1 construct.works	7	36	2.50	1	4	32
1 construct.works	8	45	2.50	1	3	42
1 construct.works	9	22	2.50	1	1	21

1 product.under.rooms	1	598	2.50	15	149	449
1 construct.works	2	18	2.50	0	4	14
1 construct.works	2	40	2.50	1	9	31
1 construct.works	2	40	2.50	1	9	31
1 construct.works	3	51	2.50	1	10	41
1 construct.works	4	29	2.50	1	5	24
1 construct.works	4	65	2.50	2	11	54
1 construct.works	4	21	2.50	1	4	17
1 construct.works	5	50	2.50	1	7	43
		1,584		61	505	1,079
1 prospecting works	1	1,850	20.00	370	1,850	0
1 ecology	1	150	20.00	30	150	0
1 design	1	50	20.00	10	50	0
		2,050		410	2,050	0
T O T A L:		4,804		617	3,72	1,077

- Gross earnings of workers were calculated for designed number of workers (estimated in the technical part) in the amount of U.S.$ 1,000 per month per worker for seven months a year.
- Compensation fee for the use of mineral resources is calculated by the legislation (3% of the net Smelter revenue).

Table 2.4. *Calculation of labor costs*

Description	Number of worker	Gross earnings	Total for 7 month
1. Mining	28	1,000	196
2. Mineral Proessing	10	1,000	70
Total	**38**	**1,000**	**266**

Table 2.5. *Calculation the compensa tion fee for the use of mineral resources in 000*

Year	Income	Fee
1	2844	85.32
2	2844	85.32
3	2963	88.89
4	2963	88.89
5	2963	88.89
6	2963	88.89
7	2963	88.89
8	2963	88.89
9	2963	88.89
10	2963	88.89

- Transport costs of concentrate to the Smelter in Bor are 4 USD/t of concentrate - annually amount $ 43,680
- Other material and nonmaterial costs were estimated in relation to the total income.

Table 2.6. *Calculation of other costs in 2010*

		DESCRIPTION	000
YEAR:	2010		
1	1	FEE	44.00
2	1	FEE	44.00
1	2	OTHER COSTS	180.00
2	2	OTHER COSTS	100.00
2	5	TRANSPORT OF CONCENTRATE	43.68
1	6	RECULTIVATION	2.79
1	101	NONMATERIAL COSTS	100.00
2	101	NONMATERIAL COSTS	86.00
Subtotal **			
			600.47

- Income tax is determined by the rate of 10%.
- Investments in the fixed assets amount to: $ 6,104,000 are financed by the loans under the following conditions:

- The amount of total loans: 4,200,000 USD $
- Loan repayment period: 5 years
- Interest rate: 12%/year
- Repayment of loans: equal annual annuities

Table 2.7. *Investments and funding sources*

	000 USD	%
1. TOTAL FIXED ASSETS	5304	0.87
2. CURRENT ASSETS	800	13.11
TOTAL INVESTMENTS	**6104**	**100.00**
SOURCES		
1. Own funds	1904	31.19
2. Loans	4200	68.81
TOTAL SOURCES	**6104**	**100.00**

Table 2.8. *Loan Repayment Plan*

TERMS:	PERIOD:5	RATE: 12.000%	UC.:0%	GRACE 0:
	Annuity	interest	payment	loan
1	1165	504	661	4200
2	1165	425	740	3539
3	1165	336	829	2798
4	1165	236	929	1969
5	1165	125	1040	1040
Total:	5825	1626	4199	
Aver.:	1165	325	840	
Total				
	5825	1626	4199	

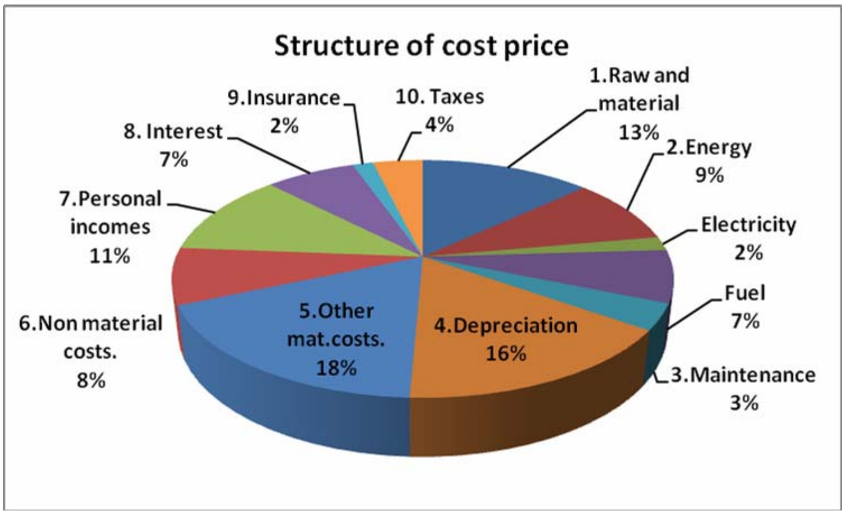
3. CALCULATION OF THE COST PRICE

Based on the analyzed all the necessary inputs, the cost price was formed: the total and per phases.

Full cost per ton of ore is USD $ 106.4.

Table 3.1. *Total cost price in 000*

Year	1	2	3	4	5	6	7	8	9	10	Total	Average	Per tone
1 Raw mat.&mater.	308	308	308	308	308	308	308	308	308	308	3075	308	15.4
2 Energy	211	211	211	211	211	211	211	211	211	211	2114	211	10.55
Electricity	44	44	44	44	44	44	44	44	44	44	442	44	2.2
Fuel	167	167	167	167	167	167	167	167	167	167	1672	167	8.35
3 Maintenance	95	95	95	95	95	95	95	95	36	36	833	83	4.15
4 Depreciation	604	607	608	611	612	204	206	207	34	34	3727	373	18.65
5 Other mat. costs	418	416	418	418	418	418	418	418	418	420	4178	418	20.9
6 Nonmaterial costs	180	180	180	180	180	180	180	180	180	180	1800	180	9
7 Personal incomes	266	266	266	266	266	266	266	266	266	266	2660	266	13.3
8 Interest	504	425	336	236	125						1626	163	8.15
9 Insurance	42	42	42	42	42	42	42	42	15	15	368	37	1.85
I Operative costs – cost price	2629	2549	2464	2367	2257	1723	1726	1727	1468	1471	20380	2038	101.9
10 Taxes	21	29	50	60	71	124	124	124	150	149	901	90	4,5
II Full cost price	2650	2579	2515	2427	2328	1847	1849	1850	1617	1620	21282	2127	106,4



Graph chart 1. *The structure of cost price for Čoka Marin*

Graph chart shows the percentage share of certain expenses in the total amount of costs.

Table 3.1.a *Cost price – stage of mining in 000*

YEAR	1	2	3	4	5	6	7	8	9	10	TOTAL	AVERAGE	PER t ORE
1 Raw mat.&mater.	127	127	127	127	127	127	127	127	127	127	1274	127	6.35
2 Energy	182	182	182	182	182	182	182	182	182	182	1821	182	9.10
Electricity	15	15	15	15	15	15	15	15	15	15	150	15	0.75
Fuel	167	167	167	167	167	167	167	167	167	167	1672	167	8.35
3 Maintenance	95	95	95	95	95	95	95	95	36	36	833	83	4.15
4 Depreciation	577	580	581	584	585	177	179	180	34	34	3511	351	17.55
5 Other mat.costs.	230	226	227	227	227	227	227	227	227	230	2274	227	11.35
6 Nonmaterial costs.	100	100	100	100	100	100	100	100	100	100	1000	100	5.00
7 Personal incomes	196	196	196	196	196	196	196	196	196	196	1960	196	9.80
8 Interest	504	425	336	236	125						1626	163	8.15
9 Insurance	38	38	38	38	38	38	38	38	15	15	333	33	1.65
I Operative costs	2049	1969	1882	1786	1676	1142	1144	1145	918	921	14632	1463	73.15

Cost price per ton of ore for the mining stage is $ 73.15.

Tabela 3.1.b. *Cost price – stage of mineral processing in 000*

YEAR	1	2	3	4	5	6	7	8	9	10	TOTAL	AVERAGE	PER t ORE
1 Raw mat.&mater.	180	180	180	180	180	180	180	180	180	180	1802	180	9.23
2 Energy	29	29	29	29	29	29	29	29	29	29	293	29	1.49
Electricity	29	29	29	29	29	29	29	29	29	29	293	29	1.49
3 Maintenance													
4 Depreciation	27	27	27	27	27	27	27	27			216	22	1.13
5 Other mat.costs.	189	190	191	191	191	191	191	191	191	191	1904	190	9.74
6 Nonmaterial costs.	80	80	80	80	80	80	80	80	80	80	800	80	4.10
7 Personal incomes	70	70	70	70	70	70	70	70	70	70	700	70	3.59
8 Interest													
9 Insurance	4	4	4	4	4	4	4	4	4	4	35	4	0.21
I Operative costs	579	580	581	581	581	581	581	581	550	550	5748	575	**29.49**

Cost price per ton of ore for the processing stage is $ 29.49.

4. CONCLUSION

Economic evaluation defines the cost side of balance sheets, what is present on the example of mining design for the Čoka Marin deposit. Inputs for defining the calculation costs of metal production in the deposit were determined on the basis of: the designed investments, determined normative materials and energy in the technical section, then the specific rate depreciation rate, interest on loans, the required labor, maintenance, and other necessary expenses and legal obligations. The total price was determined as well as the stage cost price.

REFERENCES

[1] B. Cavender, Mineral Production Costs - Analyses and Management, SME, (1999)

[2] N. Dondur, Economic Analysis of the Project, Faculty of Mechanical Engineering, Belgrade (2002), (in Serbian)

[3] G. Mankju, Principles of Economics, Faculty of Economics, Belgrade (2005),(in Serbian).

[4] M. Bugarin, G. Slavković Techno-economic Evaluation, Copper Institute, Bor (2006), (in Serbian).

[5] T. Kuronen: Capital Budgeting in a Capital-Intensive Industry, Helsinki University of Technology, Mat-2.108 Independent research projects in applied mathematics (2007)

[6] M. Bugarin, G. Slavković, M. Maksimović, Valuation of raw mineral for the Coka Marin deposit, Mining Engineering, 2-2010, pp. 23-29

UPUTSTVO AUTORIMA

Časopis RUDARSKI RADOVI izlazi dva puta godišnje i objavljuje naučne, stručne i pregledne radove. Za objavljivanje u časopisu prihvataju se isključivo originalni radovi koji nisu prethodno objavljivani i nisu istovremeno podneti za objavljivanje negde drugde. Radovi se anonimno recenziraju od strane recenzenta posle čega uredništvo donosi odluku o objavljivanju. Rad priložen za objavljivanje treba da bude pripremljen prema dole navedenom uputstvu da bi bio uključen u proceduru recenziranja. Neodgovarajuće pripremljeni rukopisi biće vraćeni autoru na doradu.

Obim i font. Rad treba da je napisan na papiru A4 formata (210x297 mm), margine (leva, desna, gornja i donja) sa po 25 mm, u Microsoft Wordu novije verzije, fontom Times New Roman, veličine 12, sa razmakom 1,5 reda, obostrano poravnat prema levoj i desnoj margini. Preporučuje se da celokupni rukopis ne bude manji od 5 strana i ne veći od 10 strana.

Naslov rada treba da je ispisan velikim slovima, bold, na srpskom i na engleskom jeziku. Ispod naslova rada pišu se imena autora i institucija u kojoj rade. Autor rada zadužen za korespodenciju sa uredništvom mora da navede svoju e-mail adresu za kontakt u fusnoti.

Izvod se nalazi na početku rada i treba biti dužine do 200 reči, da sadrži cilj rada, primenjene metode, glavne rezultate i zaključke. Veličina fonta je 10, italic.

Ključne reči se navode ispod izvoda. Treba da ih bude minimalno 3, a maksimalno 6. Veličina fonta je 10, italic.

Izvod i ključne reči treba da budu date i na engleski jezik.

Osnovni tekst. Radove treba pisati jezgrovito, razumljivim stilom i logičkim redom koji, po pravilu, uključuje uvodni deo s određenjem cilja ili problema rada, opis metodologije, prikaz dobijenih rezultata, kao i diskusiju rezultata sa zaključcima i implikacijama.

Glavni naslovi trebaju biti urađeni sa veličinom fonta 12, bold, sve velika slova i poravnati sa levom marginom.

Podnaslovi se pišu sa veličinom fonta 12, bold, poravnato prema levoj margini, velikim i malim slovima.

Slike i tabele. Svaka ilustracija i tabela moraju biti razumljive i bez čitanja teksta, odnosno, moraju imati redni broj, naslov i legendu (objašnjenje oznaka, šifara, skraćenica i sl.). Tekst se navodi ispod slike, a iznad tabele. Redni brojevi slika i tabela se daju arapskim brojevima.

Reference u tekstu se navode u ugličastim zagradama, na pr. [1,3]. Reference se prilažu na kraju rada na sledeći način:

[1] B.A. Willis, Mineral Procesing Technology, Oxford, Perganom Press, 1979, str. 35. *(za poglavlje u knjizi)*

[2] H. Ernst, *Research Policy*, 30 (2001) 143–157. *(za članak u časopisu)*

[3] www: http://www.vanguard.edu/psychology/apa.pdf *(za web dokument)*

Navođenje neobjavljenih radova nije poželjno, a ukoliko je neophodno treba navesti što potpunije podatke o izvoru.

Zahvalnost se daje po potrebi, na kraju rada, a treba da sadrži ime institucije koja je finansirala rezultate koji se daju u radu, sa nazivom i brojem projekta; ili ukoliko rad potiče iz magistarske teze ili doktorske disertacije, treba dati naziv teze/disertacije, mesto, godinu i fakultet na kojem je odbranjena. Veličina fonta 10, italic.

Radovi se šalju prevashodno elektronskom poštom ili u drugom elektronskom obliku.

Adresa uredništva je: Časopis RUDARSKI RADOVI
Institut za rudarstvo i metalurgiju
Zeleni bulevar 35, 19210 Bor
E-mail: nti@irmbor.co.rs ; milenko.ljubojev@irmbor.co.rs
Telefon: 030/435-164; 030/454-110

Svim autorima se zahvaljujemo na saradnji.

INSTRUCTIONS FOR THE AUTHORS

MINING ENGINEERING Journal is published twice a year and publishes the scientific, technical and review paper works. Only original works, not previously published and not simultaneously submitted for publication elsewhere, are accepted for publication in the journal. The papers are anonymously reviewed by the reviewers after that the editors decided to publish. The submitted work for publication should be prepared according to the instructions below as to be included in the procedure of reviewing. Inadequate prepared manuscripts will be returned to the author for finishing.

Volume and Font size. The work needs to be written on A4 paper (210x297 mm), margins (left, right, upper and bottom) with each 25 mm, in the Microsoft Word later version, font Times New Roman, size 12, with 1.5 line spacing, justified to the left and right margins. It is recommended that the entire manuscript can not be less than 5 pages and not exceed 10 pages.

Title of Work should be written in capital letters, bold, in Serbian and English. Under the title, the names of authors and institutions where they work are written under the title. The author of work, responsible for correspondence with the editorial staff, must provide his/her e-mail address for contact in a footnote.

Abstract is at the beginning of work and should be up to 200 words, include the aim of the work, the applied methods, the main results and conclusions. The font size is 10, italic.

Key words are listed below abstract. They should be minimum 3 and maximum of 6. The font size is 10, italic.

Abstract and Key words should be also given in English language.

Basic text. The papers should be written concisely, in understandable style and logical order that, as a rule, including the introductory section with a definition of the aim or problem, a description of the methodology, presentation of the results as well as a discussion of the results with conclusions and implications.

Main titles should be done with the font size 12, bold, all capital letters and aligned with the left margin.

Subtitles are written with the font size 12, bold, aligned to the left margin, large and small letters.

Figure and Tables. Each figure and table must be understandable without reading the text, i.e., must have a serial number, title and legend (explanation of marks, codes, abbreviations, etc.). The text is stated below the figure and above the table. Serial numbers of figures and tables are given in Arabic numbers.

References in the text are referred to in angle brackets, exp. [1, 3]. References are enclosed at the end in the following way:

[1] Willis B. A., Mineral Procesing Technology, Oxford, Pergamon Press, 1979, pg. 35. (for the chapter in a book)

[2] Ernst H., Research Policy, 30 (2001) 143–157. (for the article in a journal)

[3] www: http://www.vanguard.edu/psychology/apa.pdf (for web document)

Specifying the unpublished works is not desirable and, if it is necessary, as much as possible data on the source should be listed.

Acknowledgement is given where appropriate, at the end of the work and should include the name of institution that funded the given results in the work, with the name and number of project, or if the work is derived from the master theses or doctoral dissertation, it should give the name of thesis / dissertation, place, year and faculty where it was defended. Font size is 10, italic.

The paper works are primarily sent by e-mail or in other electronic form.

Editorial address : Journal MINING ENGINEERING
Mining and Metallurgy Institute
35 Zeleni bulevar, 19210 Bor
E-mail:nti@irmbor.co.rs; milenko.ljubojev@irmbor.co.rs
Telephone: +381 (0) 30/435-164; +381 (0) 30/454-110

We are thankful for all authors on cooperation