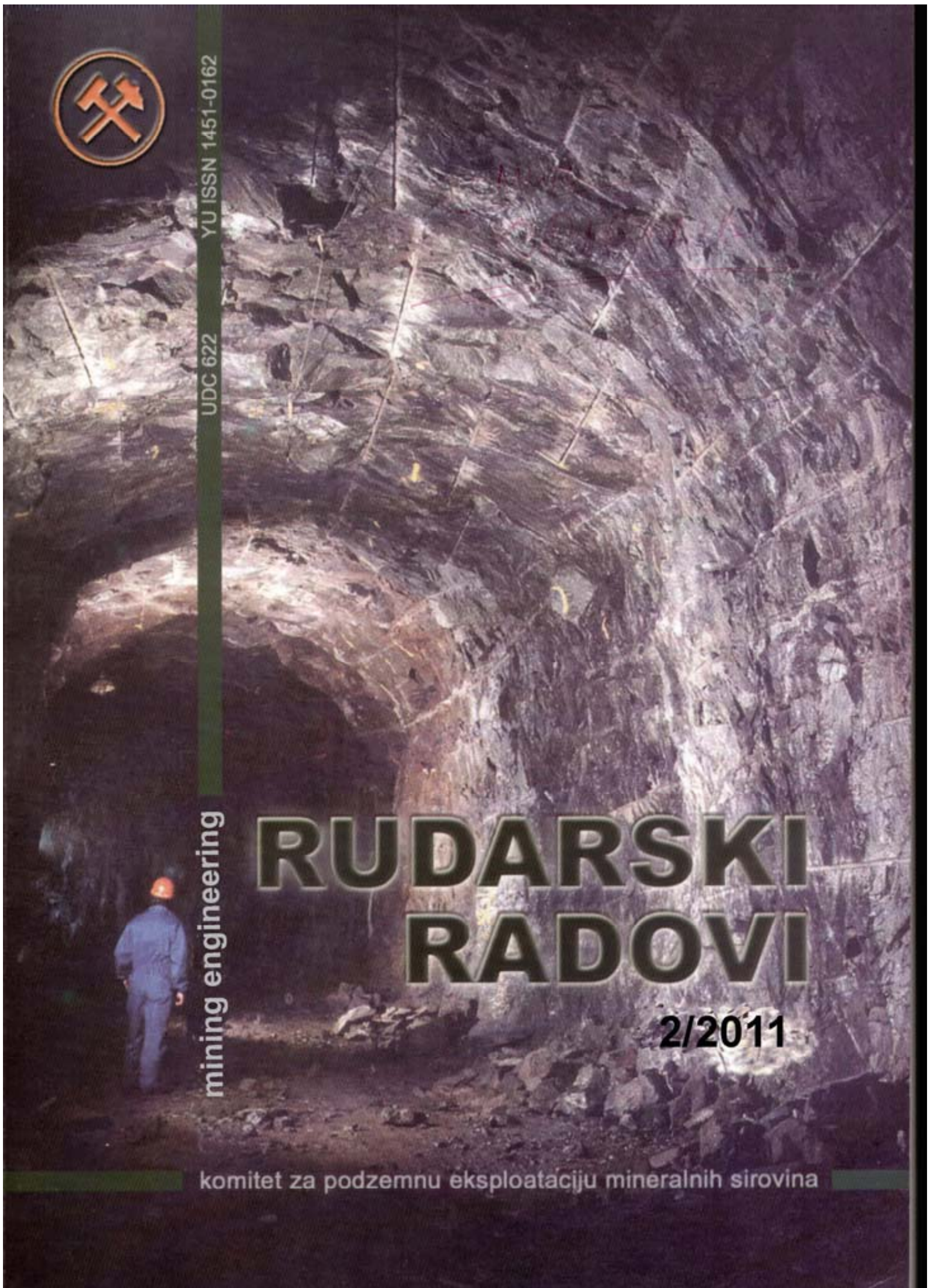
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KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA

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KOMITET ZA PODZEMNU EKSPLOATACIJU MINERALNIH SIROVINA UDK: 622

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

TRIAKSIJALNI CD OGLED NA UZORKU FLOTACIJSKE JALOVINE SA BRANE 3A**

Izvod

Određivanje elemenata unutrašnjeg otpora tla, kohezije i ugla unutrašnjeg trenja, moguće je odrediti pomoću laboratorijskih ogleda, među kojima su najpoznatiji: ogled direktnog smicanja, ogled triaksijalne kompresije i ogled jednoaksijalne čvrstoće na pritisak. Najprecizniji rezultati se dobijaju ogledom triaksijalne kompresije, jer ovaj ogled omogućuje postizanje što približnijih uslova sredine iz koje potiče uzorak koji se ispituje. Zbog toga se mogu očekivati veoma precizni rezultati merenih i traženih mehaničkih osobina materijala.

U ovom radu je opisan ogled i dati su rezultati CD triaksijalne kompresije na uzorku flotacijske jalovine sa brane 3A, ispod koje, jednim svojim delom, prolazi tunel Kriveljske reke. Rezultati su prikazani tabelarno i grafički.

Ključne reči: *kohezija, ugao unutrašnjeg trenja, CD triaksijalni ogled*

UVOD

Drenirani CD opit triaksijalne kompresije se vrši sa konsolidacijom uz dreniranje u toku celog opita, kako za vreme primene svestranog pritiska tako i vertikalnog opterećenja. CD ogled triaksijalne kompresije je opit koji se veoma retko vrši zbog dugog trajanja.

Zbog potreba firme GEOTRON iz Holandije, vršena su ispitivanja flotacijskog materijala sa brane 3, i u daljem tekstu će biti prezentovani rezultati ispitivanja.

ISPITIVANJE SMICANJA U TRIAKSIJALNOM APARATU

Triaksijalnu aparaturu su detaljno opisali Bišop i Henkel (1962.). Uzorak za testiranje je obično cilindar sa odnosom visine i prečnika 2:1. Uzorak je „zapečaćen“ gumenom membranom koja je pričvršćena prstenastom okruglom gumicom za pijedestal i za gornju kapu. Pritisak vode unutar ćelije obezbeđuje horizontalni ukupni napon, dok se vertikalni pritisak na gornjoj kapi ostvaruje preko pritiska fluida i sile nabijača.

* *Institut za rudarstvo i metalurgiju Bor*

** *Ovaj rad je proistekao iz Projekta broj 33021 „Istraživanje i praćenje promena naponsko deformacionog stanja u stenskom masivu“ in-situ“ oko podzemnih prostorija sa izradom modela sa posebnim osvrtom na tunel Kriveljske reke i Jame Bor“, koga finansira Ministarstvo za prosvetu i nauku Republike Srbije*

Glavna prednost opita triaksijalne kompresije u odnosu na opit direktnog smicanja je u homogenijem polju napona, uz mogućnost merenja pornih pritisaka ili promene zapremine u procesu deformisanja.

Tri najčešća oblika testa su:

1. nekonsolidovani nedrenirani triaksijalni opit UU, bez merenja pornog pritiska;
2. konsolidovani nedrenirani opit CU, sa merenjem pornog pritiska i
3. konsolidovani drenirani opit CD, sa merenjem promene zapremine.

REZULTATI ISPITIVANJA

Materijal, koji je korišćen za triaksijalno CD ispitivanje, je uzorkovan sa flotacijske brane 3A. To je poremećeni uzorak iz bušotine D3-B2 (1,50-5,40). Karakteristike materijala su sledeće:

- zapreminska masa 1,645 g/cm^3
- specifična masa 25.70 g/cm^3
- vlaga u prirodnom stanju 17,62%.

Sl. 1. *Triaksijalni aparat*

Za kompletan opit se najčešće ispituju najmanje 3 uzorka sa različitim veličinama svestranih (bočnih) pritisaka σ_3. U suštini, prvi uzorak se izlaže najmanjim usvojenim naponima, drugi sa najmanje dvostrukom veličinom pritiska primenjenim na prvom uzorku, a treći sa najmanje dvostrukom veličinom pritiska upotrebljenim za drugi uzorak.

Sl. 2. *Uzorak nakon loma*

U daljem tekstu su prikazani rezultati merenja tokom CD testa na uzorku D3-B2 (1,50-5,40) sa vrednostima za koheziju C i ugao unutrašnjeg trenja φ.

GROUP OF CD TESTS 1/4

Customer data

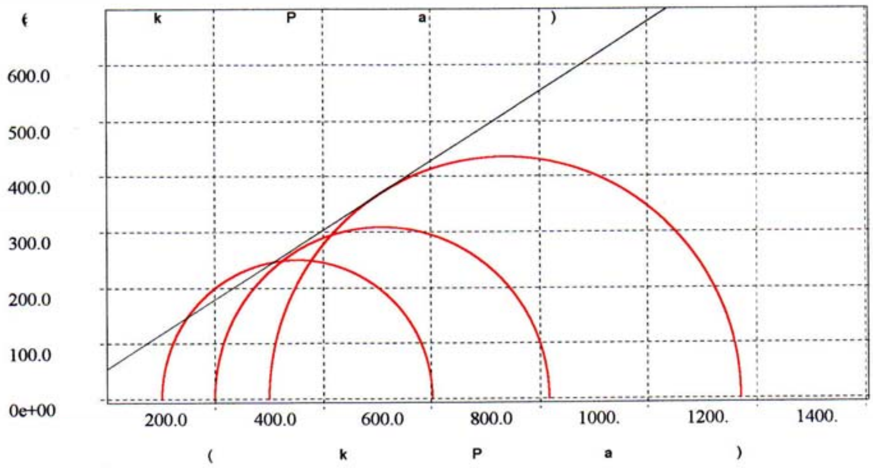
Customer	: Geotron
Address	: Grey, slightly silty medium fine sand
Site	: Dam 3
Boring	: D3-B2
Sample	: 1
Depth	: 1.5-5.4

Specimen characteristics

Sample	H_0 mm	A_0 cm2	γ_n g/cm3	γ_d g/cm3	W_0 %	W_f %	S_0 %	S_f %
1	101.00	20.427	1.645	1.333	23.410	18.446	61.924	73.245
2	101.00	20.427	1.636	1.325	23.496	18.015	61.402	66.570
3	100.00	20.427	1.643	1.341	22.462	18.855	60.154	68.181

Characteristics of failure stage

Sample	σ'_{1c} kPa	σ'_{3c} kPa	BP kPa	ε %	$\sigma_1-\sigma_3$ kPa	σ_1'/σ_3'		
1	200.00	200.00	100.00	12.380	501.92	5.986		
2	300.00	300.00	100.00	12.973	617.01	4.085		
3	400.00	400.00	100.00	12.108	870.08	3.884		



Results

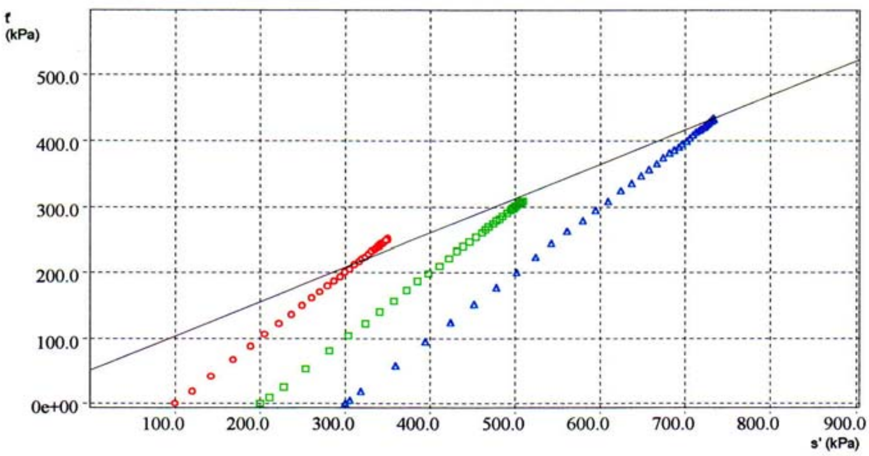
Φ'	: 31.89 Deg.
c'	: 55.30 kPa

Sl. 3. *Izgled Morovih krugova, vrednosti kohezije C i ugla unutrašnjeg trenja φ*

GROUP OF CD TESTS 2/4

Customer data

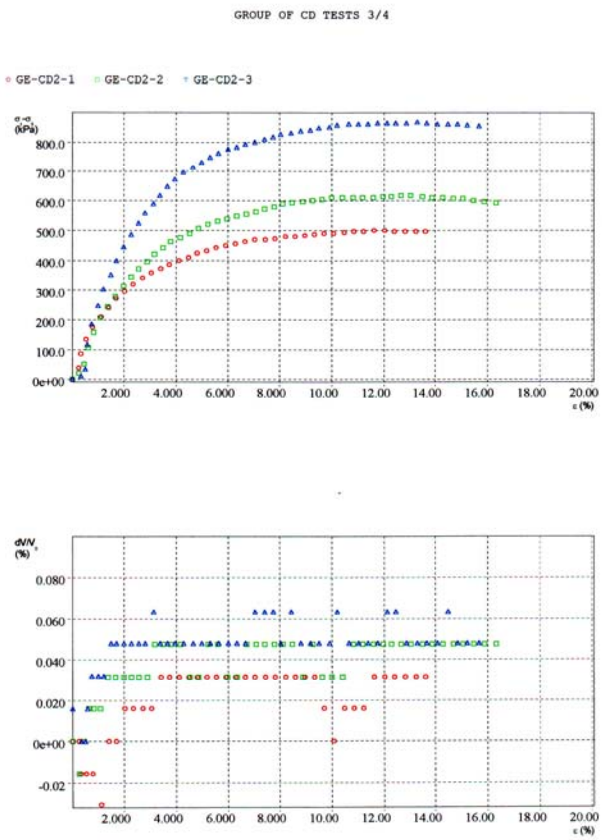
Customer	:	Geotron
Address	:	Grey, slightly silty medium fine sand
Site	:	Dam 3
Boring	:	D3-B2
Sample	:	1
Depth	:	1.5-5.4



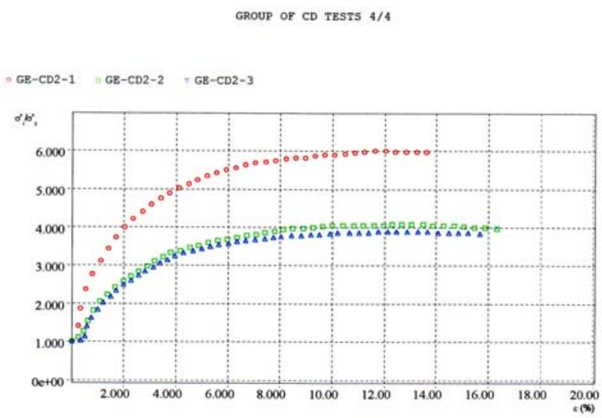
Results

Φ'	:	31.53 Deg.
c'	:	59.08 kPa

Sl. 4. *Odnos s' i t'*



Sl. 5. *Prvi grafik: Odnos ε i $\sigma_1 - \sigma_3$;*
Drugi grafik: Odnos ε i d_V/V_0



Sl. 6. *Odnos ε i σ_1/σ_3*

ZAKLJUČAK

CD triaksijalni opit, sa merenjem promene zapremine tokom smicanja, se izvodi na sličan način kao i konsolidovani nedrenirani opit, ali tokom smicanja protivpritisak ostaje povezan sa uzorkom koji je opterećen dovoljno sporo da bi se izbegao porast pornog pritiska. Koeficijent konsolidacije tla je proizašao iz merenja promene zapremine koja je nastala tokom faze konsolidacije. Gibson i Henkel (1954) su pronašli da je prosečan ugao konsolidacije u fazi loma povezan sa vremenom od početka testa.

Dakle, za fazu smicanja ovog testa može se reći da traje 7 do 15 puta duže od nedreniranog testa sa merenjem pornog pritiska. Kada se smicanje završi, rezultati se prikazuju u vidu grafika količnika glavnih napona i promene zapremine u funkciji istezanja, a Morovi krugovi iz faze loma se crtaju da bi se dobila envelopa definisana parametrima C_d i φ_d.

Ne može se očekivati da parametri efektivne čvrstoće definisani ovim triaksijalnim testom budu isti kao i oni dobijeni nedreniranim testom, obzirom da promena zapremine u fazi loma utiče na izvršeni rad u korist ili protiv ćelijskog pritiska. U praksi rezultat ugla unutrašnjeg trenja za kohezivna tla se normalno razlikuju za 1-2°, dok se kohezija razlikuje za 5 kN/m^2. Rezultati za peskove mogu varirati značajno.

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

TRIAXIAL CD TEST ON THE FLOTATION TAILING DUMP SAMPLE FROM THE DAM 3A**

Abstract

Determing the elements of the internal resistance of the soil, cohesion and internal angle of friction, is possible to determine by using laboratory experiments such as: direct shear test, triaxial test and mono-axial compressive strength. The most precision results can be got from the triaxial test, because it provides conditions for the tested sample, which are very similar to the conditions on the field, from which sample arrives. That is the reason why precision results of measurements and calculations of the mechanical properties can be expected.

In this paper is described CD triaxial test as well as the results that are got from it. This test was performed on the flotation tailing sample from the Dam 3. It has to be mentioned that a part of the Krivelj River tunnel runs under the Dam 3. All results are shown by graphs and tables.

Key words: *cohesion, internal angle of friction, CD triaxial test*

INTRODUCTION

Drained CD triaxial compression test is performed with consolidation and drainage during whole experiment, both, during the implementation of compressive pressure and vertical load. CD test is rarely performed due its long duration.

This CD test was performed on samples from the flotation tailing, Dam 3, on the request of GEOTRON company from the Netherlands. These test results will be presented in further text.

TESTING IN TRIAXIAL APPARATUS

The triaxial apparatus has been described in great deal by Bishop and Henkel (1962). The test speciment is normally a cylinder with an aspect ratio of two, which is sealed on its sides by a rubber membrane attached by rubber „O“ rings to a base pedestal and top cap. Water pressure inside the cell provides the horisontal principal total stresses, while the vertical pressure at the top cap is produced by the cell fluid pressure and the ram force.

[*] *Mining and Metallurgy Institute Bor*
[**] *This paper is produced from the project no. 33021 “Researching and monitoring changes in stress-deformation condition of rock massif “in-situ” around undergraund facilities with development of model with special emphasis on Krivelj river tunnel and Bor pit”, which is funded by means of the Ministry of Education and Science of the Republic of Serbia*

The main advantage of triaxial compression test over shear test is homogenious stress field, with possibility of measuring pore pressure or volume changing during process of deformation.

The three most common forms of tests are:

1. The unconsolidated undrained triaxial compression test UU, without pore water pressure measurement,
2. The consolidated undrained triaxial compression test CU, with pore water pressure measurement and,
3. The consolidated drained triaxial compression test CD, with volume change measurement.

TEST RESULTS

Material, whose been used for CD triaxial test, was sampled from the flotation tailing dam 3A. It was desturbed sample from the borehole D3-B2 (1,50-5,40). Characteristics of the material are:

- bulk density	1,645 g/cm3
- specific density	2,690 g/cm3
- moisture content	17,62%.

Figure 1. *Triaxial apparatus*

For complete test it is necessary to test at least three speciments with different effective pressures σ_3. Basicly, first sample is exposed to the minimum stress, second sample to the doubled value of the first stress and the third sample is exposed to the stress that is double value of the second stress.

Figure 2. *Sample after a failure stage*

Test results are shown in following text, during CD test on the sample D3-B2 (1,50-5,40) with values of the cohesion C and internal angle of friction φ.

GROUP OF CD TESTS 1/4

Customer data

Customer	: Geotron
Address	: Grey, slightly silty medium fine sand
Site	: Dam 3
Boring	: D3-B2
Sample	: 1
Depth	: 1.5-5.4

Specimen characteristics

Sample	H_0 mm	A_0 cm2	γ_b g/cm3	γ_d g/cm3	W_0 %	W_f %	S_0 %	S_f %
1	101.00	20.427	1.645	1.333	23.410	18.446	61.924	73.245
2	101.00	20.427	1.636	1.325	23.496	18.015	61.402	66.570
3	100.00	20.427	1.643	1.341	22.462	18.855	60.154	68.181

Characteristics of failure stage

Sample	σ'_{1c} kPa	σ'_{3c} kPa	BP kPa	ε %	$\sigma_1-\sigma_3$ kPa	σ_1'/σ_3'		
1	200.00	200.00	100.00	12.380	501.92	5.986		
2	300.00	300.00	100.00	12.973	617.01	4.085		
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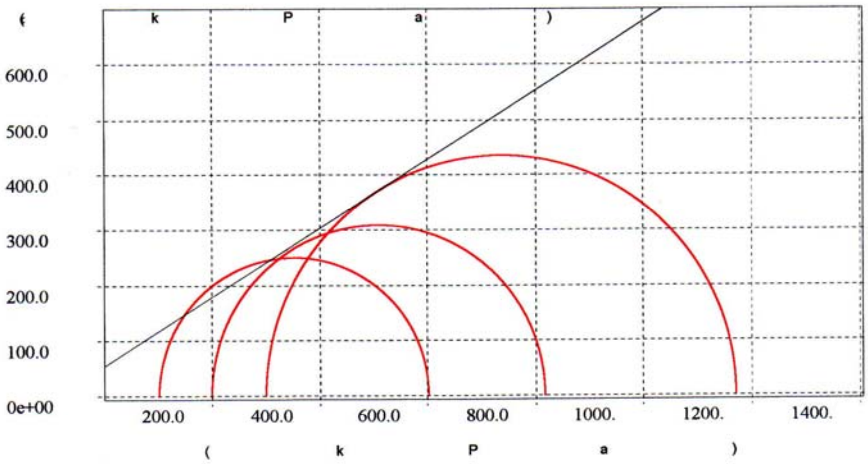


Results

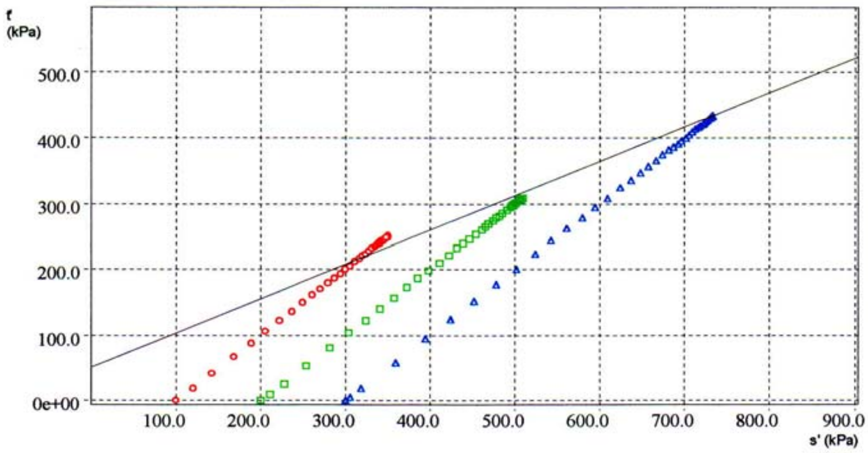
Φ'	: 31.89 Deg.
c'	: 55.30 kPa

Figure 3. *Mohr circles, values of the cohesion C and internal angle of friction φ*

GROUP OF CD TESTS 2/4

Customer data

Customer	:	Geotron
Address	:	Grey, slightly silty medium fine sand
Site	:	Dam 3
Boring	:	D3-B2
Sample	:	1
Depth	:	1.5-5.4



Results

Φ'	:	31.53 Deg.
c'	:	59.08 kPa

Figure 4. *Ratio between s' and t'*

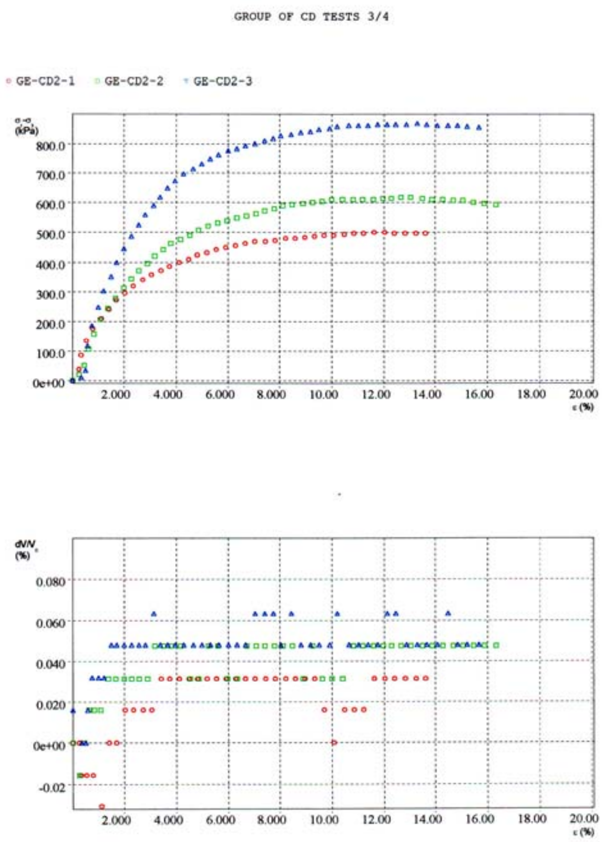


Figure 5. *First graph: Ratio between ε and $\sigma_1 - \sigma_3$;*
Second graph: Ratio between ε and d_V/V_0

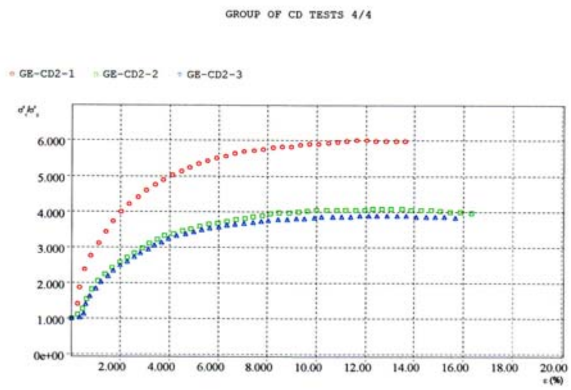


Figure 6. *Ratio between ε i σ_1/σ_3*

CONCLUSION

The CD triaxial compression test, with volume change measurement during shear is carried out in a similar sequence to the consolidated undrained test, but during shear the back pressure remains connected to the speciment which is loaded sufficiently slowly to avoid the development of excess pore pressure. The coefficient of consolidation of the soil is derived from the volume change measurements made during the consolidation stage. Gibson and Henkel (1954) found that the average degree of consolidation at failure is related to the time from the start of the test.

Thus the shear stage of this test can be expected to take between 7 and 15 times longer than that of an undrained test with pore pressure measurement. Once shearing is complete, the results are presented as graphs of principal stress difference and volume change as a function of strain, and the failure Mohr circles are plotted to give the drained failure envelope defined by parameters C_d and φ_d.

The effective strength parameters defined by drained triaxial testing should not be expected to be precisely the same as those for an undrained test, since volume changes occuring at failure involve work being done by or against the cell pressure. In practice the resulting angles of friction for chohesive soils are normaly within 1-2°, and the cohesion intercepts are within 5 kN/m^2. The results of tests on sands can vary very greatly.

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Živorad Milićević, Igor Svrkota*, Dejan Petrović**

RAZMATRANJE PROBLEMA OTVARANJA LEŽIŠTA FOSFORITSKOG PEŠČARA „LISINA“ KOD BOSILEGRADA

Izvod

Poslednje dve godine ležište fosforitskog peščara „Lisina“ kod Bosilegrada u Jugoistočnoj Srbiji ponovo je predmet pažnje i istraživanja mogućnosti ekonomične eksploatacije, budući da se radi o mineralnoj sirovini koja se uvozi, a koja je neophodna u procesu proizvodnje veštačkih mineralnih djubriva. Osnovni problem je u rudi sa niskim sadržajem korisne materije P_2O_5 zbog čega se ekonomična eksploatacija može izvoditi samo i slučaju visoke cene ove sirovine na svetskom tržištu.

U istraživačkoj studiji razmatrana je i mogućnost podzemne eksploatacije ovog ležišta, a u radu se govori o problemu izbora načina otvaranja i lokacije objekata otvaranja, što je uslovljeno i nepovoljnom konfiguracijom terena.

Ključne reči: *ležište fosforita, podzemna eksploatacija, otvaranje ležišta, lokacija objekata otvaranja.*

1. UVOD

Šezdesetih godina prošlog veka u neposrednoj okolini Bosilegrada u jugoistočnoj Srbiji, pronadjeno je i delimično istraženo ležište fosforitskog peščara „Lisina“. Ležište se javlja u vidu sloja moćnosti 4 – 20 m, koji izdanjuje u kosinama doline Božičke reke na dužini skoro 6 km, (sl. 1), /1/.

U nekoliko navrata sedamdesetih i devedesetih godina prošlog veka radjene su studije istraživanja mogućnosti ekonomične eksploatacije ovog ležišta sa ciljem da se ispita i utvrdi mogućnost racionalne eksploatacije ovog ležišta. U prvim dvema, sedamdesetih godina, kao osnovna, razmatrana je mogućnost podzemne ksploatacije, a u pretposlednjoj studiji, krajem devedesetih godina, razmatrana je mogućnost površinske eksploatacije u delu ležišta na lokalitetu Panjeviće, (sl. 1). U ovoj studiji je razmatrana mogućnost zahvata samo dela ležišta sa sadržajem P_2O_5 iznad 14 %, što je svakako neracionalno sa stanovišta količine zahvaćenih rudnih rezervi.

U trenutku bitnog poboljšanja cene fosfata na svetskom tržištu, naša poznata firma iz oblasti agrarnog biznisa „Victorija Group“, preuzela je inicijativu da se ponovo razmotri mogućnost ekonomične eksploatacije ovog ležišta, pa je radjena studija /2/, u kojoj je, takodje, razmatrana mogućnost podzemne, ali i površinske

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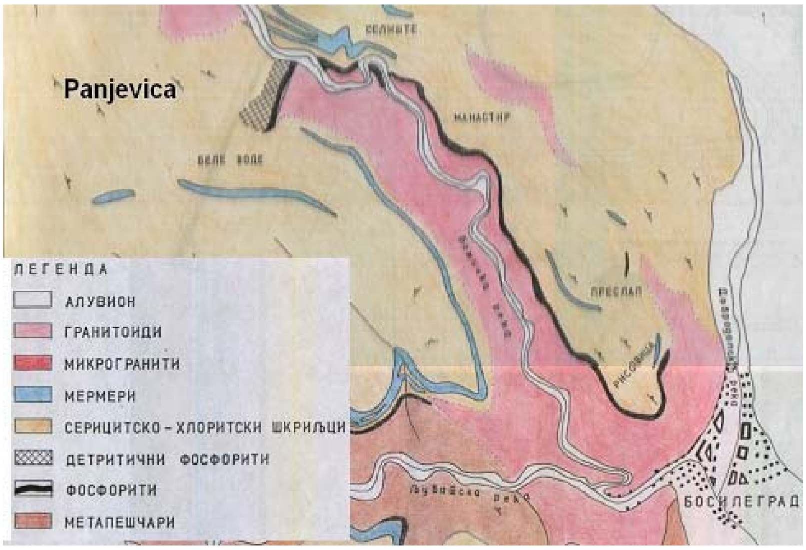
eksploatacije. Deo rezultata tih istraživanja prikazan je u ovom radu, a verovatno će biti i prezentiran u više budućih radova.

2. OSNOVNI PODACI O LEŽIŠTU

Više podataka o geološkom istraživanju i karakteristikama ovog ležišta dato je u radu /1/, a ovom prilikom se daju samo osnovni, radi lakšeg sagledavanja problematike. Mineralna sirovina P_2O_5 javlja se u sedimentnim stenama - ordovicijumskim peščarima, koji u bliskoj okolini Bosilegrada čine deo metamorfisanih stena granitoida i škriljaca, tačnije javljaju se izmedju podinskih granitoida i povlatnih škriljaca. Navedene serije metamorfisanih stena na znatnoj visini su presečene dolinom Božiške reke, koja je usekla uzanu dolinu dubine 300 - 500 m u odnosu na vrhove bočnih strana. Najveći deo ležišta, kao što se sa slike vidi javlja se u vidu izdanka u levom boku doline, dok se samo mali deo javlja na desnoj obali u vidu plitkog izdanka (lokalitet Panjevica). Pojava fosforitskih peščara ima na još nekoliko lokacija, ali su one od manjeg značaja i nisu predmet razmatranja.



Sl. 1. *Geološka karta područja ležišta fosforita Lisina kod Bosilegrada – JI Srbija, /1/*

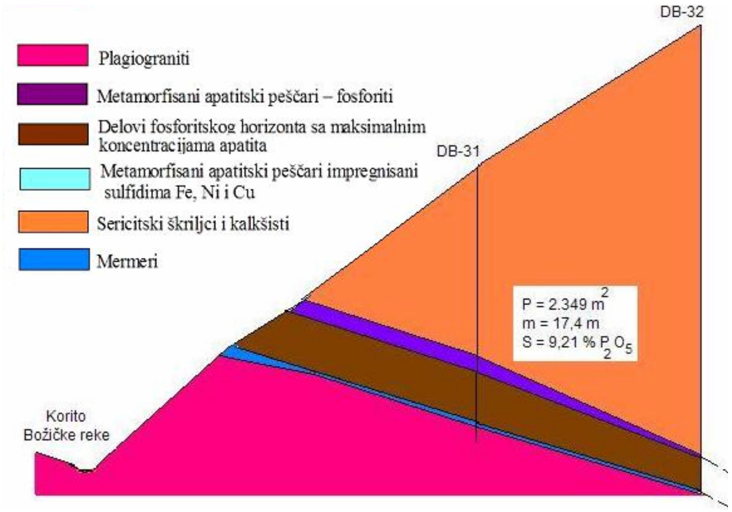
Sloj fosforitskog peščara uglavnom pravilno zaleže na relativno velikoj dužini od blizu 6 km, ali je moćnost promenljiva u širokim granicama (2 do preko 20 m). Mestimično se javljaju tektonski poremećaju, pa je u jednom delu sloj znatno manjeg generalnog pada od prosečnog koji se uzima da iznosi 22^o.

Kao što se s sl. 2 vidi krovinske stene su predstavljene sericitsko-hloritskim škriljcima, dok se u podini javljaju granitoidne stene. Koristan mineral apatit se javlja u slabo metamorfisanim peščarima, u kojima se u gornjoj zoni javljaju i sulfidni minerali gvoždja, nikla i bakra. Inače, mineralni sastav peščara predstavljen je mineralima: kvarc, sericit, apatit, karbonati, biotit sa hloritom, sitnozrni epidot, turmalin, neprovidni minerali-organska materija, limonit i hematit.

Od najvećeg začaja je sadržaj korisne materije P_2O_5, koji se kreće, takodje, u širokim granicama od 1,5 – 19 %. Kao rezultat dosadašnjih istraživanja u više navrata su overavane rudne rezerve, a relevantni podaci o njima, (1975.g.), su:

A kategorija
4.169.256 t rude sa 11,87% P_2O_5
B kategorija
42.353.450 t rude sa 9,54% P_2O_5
C_1 kategorija
1.861.771 t rude sa 9,99% P_2O_5
Ukupno:
384.477 t rude sa 9,83% P_2O_5

Medjutim, ako se ima u vidu da su uvozni koncentrati fosfata sa znatno većim sadržajem P_2O_5, (i preko 30 %), onda je jasno da je ova okolnost od osnovnog uticaja na donošenje konačne odluke o pristupanju eksploatacionim radovima na ovom ležištu. Postoji još jedan značajan problem, a on je vezan za vrlo nepovoljan lokalitet na kome se ležište javlja, udaljeno od raspoloživih preradjivačkih kapaciteta (Prahovo, Šabac i dr.).



Sl. 2. *Karakterističan poprečni profil, (br. VII), kroz ležište fosforita „Lisina*

3. IZBOR NAČINA OTVARANJA I LOKALITETA ZA IZGRADNJU POVRŠINSKIH OBJEKATA I PROSTORIJA OTVARANJA LEŽIŠTA

U ranijim istraživanjima prednost je davana podzemnom načinu eksploatacije, ali je u najnovijoj istraživačkoj studiji veća pažnja poklonjena površinskom načinu. Bez obzira na to, problem otvaranja ležišta za podzemnu eksploataciju je i dalje aktuelan, budući da se površinskim otkopavanjem može zahvatiti samo manji deo ležišta na izdanku, a naš je stav da se i to mora pažljivo razmotriti s obzirom na izuzetno velike ekološke posledice takvog načina otkopavanja.

Izbor načina otvaranja, a zatim i lokaliteta površinskih rudničkih objekata, u konkretnom slučaju, u velikoj meri zavisi i od sledećih uticajnih faktora:

- prostornog položaja ležišta i pojave njegovog izdanka na velikoj dužini osine leve strane doline Božičke reke,

- konfiguracije terena, koja je vrlo nepovoljna sa stanovišta mogućnosti lociranja površinskih objekata rudnika,
- dubine zaleganja, koja je relativno mala u odnosu na nivo korita Božičke reke, mada treba imati u vidu da ležište, zbog nepovoljne konfiguracije terena, nije istraženo i utvrdjeno do konačne dubine,
- načina otvaranja uslovljenog mogućim rešenjima izvoza rude iz jame,
- moguće lokacije za razmeštaj objekata za preradu i obogaćivanje mineralne sirovine,
- mogućeg izbora odgovarajuće lokacije za odlaganje flotacijske jalovine,
- perspektivnog smera daljeg transporta dobijenog koncentrata, i dr.

Problem izbora načina otvaranja i izbora lokacije za površinske objekte rudnika jasno se uočava na slici br. 3. Vidi se da je konfiguracija terena predstavljena dosta uskom dolinom Božičke reke, pa se na najvećoj dužini ležišta ne javljaju pogodni širi prostori za lociranje površinskih objekata rudnika. Nešto povoljnija konfiguracija se javlja u krajnjem severozapadnom delu ležišta na lokalitetu Panjevica, gde je opravdano moguće površinsko otkopavanje dela ležišta koji se javlja na desnoj obali Božičke reke.

To je bio dovoljno ubedljiv razlog što je u okviru istraživačkih studija, radjenih početkom sedamdesetih godina prošlog veka, lokacija rudničkog kruga izabrana na krajnjoj južnoj strani ležišta, neposredno u blizini naselja Bosilegrad (severoistočno od njega). Na tom delu postojao je i pogodan prostor za razmeštaj flotacijskog postronjenja i njegovog jalovišta. Medjutim, u toku minulih godina na tom prostoru su izgradjeni brojni stambeni objekti, tako da ta lokacija sada nije aktuelna.

Sl. 3. *Prikaz površine terena sa dolinom Božičke reke u kojoj se javlja izdanak fosforitskog peščara:*

1- prva razmatrana lokacija lociranja rudničkih objekata i flotacije, usvojena u studiji 1975 godine;
2- lokacija usvojena na početku istraživanja u 2008. godini;
3- usvojena lokacija u prethodnoj studiji;
4- perspektivna lokacija za slučaj lociranja flotacijskog postrojenja na lokalitetu Panjevica.

Na inicijativu kompanije Victorija Group u toku 2008. godine vršena su nova istraživanja mogućnosti eksploatacije fosforita iz ležišta Lisina. U okviru razmatranja problema otvaranja ležišta uzete su u obzir sledeće važne okolnosti:

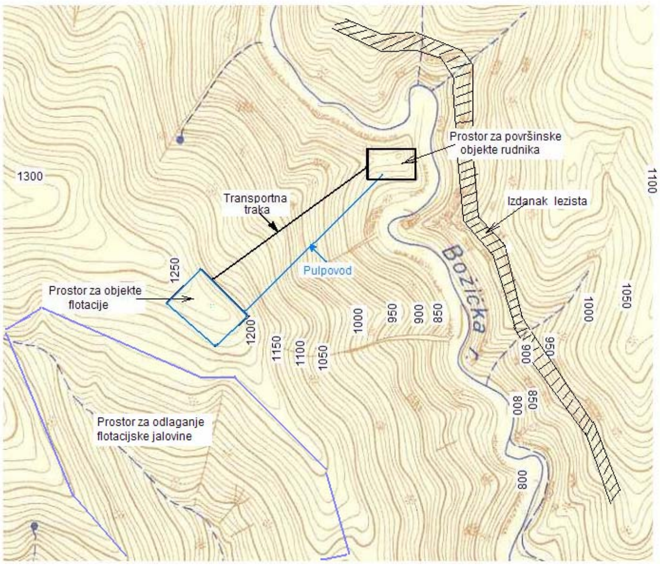
1. Način zaleganja ležišta je takav da sugerira otvaranje kosim jamskim prostorijama, kada prevashodno u obzir dolazi mogućnost izvoza iskopina transportnim trakama, kao najracionalnijem vidu. Druga mogućnost je izvoz jamskim kamionima, ali ta okolnost u prvi mah nije uzeta u obzir, budući da je prednost data železničkom transportu na horizontima, što je uslovljeno, pre svega, velikom dužinom ležišta, odnosno i velikom dužinom transporta na horizontima;

2. Velika dužina jamskog transporta na horizontima nameće najpovoljnije otvaranje ležišta približno u sredini njegove dužine;

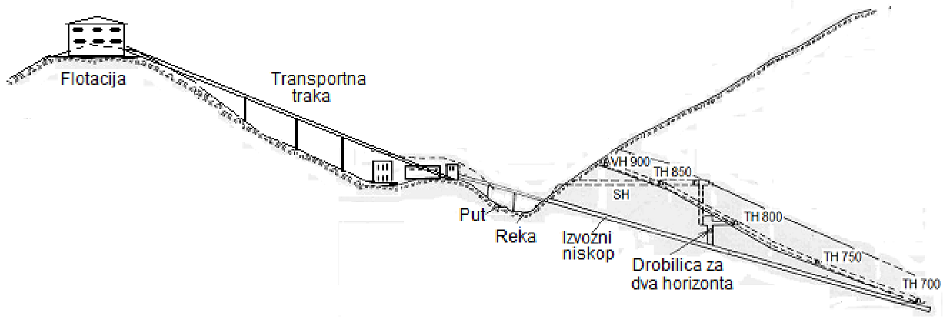
3. Prvobitni predlog lokacije flotacijskog postrojenja bio je na prevoju desne strane doline Božičke reke, budući da je na tom prostoru utvrdjeno i postojanje pogodne lokacije flotacijskog jalovišta, (sl 4).

Na osnovu navedenog predložena je lokacija rudničkog kruga na desnoj obali Božičke reke nasuprot lokalitetu Manastir (na prilogu br 3 označeno je brojem 2), a položaj rudničkih objekata, postrojenja za preradu i flotacijskog jalovišta prikazani su na slikama 4 i 5.

Osnovni nedostatak predloženog načina otvaranja je u nepovoljnom položaju kako objekata rudničkog kruga, tako i postrojenja za preradu. Spoljni rudnički objekti gradili bi se na skučenom prostoru na 30 m iznad nivoa puta što bi zahtevalo izradu pristupnog puta i značajne radove na pripremi terena za gradnju tih objekata. Lokacija flotacije na relativno velikoj visini iznad rudničkih objekata, bez obzira na povoljniju konfiguraciju mikrolokacije, nepovoljna je zbog trajne potrebe izvoza rude do flotacije, pri čemu se visina ivoza povećava za još 350 m. Isto tako, izrada pristupnog puta od naselja Donja Lisina do flotacije dodatno povećava investiciona ulaganja i kasnije transportne troškove na dopremi materijala i odvozu koncentrata.



Sl. 4. *Prikaz lokacija rudničkog kruga, flotacije i flotacijskog jalovišta po prvom predlogu u studiji 2008. godine*



Sl. 5. *Predloženi način otvaranja ležišta fosforita Lisina*

U toku razmatranja ovih problema, stekao se utisak da investitor nije ozbiljno razmatrao i drugačije rešenje odlaganja flotacijske jalovine, čija lokacija je presudno uticala na izbor lokacije flotacijskog postrojenja, a posredno i lokacije spoljnih rudarskih objekata. Naime, u delu studije, koji se odnosio na način otkopavanja ležišta predloženo je da se najveći deo flotacijske jalovine odlaže u otkopnim komorama nakon završetka otkopavanja u pojedinim otkopnim blokovima. Takvo rešenje omogućilo bi da se samo mali deo flotacijske jalovine i to u prvim godinama eksploatacije ležišta, (do završetka otkopavanja u prvim otkopnim blokovima), odlaže na spoljno flotacijsko jalovište, a da se kasnije u celom veku eksploatacije ležišta, to odlaganje vrši u završenim otkopima. Ne treba posebno potencirati koliko bi to doprinelo ekološkoj zaštiti površine terena oko rudnika.

U slučaju usvajanja ovog predloga, flotacijsko postrojenje bi se moglo locirati u reonu Panjevice gde se javlja pogodan prostor sa manje izraženim reljefom. Ova lokacija bi imala nekoliko veoma značajnih pogodnosti, kao što su:

- objekti bi se gradili u neposrednoj blizini postojećeg puta što bi bitno olakšalo dopremu materijala za gradnju, a kasnije i dopremu repromaterijala za rad flotacije,
- ova okacija je na krajnjem severnom delu ležišta pogodnja je i zbog toga što bi se kasniji transport koncentata vršio u istom smeru, tj. ka severu prema Vlasinskom jezeru i dalje ka lokacijama industrije za proizvodnju veštačkih djubriva,
- lokcija flotacije u neposrednoj dolini Božičke reke ne uslovljava povećanje visine izvoza, što je jedna od važnih mana prethodne lokacije.

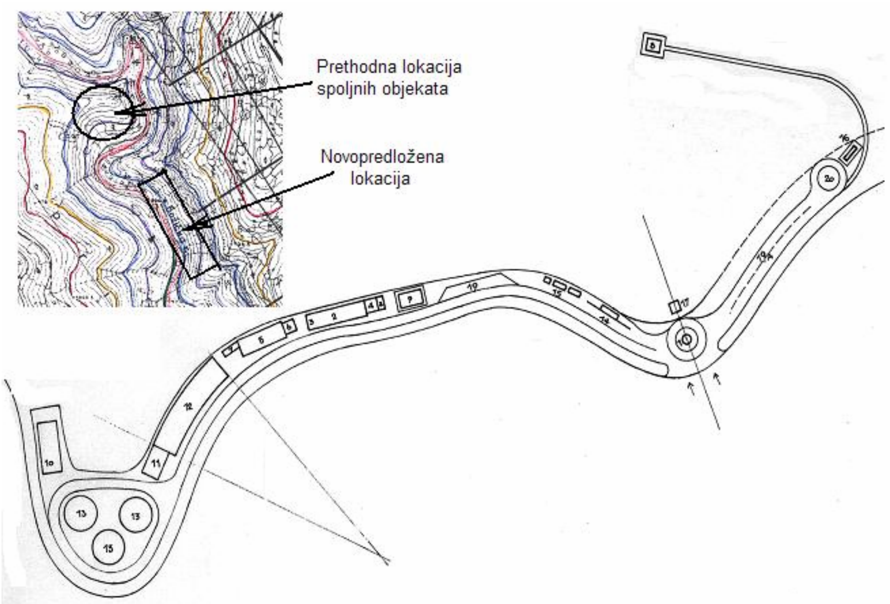
Kao nedostatak ove lokacije mogla bi se navesti blizina rečnog toka, pa bi to zahtevalo njegovu odgovarajuću regulaciju i zaštitu objekata od mogućeg plavljenja u slučaju velikih voda.

U slučaju da se usvoji lokacija lotacije u reonu Panjevice, javljaju se dve mogućnosti za izbor lokacije spoljnih rudarskih objekata:

- da se lokacija zadrži na približno istom prostoru, tj približno u sredini dužine prostiranja ležišta, što je, kao što je napomenuto, najpovoljnije sa stanovišta smanjenja dužine jamskog transporta;
- da se objekti otvaranja ležišta i spoljni gradjevinski objekti rudnika lociraju u blizini flotacije, što bi uslovilo minimalne dužine površinskog transporta, ali bi značajno povećalo dužinu transporta u jami, budući da

se takva lokacija nalazi na jednoj strani ležišta, čime se umesto centralnog usvaja dijagonalan način otvaranja.

U studiji je zadržan centralni način otvaranja, ali je predložena lokacije spoljnih objekata neposredno uz korito reka, što bi uslovilo i njihov spacifičan razmeštaj, prikazan na slici 6.



Sl. 6. *Raspored spoljnih objekata rudnika na novopredloženoj lokaciji rudničkog kruga*

4. ZAKLJUČAK

Ležište fosforita „Lisina“ bez sumnje će i nadalje biti predmet ozbiljnih razmatranja, budući da se radi o mineralnoj sirovini potrebnoj za proizvodnju veštačkih djubriva neophodnih za razvoj poljoprivrede. U skladu sa tim, razmatraće se svi aktuelni problemi uslovljeni karakterom ležišta i njegovom nepovoljnom lokacijom, kako sa stanovišta konfiguracije terena, tako i sa stanovišta značajne udaljenosti od mogućih korisnika ove mineralne sirovine.

Problem otvaranja ležišta i lokacije objekata otvaranja, odnosno i spoljnih rudarskih objekata je jedan od značajnijih. On je, kao što je navedeno, neposredno uslovljen i drugim rešenjima, a pre svega izborom lokacije za izgradnju flotacijskog postrojenja.

Izradjenom studijom predloženo je, kao najpovoljnije, centralno otvaranje ležišta, sa lokacijom flotacije u dolini Božičke reke, a kao veoma važno rešenje, predloženo je odlaganje flotacijske jalovine u prazne otkopne prostore, čime bi se omogućila maksimalna ekološka zaštita površine, ali i značajni ekonomski efekti na smanjenju troškova pripreme terena za flotacijsko odlagalište.

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CONSIDERATIONS THE PROBLEMS OF MINE OPENING IN THE PHOSPHORITE SANDSTONE DEPOSIT „LISINA" NEAR BOSILEGRAD

Abstract

In the last couple of years, the phosphorite sandstone deposit "Lisina" near Bosilegrad in southeast Serbia, has been reconsidered in order of its activation. The deposit became interesting again due to a fact that phosphorite sandstone is imported in Serbia and it is important component of mineral composts. But, content of P_2O_5 in the ore is very low, which means that there are serious economical challenges.

In the research study undertaken at the Technical Faculty in Bor, the analyses included both surface and underground mining. This paper shows considerations related to the opening the underground mine, location of main construction objects on e surface and their layout at unfavorable ground configuration.

Key words: *phosphorite deposit, underground mining, mine opening, location of objects.*

1. INTRODUCTION

Phosphorite sandstone deposit Lisina near Bosliegrad in Eastern Serbia was prospected and partly explored during sixties of the last century. The deposit occurs as a seam, 4 to 20 m thick, almost 6 km long, and reaches the surface at the slopes of the Bozicka River valley (Figure 1), [1].

Several studies were done during 70's and 90's; in order to analyze wheather the economicaly justified extraction is possible. At first, the underground mining was the only option, but later, the surface mining was introduced as the possibility at Panjevice location (Figure 1). However, this option included only deposit parts with over 14% of P_2O_5, what means that a huge part of deposit would be unused.

After recent increase of phosphate prices at the World market, the agricultural company Victoria Group took an initiative to reconsider the possibilities for economic production from this deposit. Study [2] was a part of these considerations and it included variants with underground and surface mining. Part of this study is shown in this paper.

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2. BASIC DATA ON THE DEPOSIT

More data on geological explorations and properties of the deposit were given in [1], while in this paper only basic data shall be given. P_2O_5 occurs in sedimentary rock – ordovicium sandstone, which is a part of metamorphic rocks of granites and shale near Bosilegrad. These series of metamorphic rocks were cut by the valley of the Boziska River, which has 300 to 500 m deep narrow valley. Most of the deposit, as it is shown in Figure 1, occurs on the left side of the valley, while only small part occurs on the right bank (Panjevica locality). There are a faw more occurrences of phosphorite sandstone, but they are less significant and were left from considerations.

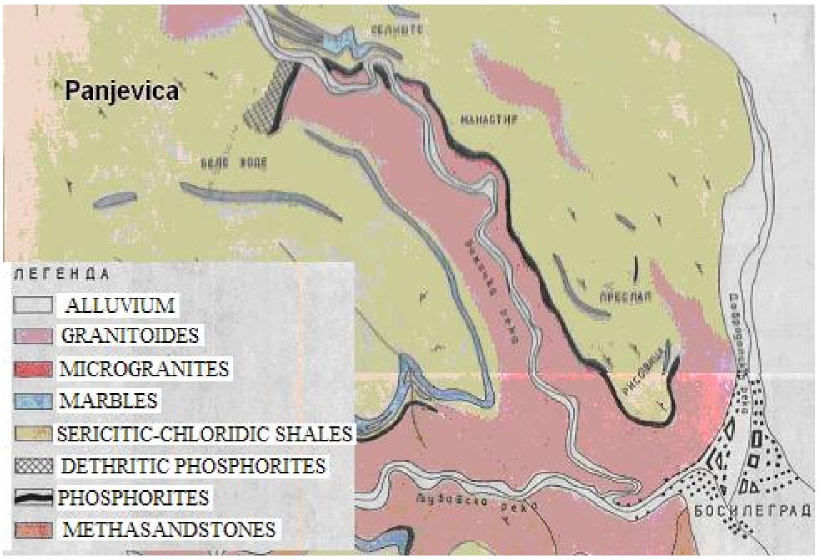

Figure 1. *Geological map of the area of phosphorite sandstone deposit Lisina near Bosilegrad, Southeast Serbia*

Phosphorite sandstone seam is mainly regular along 6 km long strike, but thickness varies significantly (from 2 to 20 m). Tectonic disturbances occur occasionally, which causes changes in the seam dip in these areas. Average seam dip is 22^{o}.

As shown in Figure 2, the hanging wall consists of sericite-chlorite shales, while footwall consists of granithoid rock. Ore-bearing mineral apatite lies in weakly altered sandstones, which also include sulphide minerals of iron, nickel and copper in their upper parts. Sandstone consists of the following minerals: quartz, sericite, apatite, carbonates, biotite with chlorite, epidot, tourmaline, limonite and chematite.

Content of P_2O_5 in the ore has the greatest importance. It varies widely, from 1.5 to 19 %. The former explorations resulted in several determinations of reserves. The following data are relevant:

A category
4,169,256 t of ore with 11.87% P_2O_5

B category
42,353,450 t of ore with 9.54% P_2O_5

C_1 category
21,861,771 t of ore with 9.99% P_2O_5

Overall:
68,384,477 t of ore with 9.83% P_2O_5

Considering the fact that imported phosphate concentrates had much higher contents of P_2O_5 (even over 30 %), it is clear that this circumstance has the largest influence on final decision about start the mining works in this deposit. There is another big problem, related to the location of deposit, which is very unfavorable due to its distance from the processing plants (Prahovo, Sabac, etc.)

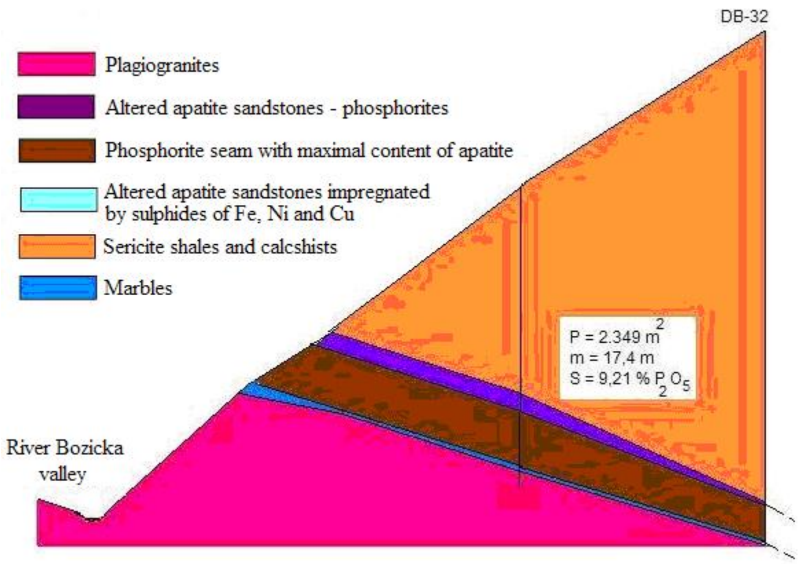

Figure 2. *Vertical cross-section (No. 7) of the deposit*

3. SELECTION OF OPENING TYPE AND LOCATION OF FACILITIES ON THE SURFACE

Former researches favored underground mining, but the newer ones pay more attention to the surface mining. Nevertheless, the way of opening the deposit for underground mining is still actual, since some parts of the deposit have to be extracted by the underground mining. Authors also think that plans for surface mining of shallow parts of the deposit should also be reconsidered, due to the serious environmental problems.

The influence factors to the way of opening and location of facilities are following:

- spatial layout of the deposit and its shallow part on the left side of the Bozicka River valley,

- terrain configuration that is very inconvenient for placement of facilities,
- deposit depth that is relatively low, although it was not fully explored,
- opening restrictions due to the ore hoisting system,
- location of the processing plant,
- location of the tailing dump,
- further transport of concentrate, etc.

Problem of opening and location the facilities can be spoted in Figure 3. Valley of the Bozicka River is very narrow and it is not wide enough for the mine facilities. In the northwestern part of the deposit, at Panjevica locality, situation is slightly better. In this area, on the right bank of the Bozicka River, the deposit is shallow and convenient for surface mining.

Considering that, the former research studies suggested that the facilities should be placed on the south, near the town of Bosilegrad (northeast from the town). That locality is suitable for mineral processing plant and its tailing dump, too. However, during the years, lots of houses were built in that location what means that it is not considerable any more.

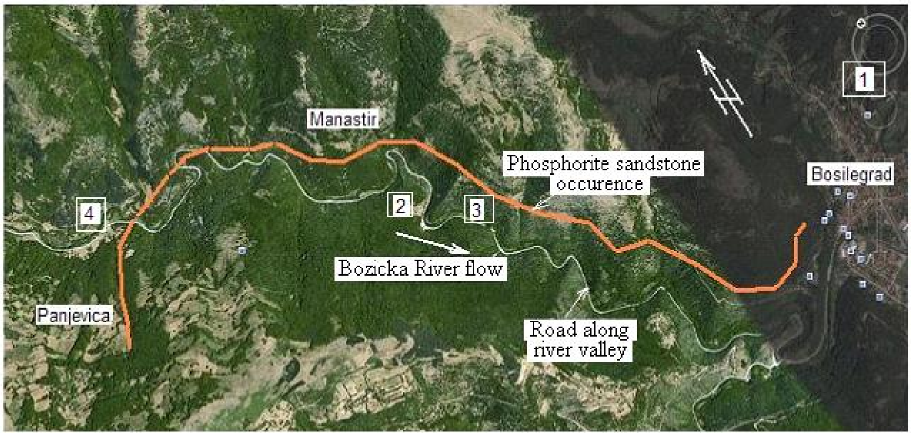

Figure 3. *Terrain surface around the Bozicka River valley*

1 – first location for facilities and mineral processing plant, suggested in the former studies, in 1975,
2 – location suggested in 2008,
3 – currently suggested location,
4 – proposed location Panjevica.

Initiated by Victoria Group, the new researches on the Lisina deposit started in 2008. Considerations of mine opening included the following circumstances:

1. Deposit layout is suitable for opening with slope drifts, with hoisting by belt conveyers. Other possibility is hoisting by trucks, but that was excluded due to a fact that horizontal transport should be per formed by rail.

2. Huge distances in the horizontal transport are the reason for placement the opening point in the middle of the deposit.

3. First idea for location of processing plant was the crook of the right side of river valley, since it has enough space for tailing dump, too. (Figure 4).

After these considerations, it was decided that facilities would be placed on the right bank of the river, across the Manastir locality (mark 2 at Figure 3), while layout of facilities, mineral processing plant and tailing dump is shown in Figures 4 and 5.

Main disadvantage of this solution is unfavorable position of both mine facilities and processing plant. These objects are situated in narrow area, 30 m above the road level, which means that access road is necessary, along with lots of ground works at the locality. Location of processing plant at elevation significantly higher than mine facilities causes increase of hoisting height for 350 m. Also, the access road from Donja Lisina to the processing plant increases investment costs, and later, transportation costs, too.

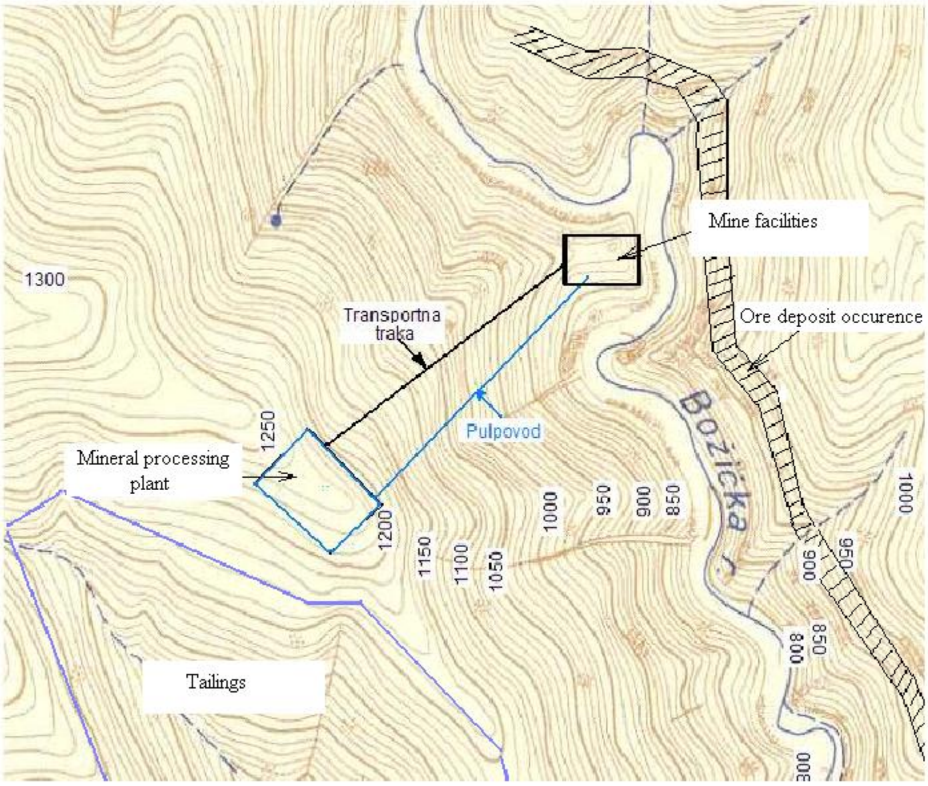

Figure 4. *Layout of mine facilities, mineral processing plant and tailing dump by the first solution, proposed in 2008*

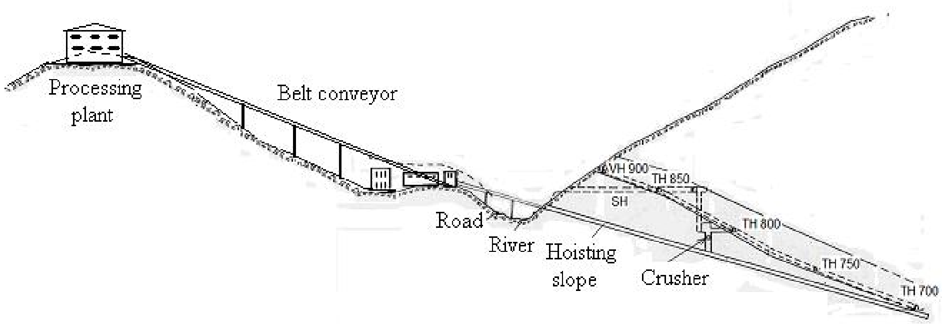

Figure 5. *Proposed mine opening for the Lisina phosphorite deposit*

During consideration of these problems, the authors concluded that some other ideas for tailings should be analized. Determination of location for facilities and processing plant mostly depended on location of tailing dump. However, the idea in the study was to dump mineral processing waste into excavated areas of underground mine. This would enable minimum waste dumping on the surface, only in the first couple years of mining (till the ends of extraction in the first underground mine sections). After that, all of the waste would be dumped in empty underground rooms. This would significantly improve the environmental protection and exclude the problem of tailings location.

In case of acceptance of this proposition, the mineral processing plant could be placed near Panjevica, in the area where topography is more suitable. There are lots of advantages that this solution brings, such as:

- the facilities would be placed near existing road, which is convenient from the aspect of transport and logistics during their installation,
- this location is in the northern part of deposit, which means that after mineral processing, concentrate would be transported in the same direction towards Vlasinsko lake and further to the fertilizer industry facilities,
- location of processing plant inside the river valley does not require increase of hoisting height, which is the case with first solution.

Disadvantage of this location is that it is close to the river, which requires protection measures for facilities.

If processing plant would be placed in the Panjevica area, there are two possibilities for location of mine facilities:

- keeping of current location, in the middle of deposit, which is favorable for underground haulage lengths,
- placement of mine facilities near processing plant, which would minimize surface transport, but significantly increase the underground haulage lengths.

Compromise could be made if the mine facilities remain in the middle of deposit, but closer to the river valley, as it is shown in Figure 6.

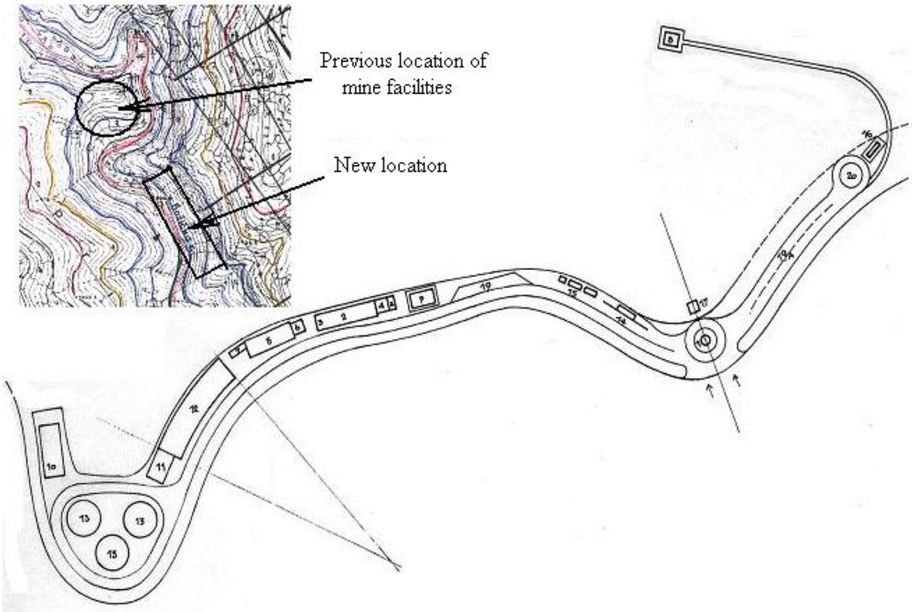

Figure 6. *Layout of the mine facilities on the surface, the newly proposed location*

4. CONCLUSION

„Lisina“ deposit will surely be a subject of further considerations and analyses, since it is a deposit of raw material necessary for production of fertilizers. Main problems are location and layout of the deposit, due to unfavorable ground surface and large distances from deposit to consumers.

Mine opening, as well as placement of mine facilities at the surface, is a very important issue. It depends on several factors and one of the most important is location of mineral processing plant.

Our suggestion is central opening, in the middle of deposit, and mineral processing plant in the river valley. Mineral processing waste would be dumped in the rooms of underground excavations, which is convenient both from the aspect of environment protection and aspect of economy, since it excludes the costs of tailings preparation and maintenance.

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SISTEMATIZACIJA PRIRODNO-GEOLOŠKIH USLOVA UTICAJNIH NA IZBOR SISTEMA PODZEMNOG OTKOPAVANJA U AKTIVNIM LEŽIŠTIMA UGLJA U SRBIJI

Izvod

U radu se obrađuje uticaj prirodno-geoloških uslova uticajnih na izbor sistema podzemnog otkopavanja u aktivnim ležištima uglja, pri čemu je poseban osvrt dat sistematizovanju uslova.
Ključne reči: *podzemna eksploatacija, prirodno-geološki uslovi eksploatacije, kratko mehanizovano čelo.*

UVOD

U složenim i različitim uslovima ležišta uglja u Srbiji primenjivana su brojna i specifična tehničko-tehnološka rešenja procesa podzemnog otkopavanja slojeva kamenog uglja, mrkog uglja i lignita, uz stalna nastojanja da se što više prilagode konkretnim prirodno-geološkim uslovima.

Može se istaći da su aktivnim, ležištima uglja u Srbiji relativno otežani uslovi eksploatacije, sa znatnim promenama od ležišta do ležišta, a često i između pojedinih otkopnih polja u istom ležištu. Ove promene posebno dolaze do izražaja spuštanjem eksploatacionih radova po dubini.

Raznolikost minerološko-petrografskih karakteristika, širok dijapazon vrednosti fizičko-mehaničkih svojstava uglja i pratećih stena, izražena tektonika, i variranje debljine ugljenig slojeva i dr. doprineli su u mnogome da se kao osnovni sistemi otkopavanja koriste stubne i stubno-komorne metode, a koje karakteriše:

- široka primena zbog velike mogućnosti prilagođavanja geometrije radne fronte otkopa i taktike otkopavanja složenim uslovima eksploatacije;
- niska proizvodnost i produktivnost, te potreba rada većeg broja otkopnih jedinica;
- dekoncentrisanost otkopnih i pripremnih radova;
- nizak stepen mehanizovanosti (izuzev transporta) na otkopnim jedinicama;
- visoko učešće pripremnih radova.

Primena metode mehanizovanih širokih čela dala je dobre proizvodne rezultate samo u određenim otkopnim poljima u ležištima rudnika „Bogovina“ i „Rembas“.

Treba istaći i metodu širokog čela sa kratkim radnim frontom i kompleksnom mehanizacijom koja je bila u eksperimentalnom radu u jami „Senjski Rudnik“.

Ovo čelo (takozvano „slepo čelo“) sa jednom transportno-ventilacionom prosto-

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rijom i separatnim provetravanjem otkopa primenjivo je u sistemu vertikalne koncentracije, sa dužinom otkopa od cca 25 m i ostvarilo je zadovoljavajuće proizvodne i ekonomske efekte.

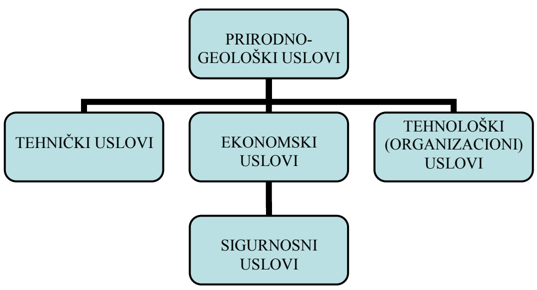
OPŠTI I SPECIFIČNI USLOVI EKSPLOATACIJE U AKTIVNIM LEŽIŠTIMA

Na osnovu prikupljenih i obrađenih podataka, osnovne prirodno-geološke uslove aktivnih ležišta uglja karakteriše sledeće:

- Najčešće se radi o debelim ugljenim slojevima, sa čestim izmenama debljine;
- Ugljeni slojevi su uglavnom složene strukture sa manjim ili većim učešćem proslojaka stena;
- Usled intenzivne makro i mikro tektonike, ležišta su po pravilu izrasedana u više sistema raseda, koji su formirali otkopna polja napravilnih oblika i relativno malih dimenzija;
- Aktivna ležišta uglja nisu izraženi nosioci metana. Izuzetak čine ležišta „Vrška Čuka“ i „Soko“ sa nešto većim vrednostima metanonosnosti. Pored toga za ležište „Soko“ je karakteristično prisustvo u krovini slabovezanog peščara koji je kolektor metana pod pritiskom naročito u području markantnih raseda, što pod određenim uslovima može izazvati izboj gasa odnosno gasa i materijala;
- U odnosu na pad ugljenih slojeva, ležišta pripadaju uglavnom blago nagnutim ležištima, a ređe nagnutim i strmim (ležište mrkog uglja „Soko“);
- U pogledu dubine zaleganja ugljenog sloja većina ležišta sada pripadaju grupi rudnika sa srednjom dubinom eksploatacije (do 400 m), izuzev jame „Jarando“, „Strmosten“ i „Soko“ koji se odlikuju nešto većom dubinom eksploatacije;
- Stabilnost rudarskih podzemnih objekata direktno je vezana za fizičko-mehanička svojstva ugljenih slojeva i pratećih naslaga (sa naglašenim učešćem glinovitih komponenti) u kojim se primenjuju uobičajeni sistemi podrađivanja;
- U pogledu vodonosnosti sva ležišta, izuzev ležišta „Štavalj“ pripadaju grupi rudnika sa dotokom manjim od 1,0 m^3/min odnosno grupi slabo ovodnjenih rudnika;
- Samozapaljivost i eksplozivnost ugljene prašine, kao i prirodna sklonost ka samozapaljenju je opšta karakteristika svih ležišta uglja, izuzev ležišta „Vrška Čuka“.

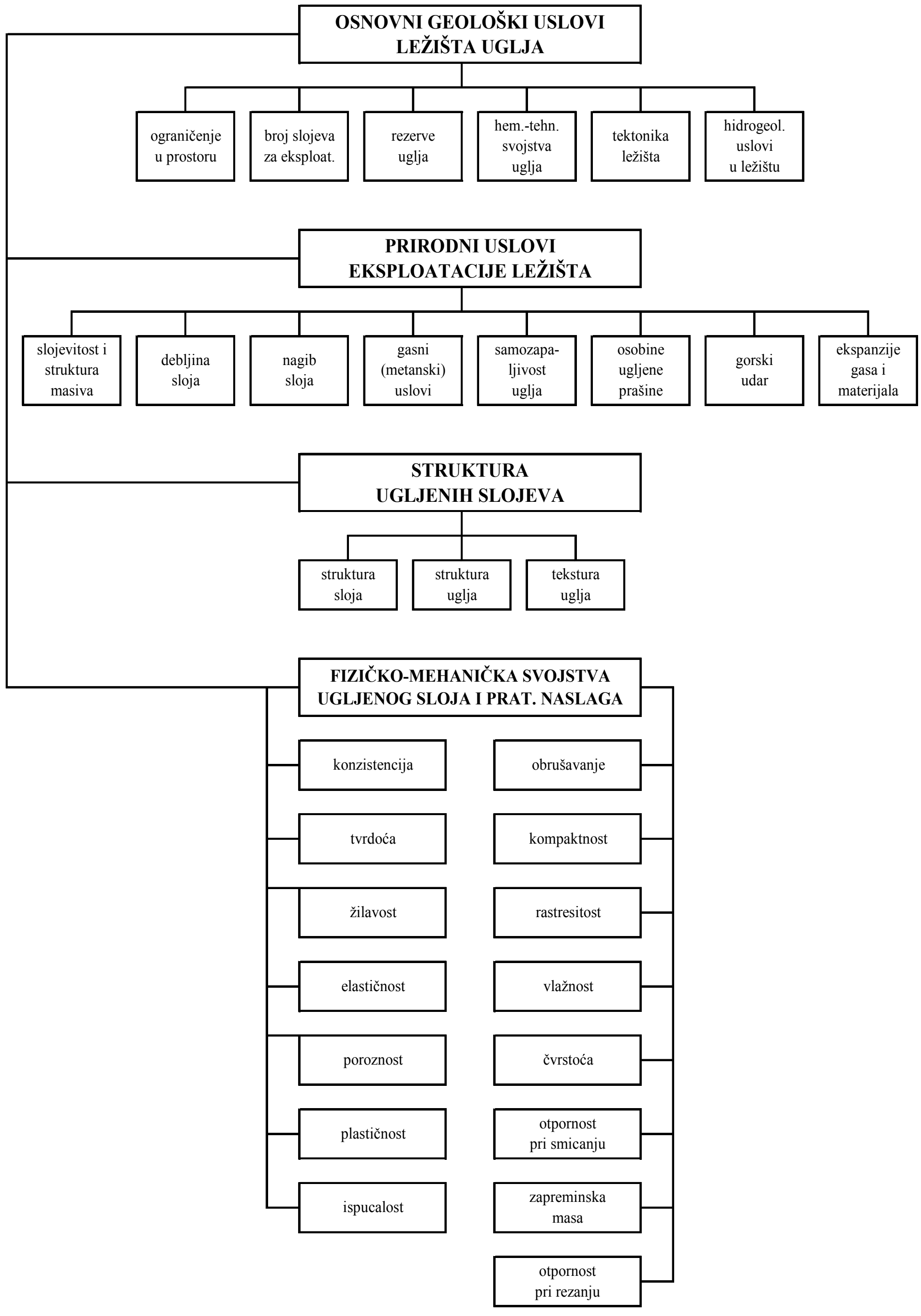
SISTEMATIZACIJA PRIRODNO-GEOLOŠKIH USLOVA UTICAJNIH NA IZBOR SISTEMA OTKOPAVANJA

Uopšteno na izbor sistema podzemnog otkopavanja ugljenih slojeva u određenom ležištu dominantan je sistem uslova predstavljen na slici 1.



Sl. 1. *Sistem uslova kod izbora metode i tehnologije otkopavanja slojeva uglja*

Pri izboru sistema otkopavanja polazni element je definisanje prirodno-geoloških uslova u ležištu, a što je detaljnije predstavljen na slici 2.



Sl. 2. *Karakterističan poprečni profil, (br. VII), kroz ležište fosforita „Lisina*

ZAKLJUČAK

Na izbor racionalnih sistema podzemnog otkopavanja u ležištima uglja u Srbiji odlučujući uticaj imaju prisutni prirodno-geološki uslovi eksploatacije.

U aktivnim ležištima dominantni geološki oblici su slojevite, nagnute strukture sa izraženim tektonskim deformacijama, čije su posledice nepravilni oblici ograničenih eksploatacionih područja i relativno kratke dužine otkopnih polja i blokova sa čestim promenama, pravca pružanja i uglova pada slojeva. Ugljeni slojevi debljine nekoliko do 40 m imaju horizontalno zaleganje do nagnut pad, ređe strm. U nekim ležištima su nepovoljna osnovna fizičko-mehanička svojstva pratećih naslaga te se može konstatovati da se u mnogome, kod većine sada aktivnih ležišta, sužavaju mogućnosti primene velikih mehanizovanih proizvodnih sistema i koncentracije proizvodnje u istim. Za ovakve uslove prioritetno je definisati racionalan sistem (metoda i tehnologija) otkopavanja.

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SYSTEMATIZATION OF INFLUENTIAL NATURAL-GEOLOGICAL CONDITIONS ON SELECTION THE SYSTEM OF UNDERGROUND MINING IN THE ACTIVE COAL DEPOSITS IN SERBIA

Abstract

This paper discusses the impact of natural-geological conditions on selection the system of underground mining in the active coal deposits, with special emphasis on systematization of conditions.

Key words: *underground exploitation, natural-geological conditions of mining, short mechanized face*

INTRODUCTION

In the complex and varying conditions of coal deposits in Serbia, a number and specific technical - technological solutions were applied for the process of underground mining the layers of stone coal, brown coal and lignite, with continuous efforts to adapt as much as possible to the specific natural-geological conditions.

It may be noted that in the active, coal deposits in Serbia are relatively difficult conditions of exploitation, with significant changes from deposit to deposit, and often between individual exploitation fields in the same bed. These changes are manifested in particular by lowering the exploitation works in depth.

The diversity of mineralogical-petrographic characteristics, a wide range of values of physical-mechanical properties of coal and associated rocks, expressed tectonics, and varying the thickness of coal layers, etc. have contributed greatly that the basic systems of mining are pillar and pillar-chamber methods, which are characterized by:

- widely use because of the great adaptability of geometry the working front of stope and tactics of excavation in the complex conditions of exploitation;
- low production and productivity, and need for more excavation units;

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- deconcentration of excavation and preparation works;
- low level of mechanization (excluding transport) at the excavation units;
- high share of preparation works.

The use of method of mechanized wide face gave good production results only in the certain excavation fields in the mine deposits „Bogovina“ and „Rembas“.

It should be also noted that the method of wide face with short operation front and complex mechanization that was in the experimental work in the pit „Senjski Mine“. This face (so-called “blind face”) with one transport-ventilation room and a separate ventilation of the adit is applicable in the system of vertical concentration, with length of approximately 25 m and it achieved the satisfactory production and economic effects.

GENERAL AND SPECIFIC CONDITIONS OF EXPLOITATION IN ACTIVE DEPOSITS

Based on collected and processed data, the basic natural- geological conditions of active coal deposits have the following characteristics:

- Usually there are thick coal seams, with frequent changes in thickness;
- The coal layers are generally complex structures with smaller or larger share of intermediate rocks;
- Due to intense macro and microtectonics, deposits are generally more faulted in many systems of fault, which formed the excavation fields of irregular shape and relatively small sizes;
- Active coal deposits are not expressed methane carriers. Exceptions are deposits „Vrška Čuka“ and „Soko“ with slightly larger values of methane. In addition to this, the deposit „Soko“ is characterized by the presence of sandstone in the top soil which is the methane collector under pressure particularly in the striking faults, which under certain conditions can cause a discharge of gas or gas and material;
- In relation to the decline in coal seams, the deposits mostly belong to the slightly sloped deposits, and rarely to the inclined and steep (deposit of brown coal „Soko“;
- Related to the depth of coal seam, mostly deposits now belong to a group of mines with average depth of exploitation (400 m) except the pits „Jarando“, „Strmosten“ and „Soko“ which are characterized by slightly greater depth of exploitation;
- The stability of underground mining facilities is directly related to the physical-mechanical properties of coal seams and associated deposits (with emphasis on the participation of clay components) in which the common systems of support are applied;

- Regarded to the water bearing of all deposits, except the deposit „Štavalj“, they belong to a group of mines with a lower inflow of 1.0 m^3/min and a group of poorly water bearing mines;
- Self-ignition and explosiveness of coal dust as well as the natural tendency to self-ignition is a general characteristic of all coal deposits, except deposit „Vrška Čuka“.

SYSTEMATIZATION OF NATURAL-GEOLOGICAL CONDITIONS INFLUENTIAL ON A SELECTION OF MINING SYSTEM

Generally, a selection of system on underground mining of coal seams in the certain deposit is a dominant system of conditions presented in Figure 1.

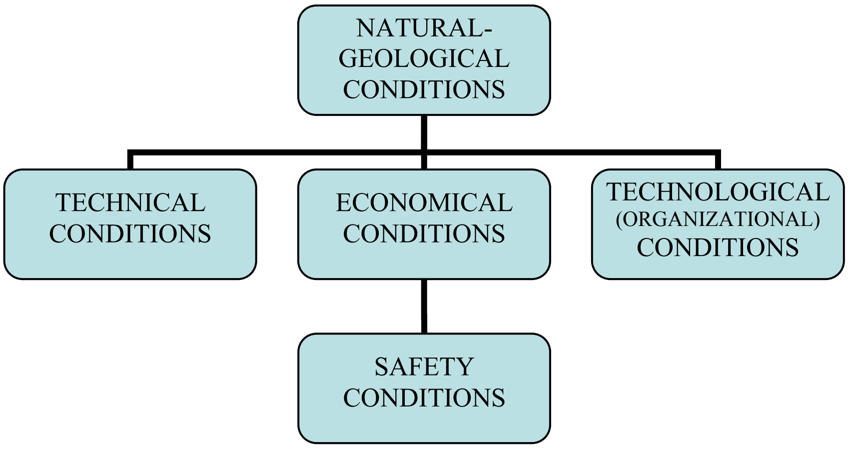

Figure 1. *System of conditions in selection of method and technology of mining the coal seams*

In selection the mining system, the starting element is defining the natural-geological conditions in a deposit, what is presented in detail in Figure 2.

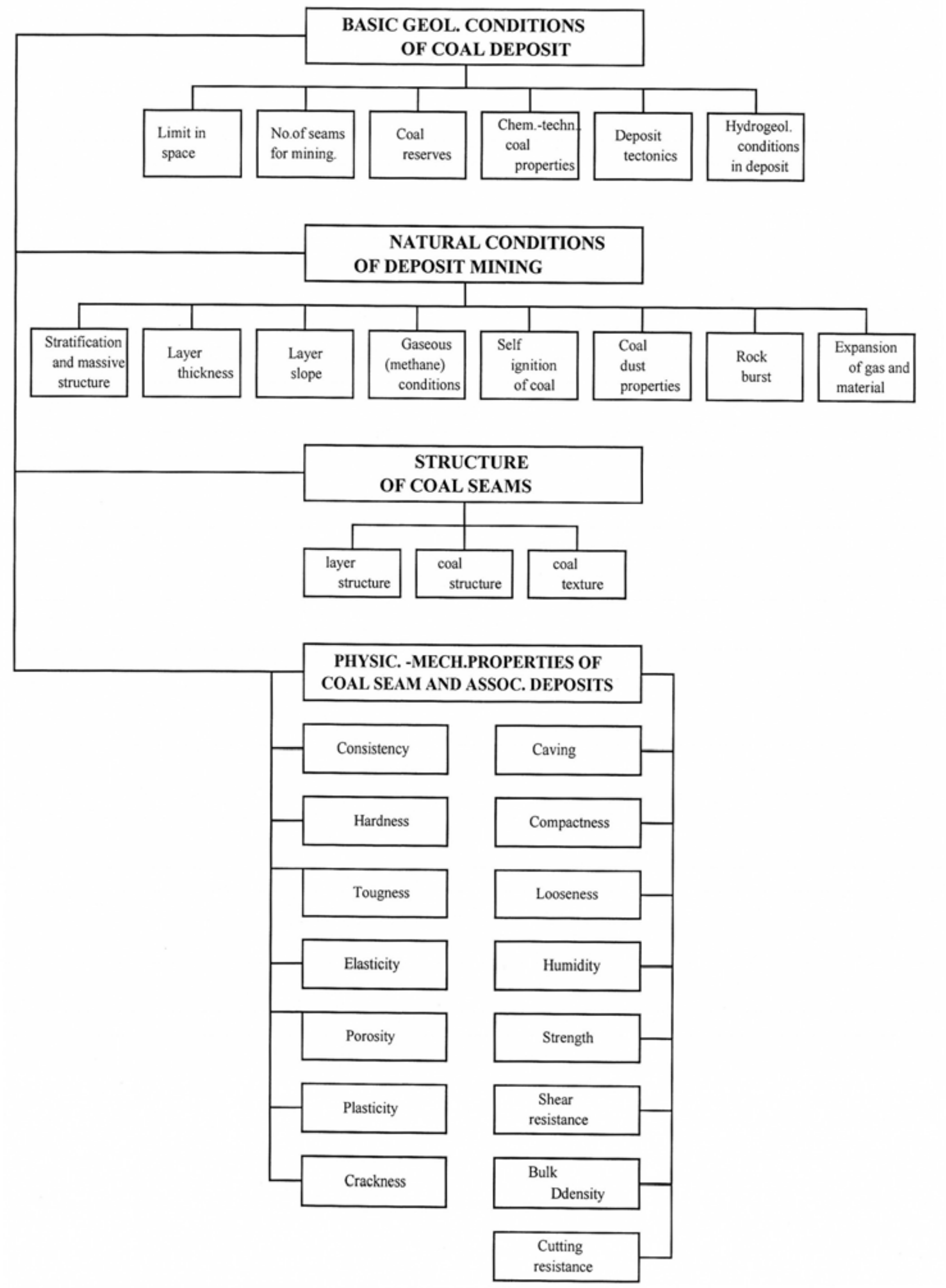

Figure 2. *Vertical cross-section (No. 7) of the deposit*

CONCLUSION

A decisive influence on a selection of rational systems of underground mining the coal deposits in Serbia have the present natural-geological conditions of mining.

In the active deposits, the dominant geological forms are layered, sloping structures with the expressed tectonic deformations, with the consequences of irregular forms of limited mining areas and relatively short length excavation fields and blocks with frequent changes, the direction of fall and angles of layers. Coal seam thickness of several to 40 m has tilted horizontal direction to the sloped fall, less steep. In some deposits, the unfavorable are basic physical-mechanical properties of the supporting layers, so it can be stated that in many ways, mostly now active deposits have narrowing possibilities of use large mechanized production systems and concentration of production in the same.. For these conditions, it is a priority to define a rational system (methods and technologies) of excavation.

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KOMPARATIVNA ANALIZA PRIMENE MEHANIZOVANOG HIDRAULIČNOG ŠIROKOG ČELA U ODNOSU NA KLASIČNE OTKOPNE METODE ZA EKSPLOATACIJU UGLJA U JAMI ″PETNJIK″ RUDNIKA MRKOG UGLJA ″BERANE″, CRNA GORA

Izvod

Podzemna eksplotacija mrkog uglja na ležištu „Petnjik" ivangradskog ugljenog basena započela je početkom osamdesetih godina otvaranjem jame „Petnjik" tadašnjeg Rudnika mrkog uglja „Ivangrad". Od otvaranja rudnika za eksploataciju uglja koristile su se uglavnom klasične metode otkopavanja, i to komorno stubne metode i metode klasičnog širokog čela. 2001 godine prvi put je u proizvodni proces uvedena metoda mehanizovanog širokog čela sa savremenom hidrauličnom mehanizacijom kao probna otkopna metoda. Uvođenjem ove metode otkopavanja omogućili smo realizaciju masovnije proizvodnje mrkog uglja, obezbeđeni su znatno kvalitetniji uslovi rada sa daleko većim otkopnim učincima i znatno boljim nivoom zaštite na radu. Komparativna analiza primene ove otkopne metode u eksploataciji uglja u jami „Petnjik" Rudnika mrkog uglja „Berane" u odnosu na ranije primenjivane klasične metode je prikazana u ovom radu.

Ključne reči: *Podzemna eksploatacija uglja, mrki ugalj, jama, metoda otkopavanja, mehanizovano široko čelo.*

UVOD

Rudnik "Berane" nalazi se na prostorima Ivangradskog ugljonosnog basena, koji se nalazi se u neposrednoj okolini grada Berana na nadmorskoj visini između 650 i 800 m. Otkopavanje uglja u ovom basenu otpočelo je šezdesetih godina prošlog vijeka u reviru ″Budimlja″. Eksploatacija u tom reviru vršena je podzemnim metodama eksploatacije, na dubini od oko 60 m ispod nivoa zemlje, dok su pojedini izdanci uglja manjeg obima otkopavani površinski. Eksploatacija ovog revira završena je krajem sedamdesetih godina, kada je i započeta investiciona izgradnja novog rudnika u u reviru ″Petnjik″, da bi 1981. godine otpočela proizvodnja u istoimenoj jami, gde se i danas vrši ekploatacija uglja. Kako u reviru „Budimlja", tako i kasnije u „Petnjku" proizvodni proces odvijao se u veoma teškim uslovima, primjenom klasičnih niskoproduktivnih metoda otkopavanja, komorno-stubnih metoda i

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metoda klasičnog širokog čela, usled čega je dolazilo i do čestih prekida proizvodnje.

Godine 2001. Rudnik „Ivangrad“ je prvi put privatizovan od strane Preduzeća ″Gradex-HBP″, kada je i prvi put u proizvodni proces uvodena savremene tehnologije eksploatacije uglja. Efekti uvođenja savremene mehanizacije, po našem mišljenju, bili su izvanredni, ali je, na žalost, godinu dana nakon pokretanja proizvodnje, ista obustavljena zbog problema nalaženja izvora dugoročnog finansiranja, kao i problema imovinsko pravne prirode.

Danas je Rudnik u Beranama vlasništvo firme „Balkan Energy“ D.O.O. Podgorica, koja je dio grčke Restis Grupe koja pokušava da iznađe najbolja tehnološka rešenja za realizaciju dugoročno održive proizvodnje u jami „Petnjik“ kako bi se na najbolji način valorizovale postojeće rezerve kvalitetnog mrkog uglja.

OPŠTE KARAKTERISTIKE

Lokacija ležišta

Rudnik mrkog uglja ″Berane″ nalazi se na prostorima ivangradskog ugljenog basena, koji se nalazi u kotlinskim zaravnima gornjeg toka reke Lim i njenih pritoka Bistrice i Sušice, (leve pritoke) kao i vodotoka Budimke, Makve, Brnjice i Kaluđerske reke (desne pritoke), istočno od planine Bjelasica, uokviren sa severa planinama Tivranom, Kamenicom i Jugovinom, sa južne i jugoistočne strane planinama Vukodaricom, Raščom i Bukovcem, a sa istočne strane Prosjenom i Gradacem. Ležište mrkog uglja “Petnjik” kao deo Ivangradskog ugljonosnog basena zahvata površinu od oko 12 km^2.

Komunikaciona povezanost je relativno povoljna i odnosi se na putnu mrežu koja pretežno ide koritom reke Lim. Putevi su II i III reda i usmereni su u tri pravca i to u pravcu severa ka Bjelom Polju, istoka ka Rožajama i juga ka Andrijevici. Preko ovih putnih pravaca ostvarena je povezanost sa svim ostalim delovima Crne Gore, kao i sa Srbijom. Najbliža železnička stanica na pruzi Beograd-Bar nalazi se u Bjelom Polju na udaljenosti oko 35 km od Rudnika.

Beranski kraj je poznat po izraženoj kontinentalnoj klimi, kao i izuzetno niskim temperaturama i snežnom pokrivaču u zimskom periodu koje znatno otežavaju rad i ometaju transport uglja i dopremu repromaterijala na površini.

Geološke karakteristike ležišta

Geološka istraživanja ležišta ″Petnjik″ izvršena su osamdesetih godina na osnovu 126 istražnih bušotina, a važeća interpretacija geoloških i tektonskih uslova ležišta urađena je u Elaboratu o klasifikaciji, kategorizaciji i proračunu rezervi uglja u reviru Petnjik ivangradskog ugljenog basena, stanje na dan 31.12.2001. godine iz 2002. godine, u kome su i overene rezerve mrkog uglja ovog ležišta.

Ležište “Petnjik” karakteriše postojanje 4 ugljena sloja, zavisno od lokaliteta, od kojih, konstantno razviće po celom ležištu poseduje samo glavni ili najmlađi sloj. Prvi i drugi podinski ugljeni sloj nisu razvijeni po celom ležištu, već samo lokalno, dok se treći javlja samo sporadično. Debljina ugljonosne serije je promenljiva, ali u ležištu “Petnjik” ne prelazi 40 m. Dubina zaleganja ugljenih slojeva iznosi 150-200 m. Glavni ugljeni sloj prosečne je debljine 3,5-4,8 m sa mestimičnim zadebljanjima na manjim površinama koja iznose 5,0-7,6 m. Glavni sloj je protkan proslojcima jalovine izrazito promenljive debljine od 0,01 do 1,5 m. Rastojanje između slojeva uglja je od 5 do 10 m.

Neposrednu krovinu glavnog sloja čine laporci i vrlo retko gline i peskovite gline. Podina je laporac mestimično peskovit. Generalni pad sloja je u pravcu zapada (15%). Sloj u zapadnom delu ima mali pad i skoro je horizontalan za razliku od ostalog dela gde se pad sloja kreće od 10°-25°.

Tektonika ležišta ″Petnjik″, kao i celog ivangradskog ugljenog basena je vrlo složena. U tektonskom pogledu ležište je ispresecano mnogobrojnim rasedima koji su podeljeni na značajne rasede regionalnog karaktera tj. glavne rasedne linije i sporedne rasedne linije. Na osnovu istražnih površinskih bušenja ležište je u tektonskom smislu podeljeno na šest tektonskih blokova. Kapitalni rasedi se pružaju upravno jedni na druge i uglavnom su u pravcu pružanja S-J i SZ-JI. Skok krila kapitalnih raseda iznosi i preko 10 m.

Iz geološkog prikaza ležišta "Petnjik", može se ustanoviti da su rudarsko-geološki uslovi u eksploatacionim blokovima ležišta u kojima se vrši eksploatacija prilično složeni usled činjenice da je tektonika vrlo komleksna, da su blokovi ispresecani mnogobrojnim rasedima, a ugljeni slojevi promenljive debljine. Takođe, nemogućnost detaljne geološke analize eksploatacionog sloja uglja i pratećih stena, zehteva često pretpostavljanje tektonskih karakteristika, što u velikoj meri otežava rad i prouzrokuje probleme pri eksploatacji. Rudarskim radovima u jami "Petnjik" takođe je potvrđen veliki broj manjih raseda koji, predstavljaju veliki izazov pri postavljanju otkopnih stubova i otežavaju izvođenje rudarskih radova u jami.

Rezerve i kvalitet uglja

Sve istražene geološke rezerve prema ranije pomenutom Elaboratu iz 2002. godine svrstane su u bilansne, vanbilansne i potencijalne rezerve. Od proračunatih ugljenih slojeva u reviru "Petnjik" najbolje je proučen glavni ugljeni sloj. Pored bilansnih kategorija A, B i C_1 u ovom ugljenom sloju javljaju se vanbilansne B" i C_1 i potencijalne C_2. Prvi podinski ugljeni sloj je manje proučen i u njemu su proračunate bilansne i vanbilansne rezerve C_1 kategorije i potencijalne rezerve C_2 kategorije. Drugi i treći ugljeni sloj imaju samo lokalno rasprostranjenje.

Rezerve uglja po slojevima i kategorijama u ležištu "Petnjik", prikazane su u tabeli 1.

Tabela 1. *Rezerve uglja ležišta „Petnjik"*

Kategorija rezervi	Bilansne geološke rezerve (t)	Eksploatacione rezerve (t)
Glavni ugljeni sloj (B + C_1)	12.715.730	10.172.583
Prvi podinski ugljeni sloj (C_1)	450.215	360.172
Ukupno (B + C_1)	13.165.945	10.532.756

Ugalj ležišta "Petnjik" je tamno-mrke do crne boje sa sledećim karakteristikama: ne prlja prste, ogreb mu je tamno-mrk, preleom neravan a ređe školjkast sa jasno izraženom deljivošću po vertikali. Na osnovu petrografskih ispitivanja kao i na bazi hemijsko-tehnoloških proba ugalj pripada grupi mrko-lignitskih ugljeva, odnosno tvrdim mrkim ugljevima. Na osnovu rezultata dobijenih laboratorijskim ispitivanjem kvaliteta rovnog uglja, dobijen je kvalitet uglja glavnog sloja koji je prikazan u tabeli 2. [1]

Tabela 2. *Kvalitet uglja glavnog sloja*

Vlaga %	Pepeo %	S %			Koks %	C/fix	Sag. materije	Ispar. mater	GTE KJ/kg	DTE KJ/kg	Zapr. tež. t/m^3
		uk.	sag.	pepeo							
22,58	22,55	1,61	1,03	0,62	46,78	23,83	54,18	30,76	15,13	13,02	1,30

USLOVI EKSPLOATACIJE U JAMI ″PETNJIK″

Jama ″Petnjik″ je otvorena centralnim načinom otvaranja, izvoznim i ventilacionim oknom. koja su međusobno povezana navozištem. Prostorije otvaranja, glavni transportni i glavni ventilacioni hodnik, izrađeni su u podini ugljenog sloja, paralelno, na međuosnom rastojanju od 30 m. Izradom ovih prostorija poduhvaćene su rezerve uglja u polju ″B″, ″C″ i ″D″. Prostorijama osnovne pripreme otvoreno je otkopno polje ″B″ dok su otkopni stubovi polja pripremani prostorijama otkopne pripreme sa po dva paralelno otkopna hodnika (izvoznim i ventilacionim) na krajevima povezanim otkopnim čelom.

U glavnim transportnim prostorijama instaliran je sistem transportnih traka širine 800 mm. Transport sa otkopnih radilišta vrši se grabuljastim transporterima.

Provetravanje jame je protočno. Radi se o prostom paralelnom sistemu provjetravanja, pri čemu se sveža vazdušna struja uvodi u jamu izvoznim oknom i dalje se razvodi sistemom glavnih transportnih prostorija do radilišta. Izlazna vazdušna struja se sistemom glavnih ventilacionih prostorija odvodi do ventilacionog okna, i preko glavnog ventilatora koji je instaliran na površini, izvodi se u spoljnju atmosferu.

Jama ″Petnjik″ Rudnika mrkog uglja ″Ivangrad″, iako kategorisana kao metanska, odlikuje se veoma povoljnim uslovima rada. Koncentracije metana se kreću u dozvoljenim granicama, a u dosadašnjem radu nije bilo pojave jamskih požara, većih prodora vode, gorskih udara ili sličnih nesreća.

EKSPLOATACIJA UGLJA

U prethodnoj fazi eksploatacije uglja, tj. pre uvođenja metode mehanizovanog širokog čela, u jami ″Petnjik″ otkopavanje je vršeno nemehanizovanim otkopnim metodama uz primenu bušačko minerskih radova. Uglavnom su korišćene dve metode: komorno-stubna otkopna metodom – tzv. metoda ″brazda″ i metoda otkopavanja širokočelnom metoda sa frikcionim stupcima i čeličnim gredama, pri čemu se utovar uglja sa radilišta vršio ručno ili pomoću skrepera.

U delu polja „B“ jame „Petnjik“ početkom 2002. godine prvi put je postavljeno manje mehanizovano široko čelo kao probno radilište. Uvođenje mehanizovane širokočelne metode otkopavanja podrazmuevalo je primenu hidraulične štitne podgrade tipa BMV-1 i BMV-1M, kombajna tipa ESA-150L za mehanizovanim dobijanjem uglja rezanjem, kao i mahanizovan odvozom uglja sa širokog čela trolančanim grabuljastim transporterom TH 604/F. Sva navedena oprema je slovačke proizvodnje.

- **Sekcije hidraulične štitne podgrade tipa BMV-1 i BMV-1M** [2], poseduju sledeće tehničke karakteristike**:**

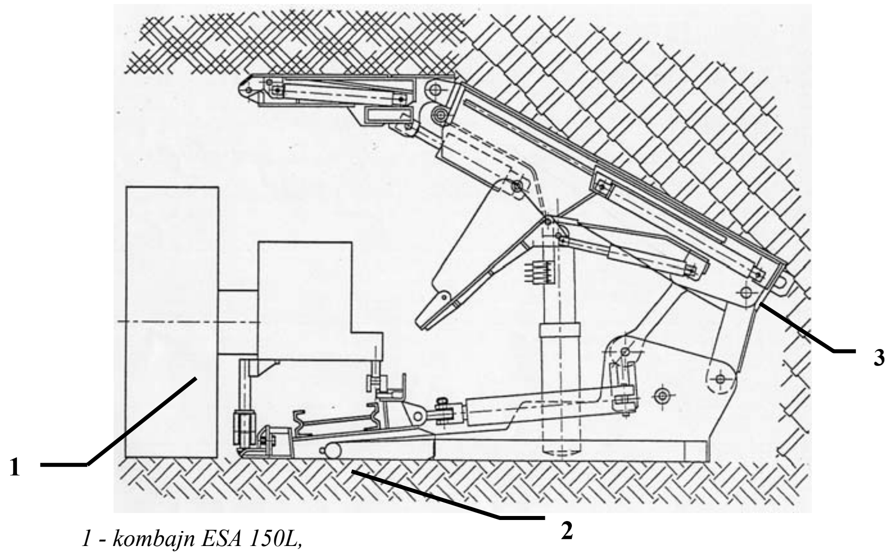
- max visina sekcije	3.400 mm
- min visina sekcije	2.320 mm
- širina sekcije	1.500 mm
- korak sekcije	700 mm
- broj podupirača	2 kom
- sila pritiska podupirača	1.216 kN
- otpor podgrade	1.296 kN
- pritisak na tlo	0,52 MPa
- sila premeštanja sekcije	391 kN
- sila premeštanja transportera	251 kN
- radni pritisak hidrauličnih pumpi	32 MPa
- masa sekcije	11.500 kg

otvor za ispuštanje krovnog uglja

- širina	850 mm
- dužina	1.100 mm

Sekcija se sastoji od sledećih osnovnih delova: podinski ram, stropni štit, greda sa stubnim osloncem i hidraulični podupirač.

Šematski prikaz mehanizovanog širokog čela sa hidrauličnom sekcijom tipa BMV-1 [3] dat je na slici br 1.



1 - kombajn ESA 150L,
2 - otkopni transporter TH 604/F,
3 - hidraulična sekcija BMV-1M.

Sl. 1. *Shematski prikaz mehanizovanog širokog čela*

- **Kombajn tipa ESA-150L** za dobijanje uglja rezanjem i utovar iskopine, ima sledeće tehničke karakteristike:

- snaga motora	150 kW
- visina kombajna od tla	1.650 mm
- prečnik radnog organa	1.600 mm
- max visina rezanja	3.330 mm
- max podrez. ispod nivoa tla	460 mm
- broj obrta radnog organa	39 obr/min
- vučna sila	250 kN
- težina	16.000 kg
- napon	3 x 1000 V
- sistem pomeranja	bezlančani

- **Trolančani grabuljasti transporter tipa TH 604/F** za odvoz uglja sa čela, sledećih tehničkih karakteristika:

- kapacitet	360 t/h
- dužina transportera u zavisnosti od nagiba	70-250 m
- snaga motora	100 kW
- brzina	0,6-1,07 m/s
- širina	642 mm
- rastojanje između prečki	1.032 mm
- rastojanje između lanaca	120 mm

- **Hidraulični agregat tipa HA 80/320 sastoji se od sledećih delova:**
  - rezervoar emulzije (voda-ulje u odnosu 97-98 % : 2-3%);
  - agregat koji se sastoji od:
    - elektromotora snage 55 kW za pokretanje troklipne hidr. pumpe,
    - elektromotora snage 3 kW za pokretanje pumpe za emulziju,
    - hidraulične troklipne pumpe,
    - pumpe za emulziju,
    - filtera za emulziju,
    - hidrauličnog cevovoda,
    - uređaja za kontrolu radnog pritiska.

Radni pritisak hidrauličnog agregata iznosi 320 MPa. Agregat radi u zatvorenom sistemu. Pored hidrauličnog agregata koji je u radu, neophodno je instalisati i rezervni agregat.

Tehnologija otkopavanja mehanizovanim širokim čelom

Primenom napred opisane mehanizacije za širokočelno otkopavanje uglja u zavisnosti od moćnosti ugljenog sloja, dobijanje uglja može se vršiti na dva načina:

- dobijanje uglja rezanjem u širokom čelu bez miniranja
- dobijanje uglja rezanjem u frontu i miniranjem natkopnog dela sloja.[4]

Širokočelno mehanizovano otkopavanje sa dobijanjem uglja rezanjem u frontu širokog čela i zarušavanjem krovine, primenjuje se za slojeve moćnosti do 4,5 m. Visina kopanja rezanjem u frontu čela limitirana je visinom hidraulične štitne podgrade.

Širokočelno mehanizovano otkopavanje sa dobijanjem uglja rezanjem u frontu širokog čela i miniranjem uglja iz nadkopa, uz zarušavanje krovine, primenjuje se kod otkopavanja ležišta, ili dijela ležišta veće moćnosti. Suština ove metode je da se otkopavanje uglja u čeonom stubu vrši kombajnom, dok se dobijanje uglja iz natkopa vrši miniranjem.

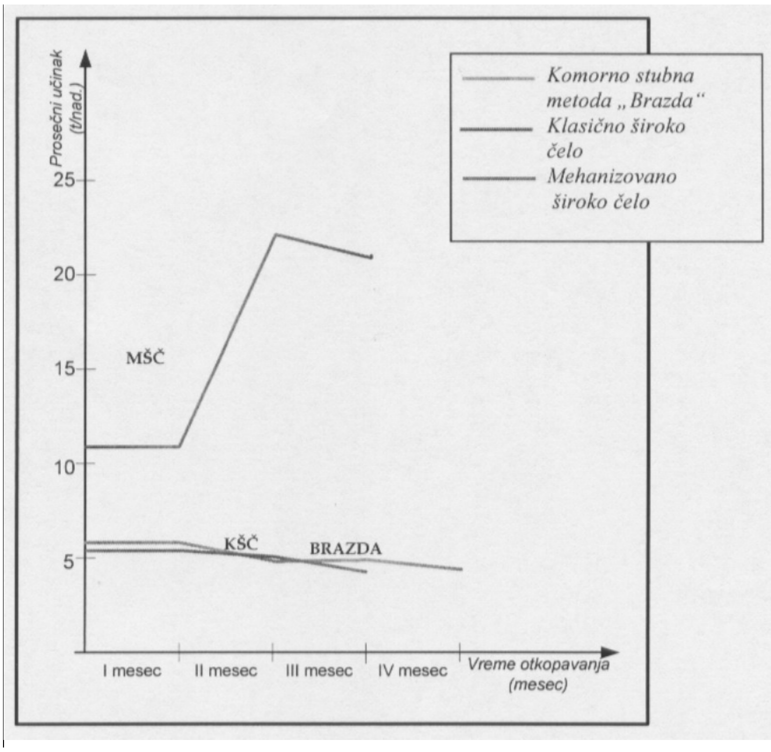
Glavne faze u tehnološkom procesu otkopavanja uglja primenom mehanizovanog širokog čela su: dobijanje uglja, utovar i odvoz iskopine, podgrađivanje otkopnog prostora i zarušavanje starih radova iza fronta otkopavanja.

DISKUSIJA

Na osnovu podataka iz proizvodnog procesa iz prethodnog perioda može se napraviti komparativna analiza otkopnih učinaka primenom sve tri analizirane otkopne metode: komorno - stubnom metodom „Brazde“, metodom klasičnog širokog čela i metodom mehanizovanog širokog čela. Ostvareni otkopni učinci (izraženi u tonama po nadnici) po mesecima su upoređeni na dijagramu koji je prikazan na slici br. 2.

Iz dijagrama se jasno može videti da učinci ostvareni klasičnim metodama kreću oko 5 t/nadnici, dok su učinci ostvareni metodom mehanizovanog širokog čela tri do četiri puta veći, tj. dostižu vrednosti do 24 t/nadnici. Pri tome treba imati u vidu da su mehanizovana široka čela koja su korišćena za otkopavanje predstavljala probno otkopavanje preostalih rezervi polja „B“, pa su bila ograničene dužine (35m) i radila u otežanim uslovima rada usled izražene tektonike i uticaja starih radova, pa smatramo da bi pri otkopavanju u novom otkopnom polju, u povoljnijim rudarsko-geološkim uslovima i sa većom dužinom širokog čela ovi učinci dostizali i veće vrednosti.

Sveobuhvatnom analizom svih efekata primene metoda otkopavanja sa mehanizovanim širokim čelom na osnovu iskustava u probnom otkopavanju u polja „B“ jame „Petnjik“ dolazi se do zaključka da je navedena metoda daleko produ-ktivnija od ranije primenjivanih nemeha-nizovanih metoda i da njena primena pokazuje mnogobrojne prednosti, kako u pogledu radnih učinaka, tako i u pogledu opštih uslova rada i sigurnosti na radu.



Sl. 2. *Dijagram učinka pri otkopavanju primenom različitih otkopnih metoda*

U poređenju sa primenjivanim metodama otkopavanja, mehanizovano širokočelno otkopavanje ima niz prednosti, i to:

- veći stepen iskorišćenja ugljene supstance;
- rad se odvija u potpuno podgrađenom prostoru;
- radilište ima dva izlaza;
- provetravanje širokog čela je protočno;
- neuporedivo veći komfor i sigurnost rada;
- bušačko minerski radovi svode se na minimum (samo na dobijanje uglja iz natkopa);
- smanjuje se učešće mauelnog rada;
- dinamika napredovanja mehanizovanog širokog čela obezbeđuje stabilnost pristupnih prostorija osnovne pripreme;
- neuporedivo veća produktivnost (ostvaruju se otkopni učinci preko 20 t/nad);
- znatno smanjen broj radnika na širokom čelu;
- manja opasnost od endogenih požara.

Neizbežno je, naravno i postojanje negativnih strana ove metode, koje nisu zanemarljive, i neophodno ih je istaći:

- povećano usitnjavanje uglja na radnom organu kombajna, što u velikoj meri smanjuje njegovu komercijalnu vrednost,
- potreba za visokostručnim osobljem za rad sa mehanizacijom i njeno održavanje,
- velika investiciona ulaganja, usled visoke cene mehanizacije,
- negativan uticaj tektonike - raseda na primenu mehanizovanog dobijanja

uglja, te veće učešće jalovine pri prolasku kroz rasedne zone,
- otežana doprema, montaža i demontaža opreme zbog velikih gabarita,
- otežano održavanje mehanizacije i nabavka rezervnih delova iz uvoza.

Pri primeni ove metode, neophodno je voditi računa o preciznom definisanju tektonike - raseda, geometrije otkopnih stubova i na osnovu toga u skladu sa postojećim uslovima formirati široka čela što većih dužina i postići što veću dužinu otkopnog stuba, čime se obezbeđuje bolji kontinuitet otkopavanja i smanjuju gubici zbog premeštanje otkopne mehanizacije.

Osim toga, vrlo je bitno uporedo sa otkopavanjem izvoditi radove na pripremi - razradi novih otkopnih stubova u otkopnom polju, odnosno ležištu, radi ostvarenja kontinuiteta proizvodog procesa.

ZAKLJUČAK

Rezerve uglja ležišta „Petnjik“, koje iznose približno 10 miliona tona kvalitenog mrkog uglja uslovljavaju neophodnost da se proizvodnja u Rudniku „Berane“ što pre ponovno pokrene. Pri tome treba imati u vidu sva iskustva stečena u prethodnom periodu, i odabrati tehnologiju rada koja je usklađena sa rudarsko-geološkim uslovima ležišta, a posebno izraženom tektonikom. Takođe, potrebno je imati u vidu sve prednosti i mane ranije primenivanih tehnologija rada i naći najbolja rešenja za prevazilaženje nedostataka. Prethodno opisane karakteristike i brojne pogodnosti primene mehanizovanog širokočelnog otkopavanja uglja treba da budu pravac u kome modernizacija podzemne eksploatacije uglja treba da teži, bez obzira na značajne investicije za nabavku opreme koje su neophodne.

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COMPARATIVE ANALYSIS OF USE THE MECHANIZED HYDRAULIC LONG-WALL FACE COMPARED TO THE CLASSICAL MINING METHODS FOR COAL MINING IN THE PETNJIK PIT OF THE BERANE BROWN COAL MINE, MONTENEGRO

Abstract

Underground mining of brown coal on Petnjik deposit of Ivangrad's coal basin began by the beginning of eighties with opening of Petnjik pit of what use to be Ivangrad Brown Coal Mine. Since the Mine was opened, mainly classical mining methods were used for the coal exploitation, meaning room and pillar methods and classical long-wall face methods. In 2001, for the first time, mechanized long-wall face method with modern hydraulic machinery was introduced as a trial method. Application of this method enabled realization of mass production of brown coal, significantly better working conditions and higher level of work safety provided. Comparative analysis of application of this mining method for exploitation of coal in Petnjik pit of the Berane Brown Coal Mine comparing to classical methods previously applied is shown in this paper.

Key words: *underground coal mining, brown coal, the pit, mining method, mechanized long-wall face.*

INTRODUCTION

The Berane Brown CoalMine is situated in the area of Ivangrad's coal-bearing basin located in the direct periphery of Berane town at 650 to 800 m altitude. Coal mining at the basin started in sixties of the last century in "Budimlja" deposit. Exploitation in this mining district was performed by underground mining methods, at 60 m below surface, while simple opencast mining was used for individual minor outcrops of coal. Exploitation of this deposit was finished by the end of seventies, when investment construction of the new mine in "Petnjik" deposit started, so the production in the same pit started in 1981, which is still working today. As in the "Budimlja" deposit, as later in the "Petnjik", production process was performed in extremely difficult conditions, using classical low-efficiency mining methods, room and pillar methods and classical long-wall face methods, causing frequent stoppages of production.

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In 2001, the "Ivangrad" Mine was privatized for the first time by Company "Gradex-HBP", when modern coal mining technology was first introduced into production process. The effects of implementing the modern mechanization, in our opinion, were extraordinary, but unfortunately, a year after the production was initiated, it was terminated due to problems to find a source of long-term financing, and legal property issues.

Today, the mine in Berane is owned by the company "Balkan Energy" DOO Podgorica, a part of the Greek Restis Group, trying to find the best technological solutions for realization of long-term viable production to valorize the existing reserves of quality brown coal by the best possible way.

GENERAL CHARACTERISTICS

Deposit location

The "Berane" Brown Coal Mine is situated in the area of Ivangrad's coal basin, located in depression plateau of headwaters of River Lim and its tributaries Bistrica and Sušica (left tributary) and streams Budimka, Makva, Brnjica and Kaluđer′s River (right tributary), eastern of Bjelasica, outlined at northern side with Tivran, Kamenica and Jugovina mountains, at southern and south-eastern with Vukodarica, Rašča and Bukovac mountains, and at eastern side with Prosjen and Gradac. The "Petnjik" brown coal deposit, as a part of Ivangrad′s coal-bearing basin, covers around 12 km^2 area.

Communication links are relatively favorable referring to the roadway network mostly by the Lim riverbed. Roads are of second and third order directed in 3 routes, i.e. to north towards Bijelo Polje, to east towards Rožaje and to south towards Andrijevica. Over those routes, communication links with other parts of Montenegro and Serbia are carried out. The nearest railway station is in Bijelo Polje, 35 km distanced from the Mine.

The Berane district is known to have the expressed continental climate, extremely low temperatures and snow cover in winter time significantly complicating work and hampering coal transport and conveyance of row material at the surface.

Geological Characteristics of the Deposit

Gelogical explorations of "Petnjik" deposit were performed back in eighties based on 126 exploratory boreholes, and the valid interpretation of geological and tectonic features of the deposit was done in The Elaborate about Classification, Categorization and Calculation of Coal Reserves in Petnjik Deposit of Ivangrad′s Coal Basin, state on December 31, 2001, made in 2002, where brown coal reserves of the deposit were certified.

The "Petnjik" deposit is characterized with 4 existing coal seams, depending on the locality, from which, constant development in the whole deposit has only the main or the youngest seam. The first and the second underlying coal seam are not developed in the whole deposit, but only locally, while the third one occurs only sporadically. The thickness of the coal series is variable, but in the "Petnjik" deposit it does not exceed 40 m. Dept of occurrence of coal seams is 150 to 200 m. The main coal seam has 3.5-4.8 average thickness locally thickened to 5.0-7.6 in smaller areas. Distance between coal seams is from 0.01 to 1.5 m.

Direct roof of the main seam is marl and very rarely clay and sandy clay. Floor is marl sandy in places. General dip of the seam is in western direction (15%). The seam has a small dip in its western part and it is almost horizontal as a pose to the rest of the deposit where the dip is 10°-25°.

The tectonics of "Petnjik" deposit, same as the entire Ivangrad′s coal basin is very complex. Regarding tectonics the deposit is rugged with numerous faults including the main fault lines and the secondary fault lines. Based on exploratory surface drilling, the deposit is, regarding tectonics, divided into six tectonic blocks. The capital faults strike parallel to each other and they are mainly in N-E and NW-SE strike direction. The throw of the fault wall is over 10 m.

From geological description of "Petnjik" deposit, it can be established that mining and geological conditions in mining blocks of the deposit where excavation is done are rather complicated due to the fact that tectonics is very complex, blocks are rugged with numerous faults, and coal seams have variable thickness. Also, disability of detailed geological analysis of exploitable coal seam and adjoining rocks often requires presumption of tectonic characteristics, which significantly complicates the work and causes problems in mining. Mining works in the "Petnjik" Pit also confirmed a large number of smaller faults presenting great challenge for setting of mining panels and hampering execution of mining works in the pit.

Coal Reserves and Quality

All explored geological reserves, according to above mentioned Elaborate from 2002, are classified in balance, out of balance and potential reserves. Among calculated coal seams in "Petnjik" deposit, the best studied is the main coal seam. Beside balance categories A, B and C_1 in the coal seam, there are out of balance reserves B and C_1 and potential C_2. The first underlying coal seam is less studied and balance and out of balance reserves of C_1 categories and potential reserves of C_2 category are calculated for the seam. The second and the third coal seam are spread locally.

The coal reserves by seams and by categories in the "Petnjik" deposit are shown in Table 1. [1]

Table 1. *Coal reserves of the "Petnjik" deposit*

Category of reserves	Balance geological reserves (t)	Mining reserves (t)
Main coal seam (B + C_1)	12,715,730	10,172,583
The first underlying coal seam (C_1)	450,215	360,172
Total (B + C_1)	13,165,945	10,532,756

The "Petnjik" deposit coal is of dark brown to black color with following characteristics: does not stain, with dark-brown scratching, rough fracture, rarely shelly with clear vertical divisibility. Based on petrography tests, as well as chemical and technological tests, coal falls into the group of brown-lignite coals, i.e. hard brown coals. Based on the results from laboratory tests of run of mine coal, the quality of the main seam coal is obtained, so showed shown in Table 2. [1]:

Table 2. *Quality of the main seam coal*

Moisture %	Ash %	S %			Coke %	C/fix	Combustible matters	Evaporable matters	UCV KJ/kg	LCV KJ/kg	Unit mass t/m^3
		total	com-bust.	in ash.							
22.58	22.55	1.61	1.03	0.62	46.78	23.83	54.18	30.76	15.13	13.02	1,30

MINING CONDITIONS IN THE "PETNJIK" PIT

The "Petnjik" pit is opened centrally with haulage and ventilation mutually shaft connected with underground collecting station. Opening roadways, main transport and main ventilation road, are made in the floor of coal seam, parallel, at 30 m mutual distance. Development of those roads undertook the coal reserves of "B", "C" and "D" block. The basic development roads opened mining block "B", while each panels of the block is developed with two gates (haulage and ventilation) connected at their ends with a working face.

System of conveying belts of 800 mm width is installed in the main roads. Transport from working faces is carried out by the chain conveyors.

Ventilation of the pit is of flow type. It is a simple parallel flow network, where fresh air enters the pit through haulage shaft and further distributed through the system of main transport roads to worksites. Used air is lead through the system of main ventilation roads to ventilation shaft, and by the main fan, installed on the surface, out to outer atmosphere.

The "Petnjik" Pit of "Ivangrad" Brown Coal Mine, although categorized as gassy mine, features very favorable working conditions. Concentrations of methane are within permissible limits, and there were no occurrences of mine fires, water outbursts, rock bursts or similar accidents in previous work.

COAL MINING

In the previous phase of coal mining, i.e. before introduction of mechanized long-wall face method, excavation in the "Petnjik" pit was done by non-mechanized mining methods with drilling and blasting works. Mainly two methods were used: room and pillar method – so called the "Brazda" method and long wall face mining method with friction props and steel bars, where loading of coal at faces is hand loading or scraper loading.

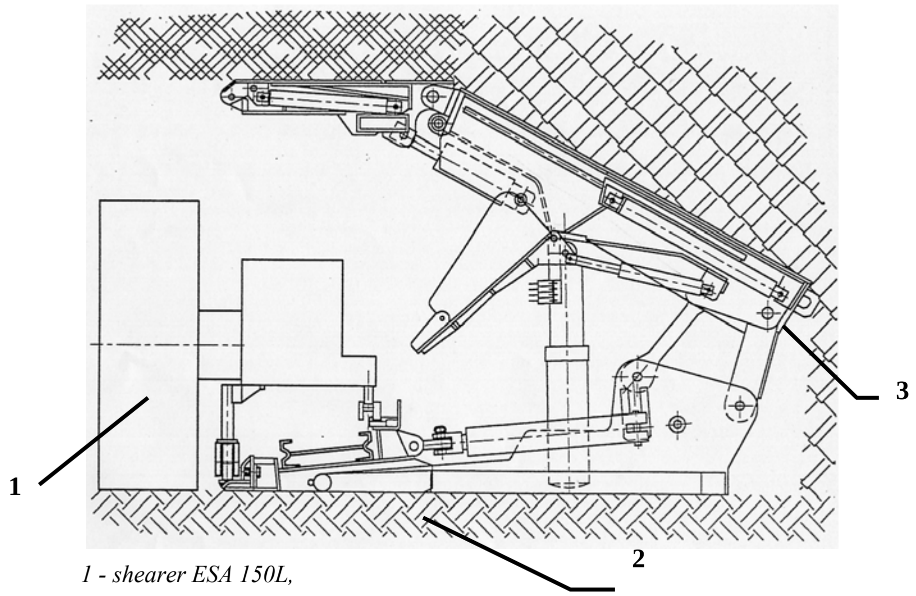
By the end of 2002, in the part of "B" block of the "Petnjik" pit, for the first time, smaller mechanized long-wall face as a trial worksite was installed. Introduction of mechanized long-wall face mining method implied the use of hydraulic shield support type BMV-1 and BMV-1M, shearer type ESA-150L for mechanized cutting coal-winning, and mechanized conveyance of coal from long-wall face with three-chain flight conveyor type TH 604/F. All listed equipment is made in Slovakia.

- **Shield support units type BMV-1 and BMV-1M** [2], possesses following technical characteristics**:**

- maximal height of the unit	3,400 mm
- minimal height of the unit	2,320 mm
- width of the unit	1,500 mm
- pace of the unit	700 mm
- number of props	2 pieces
- pressure force of props	1,216 kN
- resistance of support	1,296 kN
- floor pressure	0.52 MPa
- unit pushing force	391 kN
- conveyor pushing force	251 kN
- operating pressure of hydraulic pumps	32 MPa
mass of the unit	11,500 kg
hole for releasing roof coal	
- width	850 mm
- length	1100 mm

The unit consists of the following basic parts: foot frame, upper shield, bar with column and hydraulic prop.

The schematic review of mechanized long wall face with hydraulic shield support type BMV-1 [3] is shown in Figure 1.



1 - shearer ESA 150L,
2 – face conveyor TH 604/F,
3 – hydraulic support unit BMV-1M.

Figure 1. *Schematic view of the mechanized long wall face*

- **Shearer type ESA-150L** for cutting coal-winning and loading of excavated material, has following characteristics:

- engine power 150 kW
- shearer height from ground 1,650 mm
- working body diameter 1,600 mm
- max cutting height 3,330 mm
- max cutting below ground level 460 mm
- body RPM 39 r/min
- traction force 250 kN
- weight 16,000 kg
- voltage 3 x 1000 V
- pushing system without chain

- **Three-chain flight conveyor type TH 604/F** for conveyance of coal from face with the following characteristics:

- capacity 360 t/h
- length of transporter depending on inclination 70-250 m
- engine power 100 kW
- speed 0.6-1.07 m/s
- width 642 mm
- distance between jibs 1,032 mm
- distance between chains 120 mm

- **Hydraulic generator type HA 80/320** consists of the following parts:
  - emulsion tank (water-oil in 97-98 % : 2-3% ratio);
  - generator consisting of:
    - electromotor of 55 kW power for starting the three-piston hydraulic pump,
    - electromotor of 3 kW for starting the emulsion pump,
    - hydraulic three-piston pump,
    - emulsion pumps,
    - emulsion filters
    - hydraulic pipeline,
    - device for control of operating pressure.

Operating pressure of generator is 320 MPa. Generator operates in the closed system. Beside the operating hydraulic generator, it is necessary to install a backup generator, too.

Technology of long-wall face mining

Application of described machinery for long-wall face coal mining depending on a seam thickness, coal winning can be done by two ways:

- cutting coal-winning in long wall face without blasting,
- cutting coal-winning in face and blasting of roof part of the seam. [4]

Long wall mechanized mining with cutting coal-winning in the face and roof caving is used for seems up to 4.5 m thickness. Digging height for cutting in face is limited by the height of hydraulic shield support.

Mechanized long-wall mining with cutting coal-winning in the face and blasting of coal from the roof, with roof caving, is applied for mining of layers or their parts with greater thickness. The essence of the method is to excavate coal in front pillar with shearer, while coal-winning from the roof is done using blasting.

The main stages in technological process of coal mining with mechanized longwall face are: coal winning, loading and conveying of excavated material, supporting of the face and caving of the goaf behind the working face.

DISCUSSION

Based on production process records from previous period, comparative analysis of getting output is made for all three analyzed mining methods: room and pillar method called the “Brazda”, classical long wall face and mechanized long wall method. The achieved outputs (in tones per man shift) by months are compared on a diagram and shown in Figure N^{o} 2.

It is clear on a diagram that the achieved outputs for classical methods are about 5tpms, while the achieved outputs for mechanized long wall mining are three to four times higher, i.e. reach values up to 24t pms. Thereby we should bear in mind that long wall face used for mining presented trial mining of remind the “B” block reserves, so the face lengths were limited (35 m) and worked in harsh conditions due to expressed tectonics and influence of goafs, so we think that mining in a new block, in favorable mining and geological conditions and greater face length, those outputs would reach much higher values.

Overall analysis of all effects of usage the mining method with mechanized long wall face, based on experiences from trial excavation in the “B” block of the “Petnjik” pit, a the conclusion was made that the stated method is far more productive that previously used non-mechanized methods and that its application shows numerous advantages, as regarding working outputs, working conditions in general and work safety.

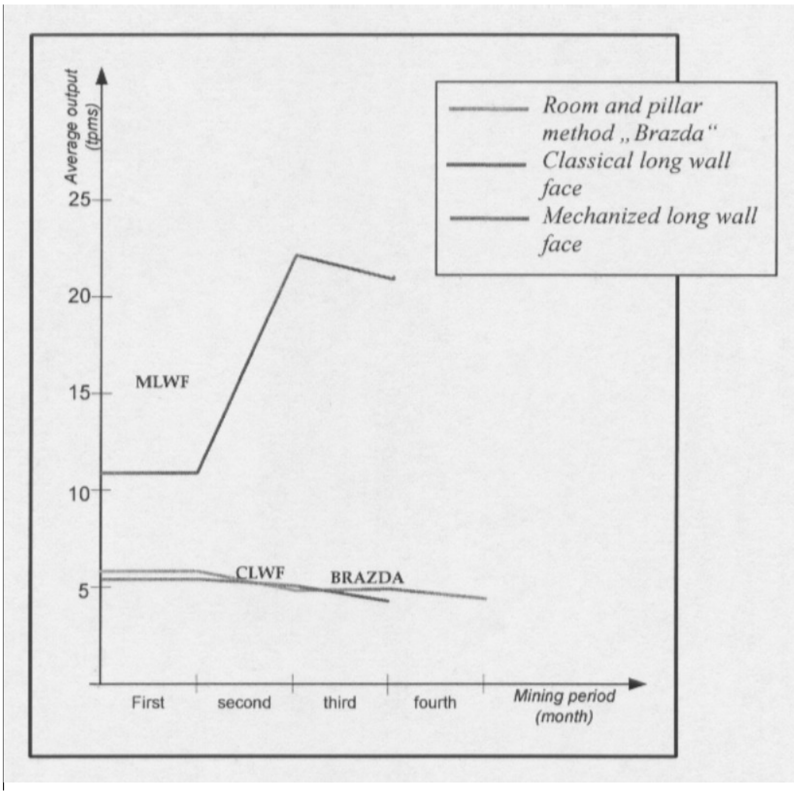


Figure 2. *Diagram of output for excavation with different mining methods*

Comparing to earlier mining methods, the mechanized long wall mining has series of advantages as follows:

- higher recovery ratio of coal substance;
- working in completely supported space;
- worksite has two gates;
- ventilation is of a flow-type;
- incomparably better comfort and safety at work;
- drilling and blasting works are reduced to minimum (only for roof coal winning);
- manual work is reduced;
- mechanized long wall face advance dynamics provide stability of gates;
- much higher productivity (over 20 tpms achieved outputs);
- significantly reduced number of workers at face;
- less danger from endogenous fire;

Inevitably there are negative aspects of this method, not negligible ones, and it is necessary to emphasize them:

- increased coal shredding at cutting body of shearer, which greatly reduces its commercial value;
- need for highly qualified staff to operate mechanization and service it;
- large investments due to a high mechanization purchase prices;
- negative impact of tectonics – faults to application of mechanized coal winning, so higher dirt percentage when going through fault zones;
- difficult transport, assembling and disassembling for large dimensions,

- difficult maintenance and procurement of spare parts from abroad.

The use of this method requires precise defining of tectonics – faults, geometry of mining panels and then, based on that and according to the existing conditions, form the longest possible faces and achieve maximum length of panel, providing better mining continuity and decreasing losses caused by mining mechanization shifting.

Besides that, along with excavation, it is very important to perform development works – open new panels in the mine field, or deposit, to achieve production process continuity.

CONCLUSION

Coal reserves of the "Petnjik" deposit, to the amount of approximately 10 million tones of quality brown coal are conditions to the necessity to activate production in the "Berane" Mine as soon as possible. To do that, all gained experiences in previous period have to be born in mind and choosing the work technology compliant with the mining and geological deposit conditions, especially with the expressed tectonics. Also,all advantages and disadvantages of previously used work technologies should be considered and to find the best solutions to overcome the shortcomings. Previously described characteristics and numerous benefits of mechanized long wall mining use should be the objective that modernization of coal exploitation should pursue, regardless the high equipment purchase prices that are inevitable.

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MOGUĆNOSTI I POTREBA ZAVRŠETKA OTKOPAVANJA RUDNOG TELA „TILVA ROŠ" U JAMI BOR***

Izvod

Nakon akcidentne situacije u borskoj jami, kada je došlo do provale vode i mulja, kojom prilikom je smrtno stradao jedan rudar, došlo je do obustave eksploatacionih radova u rudnom telu „Tilva Roš". Od strane Ministarstva rudarstva i energetike postavljen je zahtev da se uradi Glavni rudarski projekat trajne obustave radova u rudnim telima „Tilva Roš" i „P2A". Međutim, u pomenutim rudnim telima ostala je određena količina rude, koja bi mogla biti predmet daljeg otkopavanja pod uslovom da se obezbede potrebni sigurni uslovi za rad u rudnim telima. To je od posebnog značaja i zbog toga što RTB Bor nema trenutno dovoljno pripremljene rude za intenziviranje njenog dobijanja, što je od izuzetnog značaja za trenutnu situaciju u preduzeću, posebno ako se ima u vidu povoljna situacija na svetskom tržištu metala.

U radu se govori o mogućnostima nastavljanja eksploatacionih radova uz ispunjenje potrebnih uslova zaštite jame od prodora vode sa napuštenog površinskog kopa u Bou.

Ključne reči: *ležište bakra Bor, podzemna eksploatacija, završetak eksploatacionih radova, metoda otkopavanja.*

1. UVOD

U borskoj jami, jednoj od proizvodnih jedinica Rudnika bakra Bor, otkopavanje se obavlja podzemnim načinom u nekoliko rudnih tela. Najveća proizvodnja rude data je iz rudnog tela „Tilva Roš", koje se nalazi u centralnom delu Borskog ležišta. Ono se ujedno nalazi neposredno ispod dna površinskog otkopa, (sl. 1), u kome je prestalo otkopavanje, a na dnu je došlo do formiranja dve veće odvojene akumulacije vode, koje ozbiljno ugrožavaju bezbednost rada u jami. U februaru 2007. godine došlo je do prodora vode i mulja u jedan od otkopnih podetažnih hodnika, tom prilikom je jedan rudar nastradao, a voda sa muljem je potopila i niže horizonte jame. Dalje otkopavanje je prekinuto, a od strane Rudarske inspekcije dobijen je nalog da se ozbiljno analizira nastala situa-

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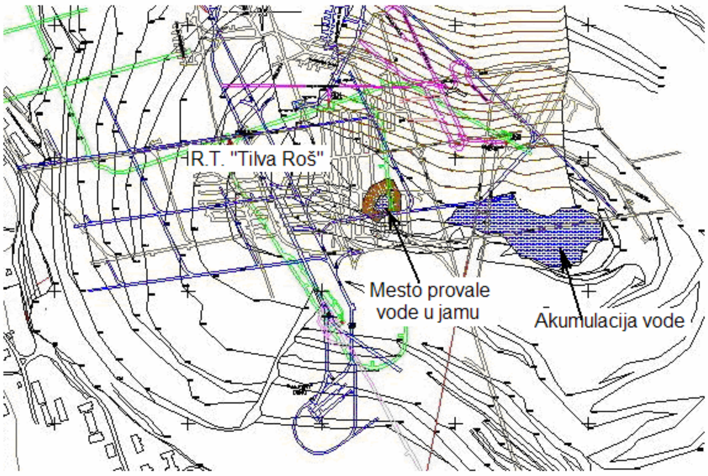
cija i predloži rešenje za dalji rad. Ekspertizu su sačinili stručnjaci Rudarsko-geološkog fakulteta u Beogradu, koji su predložili da se:

- pristupi postupnom zatvaraju rudnih tela „Tilva Roš“ i „P2A“ uz obavezu izrade Glavnog rudarskog projekta za trajnu obustavu radova na eksploataciji mineralnih sirovina;
- eksploatacija preostalih rudnih rezervi može vršiti samo u slučaju sigurne sanacije akumulirane vode kroz regulisanu drenažu ili ispumpavanje vode sa dna površinskog otkopa.

Međutim, na osnovu raspoloživih podataka o preostalim rudnim rezervama, može se utvrditi da ostaju, doduše manje količine rudnih rezervi rude, čije bi otkopavanje bilo korisno, pre svega zbog toga što Jama nema trenutno pripremljene rezerve rude za otkopavanje u drugim rudnim telima. S druge strane, okolnosti u Rudniku bakra Bor, kao i situacija na svetskom tržištu metala su takve da bi bilo vrlo poželjno trenutno povećanje proizvodne rude bakra, pogotovo ako je ona znatno boljeg kvaliteta od one koja se dobija na površinskom kopu V. Krivelj. Zbog toga se ovom prilikom daju neka razmišljanja o mogućnostima makar delimičnog otkopavanja preostalih rezervi rude u rudnom telu „Tilva Roš“:



Sl. 1. *Situacija površinskog kopa u Boru sa podzemnim rudarskim radovima u rudnom telu „Tilva Roš“ i naznačenim mestom provale vode i mulja u jamu*

Kao što se sa slike vidi, mesto provale vode se nalazi u jugozapadnom delu rudnog tela, gde je otkopavanje vršeno na podetažama na nivou K-31 m. Sagledavanje mogućnosti otkopavanja drugih delova rudnog tela moglo bi se obavljati pod sledećim uslovima:

- da se, kako je i u urađenoj analizi nastale situacije u rudnom telu predloženo, izvrši potpuno odstranjivanje akumulirane vode bilo ispumpavanjem sa površine, ili drenažom kroz postojeće jamske prostorije, pri čemu bi se proces pumpanja vode ili drenaže obavljao permanentno za svo vreme eksploatacionih radova do sada primenjivanom metodom podetažnog zarušavanja – „švedska varijanta“,
- da se otkopavanje rudnog tela obavlja istom metodom, samo u delovima koji nisu neposredno ispod akumulacija vode, niti su ugroženi mogućim pronicanjem vode kroz postojeće pukotinske sisteme,
- da se otkopavanje obavlja izvan zone uticaja akumulacija vode, i to tako, da se promeni način otkopavanja,

odnosno primeni otkopavanje bez zarušavanja rude i pretećih stena.

U ovom radu se prevashodno govori o trećoj mogućnosti, odnosno o ideji da se otkopavanje obavlja metodom otkopavanja sa održanjem otkopnog prostora (varijanta metode otkopavanja sa otvorenim otkopima). I u ovom slučaju javljaju se dve osnovne mogućnosti:

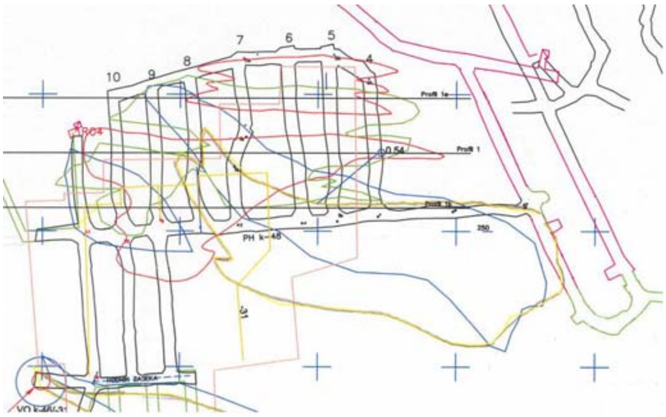
1. Otkopavanje delova rudnog tela u kojima je izvršena delimična priprema za otkopavanje primenjenom metodom podetažnog zarušavanja trebalo bi da se vrši uz maksimalno moguće korišćenje izrađenih pripremnih prostorija;

2. U delovima rudnog tela gde nisu izrađene pripremne prostorije, u potpunosti primeniti drugačiji sistem otkopavanja sa odgovarajućom izradom pripremnih prostorija. I u ovom slučaju se računa sa primenom metode otkopavanja sa otvorenim otkopima.

Ukoliko bi se pokazala zadovoljavajuća stabilnost podzemnih otkopa i ne dođe do obilnijeg dreniranja vode u otkope, otkopavanje bi se moglo primeniti i u delu rudnog tela ispod dna površinskog kopa.

Otkopavanje severnog dela rudnog tela u kome je izvršena priprema na podetaži K.-46 m

U severnom delu rudnog tela „Tilva Roš" izvršena je priprema za otkopavanje izradom otkopnih hodnika br 4 do br. 10, (slike 2 i 3).Osnovni podetažni hodnik izrađen je iz servisnog niskopa SN-16/-46 iz koga je nastavljena izrada niskopa prema rudom telu „T". Hodnik je izrađen po pravcu istok-zapad, a iz njega su izrađeni otkopni hodnici u pravcu severa. Sa plana ovih hodnika, prikazanog na slici 2, uočava se da se hodnici ne izrađuju sa potrebnom preciznošću (rastojanje hodnika 14 m), što je izuzetno važno kod primenjene metode otkopavanja.

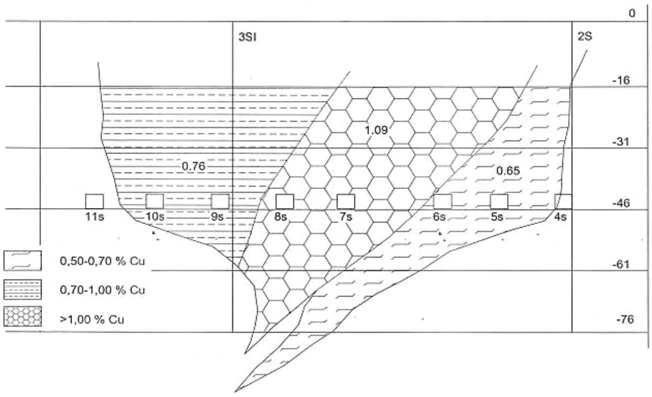


Sl. 2. *Plan izrađenih podetažnih hodnika u severnom delu rudnog tela*

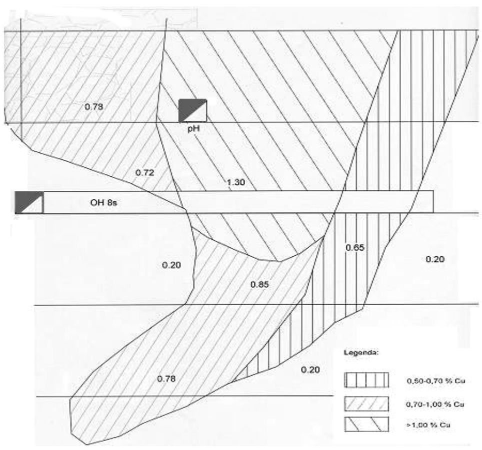
Ispod nivoa podetaže na K-46 m rudno telo se javlja na dubini još najmanje 30 m, tj. do nivoa podetaže na K-76 m, što odgovara nivou XV horizonta u borskoj jami. Međutim, ovaj deo rudnog tela se značajno sužava, što će ograničiti broj otkopa u kojima se može otkopavati po predlogu datom u ovom radu. To takođe znači da je za potrebe projektovanja metode otkopavanja neophodno precizno definisanje kontura rudnog tela, što se svakako može dobiti od strane geološke službe rudnika.

Idejno rešenje metode otkopavanja prikazano je na slici 5. Da bi se iskoristili već izrađeni pripremni hodnici za metodu podetažnog zarušavanja, predloženo je formiranje otkopa tako da se hodnici nalaze po sredini bloka i služiće za bušenje i miniranje rude.

Metoda otkopavanja sa otvorenim otkopima predviđa ostavljanje sigurnosnih stubova između otvorenih otkopa, zbog čega je predložena širina otkopa od 14 m, koliko bi iznosila i širina sigurnosnih stubova. Ona je uslovljena već postojećim parametrom horizontalnog rastojanja između podetažnih hodnika koje je 14 m. Visina otkopnih blokova bila bi promenljiva s obzirom na nepravilnu konturu rudnog tela, koje se sa dubinom postupno sužava. Budući da su hodnici na postojećem podetažnom nivou na K-46 m već urađeni, to je i lokacija otkopnih blokova njima definisana. Predlaže se da otkopni blokovi budu tako raspoređeni da se njima zahvata najveća količina rude.



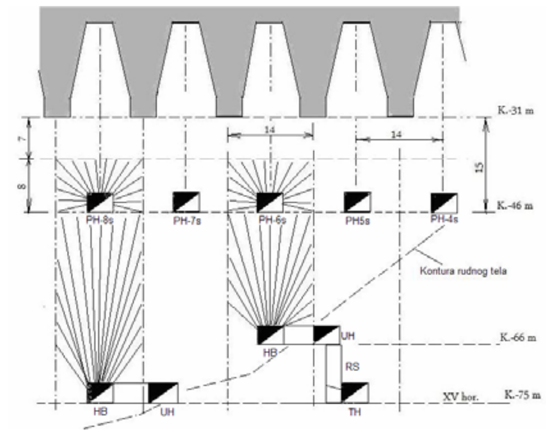
Sl. 3. *Uzdužni geološki presek 1 u severnom delu rudnog tela „Tilva Roš“*



Sl. 4. *Poprečni presek kroz hodnik OH 8s*

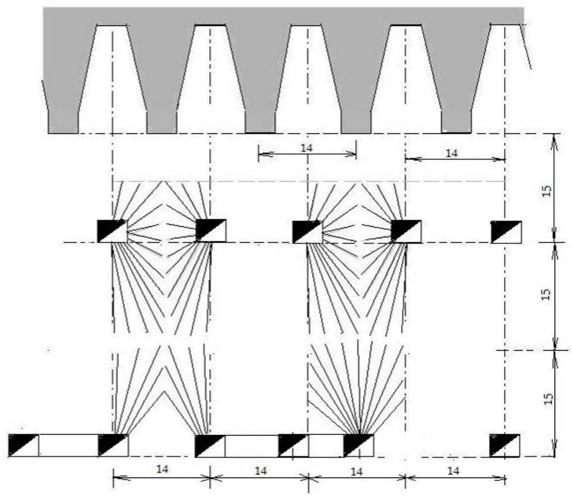
Najniži deo rudog tela „Tilva Roš“ javlja se u preseku koji odgovara podetažnom hodniku PH-8s, što se na prikazanom profilu na sici 3 vidi. Delovi rude širine, takođe po 14 m, koji odgovaraju hodnicima PH-7s i PH-9s činili bi sigurnosne stubove za prikazani otkopni blok.

Konstrukcija pripreme za predloženu metodu otkopavanja je usvojena tako da se ne predviđa izrada pripremnih hodnika na nivou sledeće podetaže na K.-61 m, već bi se hodnici bušenja, a i utovarni hodnici radili na nivou K-76 m, odnosno na nivou XV horizonta.



Sl. 5. *Predlog I varijante metode otkopavanja sa ostavljanjem otvorenih otkopa za otkopavane najnižih delova rudnog tela „Tilva Roš“*

Alternativni predlog formiranja otkopnih blokova prikazan je na slici 6. Po njemu, otkopi bi se formirali tako da su postojeći podetažni hodnici u bokovima otkopa, pri čemu je širina blokova identična prethodnim. Ovakav raspored otkopnih blokova nameće drugčiju šemu rasporeda minskih bušotina, koje se, međutim, mogu bušiti i sa drugčijim rasporedom. Izbor šeme rasporeda minskih bušotina u otkopnim blokovima zavisiće od raspoložive opreme za bušenje i njenih mogućnosti. Misli se to na već postojeće mašine za bušenje Simba-H253, koje su korišćene za bušenje dubokih minskih bušotina kod metode podetažnog zarušavanja („Švedska varijanta“). Ukoliko bi se pristupilo nabavci novih mašina za bušenje, njihova konstrukcija treba da omogući bušenje minskih bušotina sa lepeznim rasporedom (0 – 360^{o}).



Sl. 6. *Predlog II varijante metode otkopavanja sa otvorenim otkopima kod koje su podetažni hodnici za bušenje u bokovima otkopnog bloka*

Moguće su svakako i drugčija tehnička rešenja kako konstrukcije metode otkopavanja, tako i rasporeda hodnika za bušenje i rasporeda minskih bušotina u pojedinim redovima. To je i inače isuviše veliki broj mogućnosti za relativno male količine preostalih rudnih rezervi u rudnom telu. Naime, pojedini otkopni blokovi bi se razlikovali i po mogućoj njihovoj visini , a što je uslovnjeno utvrđenim konturama rudnog tela u svakom od otkopnih blokova.

U otkopnom bloku ukupne visine 36 m (od K.-76 do K.-38 m), površina bloka iznosi:

$$P_{bl} = 14 \text{ x } 36 - 4 \text{ x } 3{,}5 - 14 \text{ x } 3{,}5 - 0{,}5 \text{ x } 10 \text{ x } 5 = 504 - 14 - 49 - 25 = 416 \text{ m}^2$$

Na 1 m dužni bloka količina rude iznosi:

$$Q_{r1} = 416 \text{ m}^2 \text{ x } 2{,}8 \text{ t/m}^3 = 1.164{,}8 \text{ t/m}$$

Pri bušenju redova minskih bušotina na međusobnom rastojanju 2,5 m, količina rude koja se dobija jednim miniranjem iznosi:

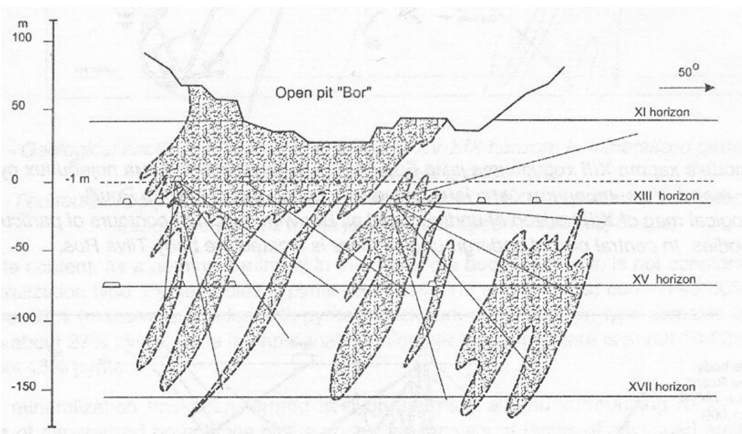
$$Q_r = 1.164{,}8 \text{ x } 2{,}5 = 2.912 \text{ t}$$

Ova količina rude može biti utovarana u toku narednih 6 – 10 smena, (detaljna dinamika bušenja i utovara rude daće se u projektu otkopavanja).

2. OTKOPAVANJA DELOVA RUDNOG TELA U KOJIMA NIJE VRŠENA PRIPREMA

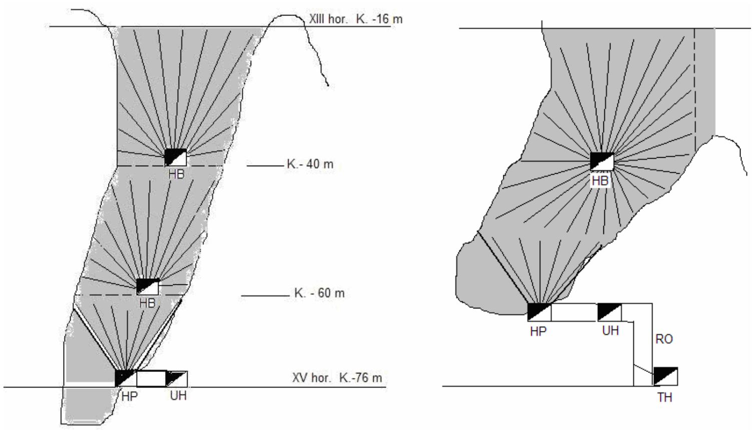
Donji deo rudnog tela „Tilva Roš" završava se u vidu većeg broja suženih delova, koji asociraju na moćne rudne žile, (slika 7.) Takvi delovi rudnog tela nisu pogodni za primenu metode podetažnog zarušavanja, budući da se između pojedinih orudnjenih delova javljaju neorudnjeni delovi masiva. To bi, za slučaj primene metode podetažnog zarušavanja uslovljavalo pojavu enormno velikog osiromašenja, ili bi se moralo primeniti selektivno otkopavanje, sa većim brojem zaseka podetaža, a što bi usporilo dinamiku otkopavanja i povećalo gubitke i osiromašenje rude.

Zbog toga, kao što se i sa slike 7 vidi, primena podetažnog zarušavanja ne bi bila efikasna, drugim rečima donji deo rudnog tela „Tilva Roš" nije pogodan za primenu projektovane metode otkopavanja.



Sl. 7. *Poprečni presek donjeg dela rudnog tela „Tilva Roš"*

Povoljnije rešenje bi se moglo naći u selektivnom otkopavanju pojedinačnih orudnjenih delova, čak i pri primeni neke od varijante sa zarušavanjem rude, o čemu se ovde ne govori, budući da se predlaže otkopavanje bez narušavanja natkopnog masiva. Na slici 8 prikazana su dva dela rudnog tela, za koje je predloženo selektivno otkopavanje u otvorenom otkopu sa podetažnim obaranjem rude. Osnovni problem kod ovakvog otkopavanja je povećan obim pripreme, koji odgovara situaciji otkopavanja rudnih tela relativno manje moćnosti (10 – 20 m), budući da se, prema postojećoj klasifikaciji rudnih ležišta radi o srednjoj moćnosti (< 20 m).



Sl. 8. *Predlog metode podetažnog okopavanja u otvorenim otkopima raspoređenim po pružanju pojedinih delova rudnog tela*

Kao što je predlogom metode prikazano, za svaki pojedinačan deo radi se pristupni (transportni hodnik (TH), a ako otkopavanje počinje sa nivoa horizonta, tada je ovaj hodnik i utovarni hodnik. U samom rudnom telu izrađuje se, u najnižem njegovom delu, hodnik podsecanja (HP) iz koda se bušenjem lepeznih minskih bušotina i miniranjem formira „tranšeja“ ili podsek do koga se iz utovarnog hodnika na međus-obnom rastojanju 8 – 10 m izrađuju kratki prečni hodnici – utovarne niše za utovar rude iz podseka. Na višim nivoima, zavisno od veličine dela rudnog tela koji se otkopava, izrađuju se podetažni hodnici za bušenja (HB) iz kojih se buše „lepeze“ minskih bušotina za obaranje rude. Podetažni hodnici se izrađuju iz pripremnog uskopa (niskopa), koji se radi izvan rudnog tela, a pre početka otkopavanja potrebno je izvršiti zasecanje bloka izradom slobodnog prostora. On se izrađuje tako što se prethodno izradi okno zasecanja čijim se proširenjem formira slobodan prostor na početku otkopne komore.

Posle obavljenog zasecanja otkopa, pristupa se bušenju minskih bušotina i obaranju rude njihovim miniranjem. Minirana ruda pada u dno bloka (podsek ili tranšeju) gde se utovarno-transportnom mašinom na dizel pogon utovara i prevozi do pretovarnog mesta odnosno rudnog okna ako je utovarni hodnik iznad nivoa horizonta.

Za pretpostavljeni otkopni blok, kao na slici 10, i pretpostavljenu širinu od 15 m i dužinu bloka min 30 m, količina rude koja se dobija u pojedinim fazama iznosi:

- pri izradi podseka
  0,5 x (4+15) x 12 x 2,8 = 319,2 t/m,
  odnosno ukupno 9576 t,
- pri obaranju rude iz podetaža
  15 x 44 x 30 x 2,8 = 55.440 t.

Navedena količina rude se dobija sa minimalnim gubicima i praktično bez osiromašenja. Pri smenskom kapacitetu na utovaru i prevozu rude od 300 t/smenu, iz otkopavanog dela rudnog tela obezbedilo bi se dobijanje rude u toku 216 smena ili 72 dana.

3. ZAKLJUČAK

Rudno telo „Tilva Roš“ završava se između XIII i XVII horizonta u vidu većeg broja odvojenih orudnjenih delova različitog oblika i moćnosti. Između njih su neorudnjeni delovi stenskog masiva, pa ta okolnost čini primenu projektovane metode podetažnog zarušavanja neracionalnom. S druge strane, postoji ograničenje za dalju primenu metode otkopavanja sa zarušavanjem rude, što je uzrokovalo predlog za promenu načina otkopavanja, koji je predložen u ovom radu. U rudnom telu nisu preostale velike količine rudnih rezervi, međutim, nepovoljna situacija sa trenutnim stanjem pripremljenosti drugih rudnih tela za otkopavanje čini da se na ovaj način može u određenom vremenskom periodu obezbediti proizvodnja rude sa relativno dobrim kvalitetom s obzirom na trenutnu njenu vrednost uslovljenu cenama metala na svetskom tržištu.

Novopredloženi način otkopavanja nije suštinski različit, a predloženo podetažno otkopavanje u otvorenim otkopima je ranije već primenjivano u razmatranom rudnom telu i postoji mogućnost primene opreme za bušenje, miniranje i utovar rude sa kojom Jama raspolaže ili je već imala u upotrebi. Izrada projekta otkopavanja po predloženoj metodi zahteva precizno definisanje granica pojedinih delova rudnog tela, što bi bio zadatak geološke službe rudnika, a postoji verovatnoća o potrebi paralelnog doistraživanja rudnog tela u navedenom segmentu.

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OPPORTUNITIES AND NEED FOR COMPLETION OF MINING THE ORE BODY TILVA ROS IN THE JAMA BOR***

Abstract

After accident, in the Bor pit, when there was an inrush of water and mud, at which time it killed one miner, the exploitation works were suspended in the ore body "Tilva Ros". The Ministry of Mines and Energy set up a request to do the Main mining project for permanent suspension of works in the ore bodies Tilva Ros and P_2A. However, in these ore bodies remained a certain amount of ore that could be subjected to further excavation under the condition of providing the necessary conditions for safe operation of the ore bodies. This is of special importance because RTB does not have prepared enough ore for intensification of its production, what is of great importance for the current situation of the company, especially regarded to the favorable situation on the world metal market.

This paper discusses the possibilities of continuing the exploitation works subjected to the fulfillment of necessary conditions for protection of pit against water pit from the abandoned open pit in Bor.

Key words: *Bor copper deposit, underground mining, finalization of exploitation of works, mining method*

1. INTRODUCTION

In the Jama Bor, one of the production units in the Copper Mine Bor copper, excavation is carried out in the underground in several ore bodies. The greatest production of ore is from the ore body Tilva Ros, located in the central part of the Bor deposit. It is also located just below the bottom of the open pit (Figure 1), in which the excavation was suspended, and at the bottom there was a formation of two or more separate reservoirs of water, which seriously threaten the safety of the pit. In February

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2007, there was an influx of water and mud in one sulevele drift and one miner died and water with mud sank lower horizons of the pit. Further excavation was stopped, and the Mining Inspectorate requested for seriously analyze the situation and to proposed a solution for future work. Expertise was prepared by the experts of Mining and Geology Faculty, University of Belgrade, who suggested that:

- Access to gradual closing of the ore bodies Tilva Ros and P_2A with the obligation of making the Main mining project for permanent suspension of works on the exploitation of mineral resources;
- Exploitation of the remaining ore reserves could be carried out only in a case of safe rehabilitation of accumulated water through the regulated drainage or pumping of water from the bottom of the open pit.

However, based on the available data on the remaining mineral reserve, it can be determined that the small amounts of ore reserves remain in a case of safe rehabilitation, whose excavation would be useful, primarily because Jama doe not have the prepared ore reserves for mining in the other ore bodies. On the other hand, the circumstances of the Copper Mine Bor and situation on the world metal market are such that it would be highly desirable to increase the current production of copper ore, especially if it is of much better quality than the one obtained at the open pit Veliki Krivelj. Therefore, on this occasion, some thoughtare given on the possibilities of at least partial excavation of the remaining ore reserves in the ore body Tilva Ros.

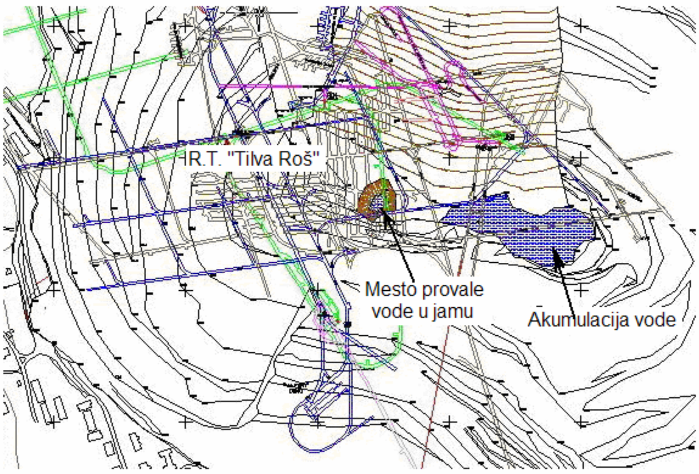

Figure 1. *Situation in the open pit in Bor with the underground mining operations in the ore body Tilva Ros and marked place of water and mud influx in the pit*

As it is seeen from Figure 1, the place of water influx is located in the south-western part of the ore body, where the excavation was carried out on the sub-benches at K-31 m. Recognition the possibility of excavation the other parts of ore body could be carried out under the following conditions:

- as it was proposed in the analysis of situation in the ore body, to carry out the complete removal of accumulated water both by pumping from the surface, or drainage through the existing underground drifts, where the process of water pumping or drainage would be continuously carried out for the entire period of exploitation works by previously used method of sublevel caving - "Swedish variant",

- to perform the excavation of the ore body by the same method, only in the parts that are not directly under the accumulation of water or the potential danger of water penetration through the existing fracture systems,
- to perform the excavation outside the influence of water accumulations such as to change the mining method, that is to use the excavation without caving of ore and associated rocks.

This paper is primarily about the third possibility, that is the idea on carried out excavation by mining method with maintenance the excavation space (a variant of excavation method with the open stopes). In this case there are two basic options:

1. Excavation of the parts of ore body with a partial preparation for the excavation of the applied sublevel caving should carried out with maximum possible use of prepared rooms;
2. In parts of the ore body where the preparation rooms are not made, to apply completely different system of mining with development of preparation rooms. And in this case, it is calculated with the use of mining method with the open stopes.

If the stability of underground excavations would be satisfactory and there is no extensive water drainage in the stopes, excavation could be also applied to the part of tore body below the bottom of the open pit.

Excavation of the northern part of ore body prepared in the sublevel K 46 m.

In the northern part of the ore body Tilva Ros, the preparation for excavation was made by development the mining drifts No. 4 to No. 10, (Figures 2 and 3). Basic sublevel drift is made from the service incline SN-16/-46 from which the development of drift was continued to the ore body "T". Drift is made in the east-west direction, and excavated drifts are developed from it towards the north. It is seen from the plan of drifts, shown in Figure 2, that the drifts are not made with the required precision (distance of a drift14 m), what is extremely important in the applied mining method.

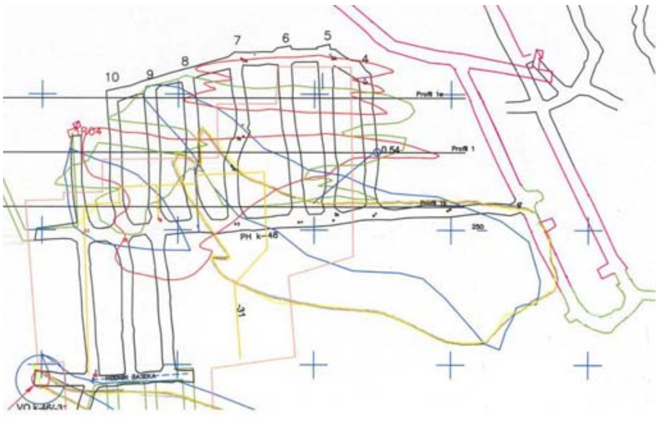

Figure 2. *Plan of developed drifts in the northern part of the ore body*

Below the level of sub bench at K-46 m, the ore body occurs at depth of at least 30 m, i.e. up to the level of sub bench at K-76 m, corresponding to the level of horizon XV in the Bor pit. However, this part of the ore body is significantly narrower, which will limit the number of stopes which can be excavated by the given proposal in this paper. It also means that the needs of design methods of excavation necessary to precisely define the contours of the ore body, which can certainly be obtained from the geological service of the mine.

Conceptual design of mining methods is shown in Figure 5. In order to take the advantage of already made preliminary drifts for sublevel caving method, the formation of slope was proposed such as the drifts are in the middle of the block and they will be used for drilling and blasting of ore. Mining method with open stopes predicts leaving of security pillars between the open stopes; due to this the predicted width of slope is 14 m, as the width of security pillars will be. It is conditioned by the existing parameter of the horizontal distance between sublevel drifts, which is 14 m. Height of mining blocks would be variable due to the irregular contour of the ore body, which gradually narrows with depth. Because the drifts of the current sub bench level at K-46 m were already done, it is also the location of mining blocks defined by them. It is proposed that the excavation blocks are arranged so that the greatest amount of ore is griped.

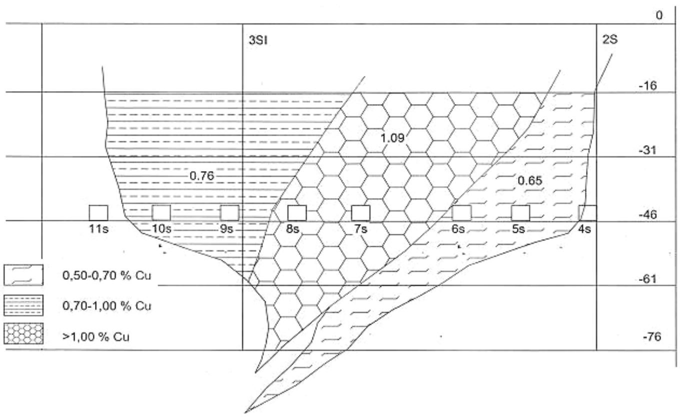

Figure 3. *Longitudinal geological section 1 in the northern part of the ore body Tilva Ros*

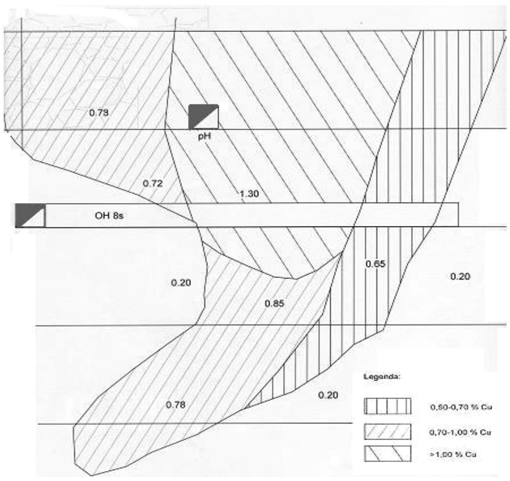

Figure 4. *Cross section through the drift OH 8s*

The lowest part of the ore body Tilva Ros appeared in the section corresponding to sublevel drift PH-8s, what is seen in the present profile in Figure 3. Parts of the ore, also width per 14 m, corresponding to the drifts PH-7s and PH-9s would make security pillars for the shown mining block.

The construction of preparation for the proposed mining method is adopted so that no preliminary development of preparation drifts is predicted at the bench next sublevel at K-61 m, but the drifts would be drilled corridors and loading drifts would be developed at K-76 m or at the level of XV horizon.

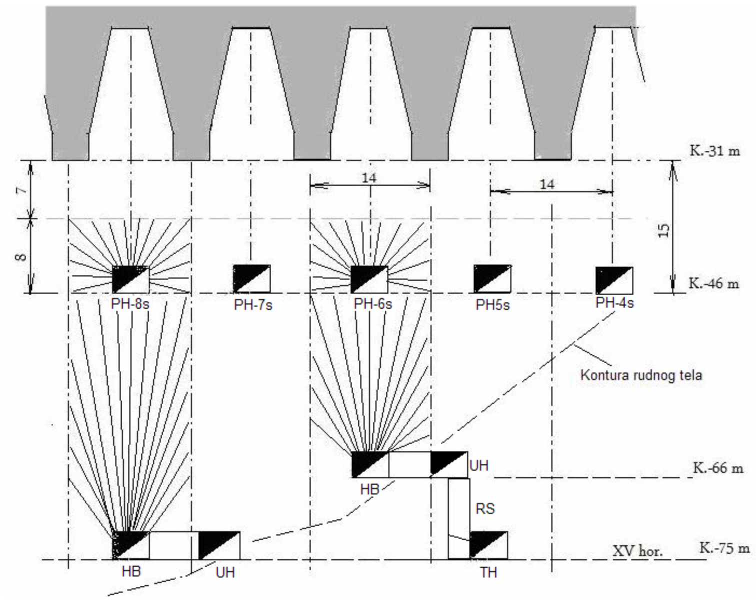

Figure 5. *Proposal of the I variant of mining method with leaving the open slopes for excavation the lowest parts of the ore body Tilva Ros*

An alternative proposal of the formation of mining blocks is shown in Figure 6. According to it, the slopes would be formed such as the existing sublevel drifts in the sides of a slope with a width of blocks identical to the previous blocks. This arrangement of mining blocks imposes a different layout of blast holes, which, however would be drilled with different layout. Selection the layout of blast holes in the mining blocks will depend on the available drilling rig and its possibilities. It means the existing machinery for drilling Simba-H 253, which were used for drilling deep blast holes in the sublevel caving method ("Swedish version"). If the procurement of new machines for drilling will be, their construction should enable drilling the blast holes with fan schedule (0 - 360ºC).

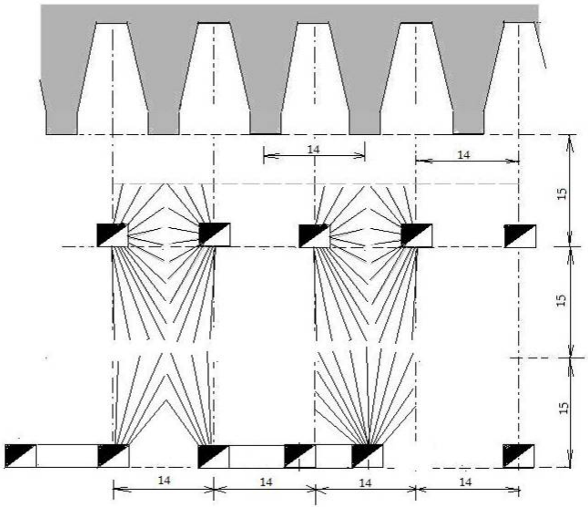

Figure 6. *Proposal of the II variant of mining method with an open stope and sublevel drifts for drilling in the sides of mining block*

Different technical solutions are also possible both for designs of mining methods and layout of drifts for drilling and schedule of blast holes in certain rows. There is otherwise too many opportunities for a relatively small amount of remaining ore reserves in the ore body. In fact, some mining blocks would also differ per possible their height what is conditioned by determined contours of ore body in each of the mining blocks.

In the mining block, the total height of 36 m (from K-76 to K-38 m), the block surface is

$$P_{bl} = 14 \text{ x } 36 - 4 \text{ x } 3{,}5 - 14 \text{ x } 3.5 - 0.5 \text{ x } 10 \text{ x } 5 = 504 - 14 - 49 - 25 = 416 \text{ m}^2$$

At 1 m length of block, the amount of ore is:

$$Q_{r1} = 416 \text{ m}^2 \text{ x } 2.8 \text{ t/m}^3 = 1164.8 \text{ t / m}$$

In drilling of blast hole rows at distance of 2.5 m, the amount of ore obtained by one blasting is:

$$Q_r = 1164.8 \text{ x } 2.5 = 2912 \text{ t}$$

This amount of ore can be loaded over the next 6 -1 0 shifts, (detailed dynamics of drilling and loading of ore will be given to the mining project).

2. MINING OF THE ORE BODY PARTS WITHOUT CARRIED OUT PREPARATION

Lower part of the ore body Tilva Ros ends in the form of large number of narrow parts which remind to the powerful mine veins (Figure 7). Such parts of the ore body are not suitable for use the sublevel caving method as non-mineralized parts of massif appear between the certain mineralized parts. This, in case the use of sublevel caving method result into appearance of enormously large depletion, or the selective mining has to be used, with a number of bench cuts, what would slow the dynamics of excavation and increase the losses and depletion of ore.

Therefore, as it is seen from Figure 6, the use of sublevel caving method would not be effective, in other words, the lower part of the ore body Tilva Rosh is not suitable for use the designed mining method.

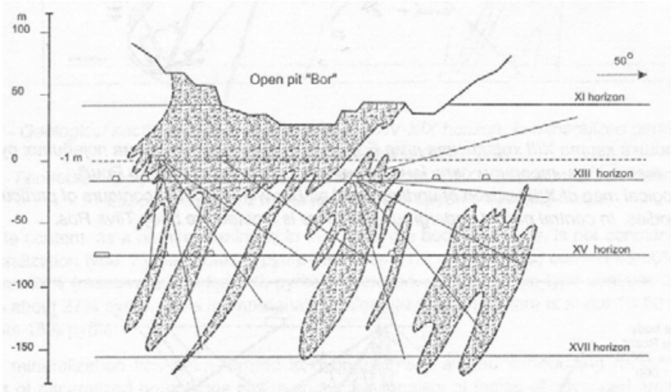

Figure 7. *Cross section of lower part of the ore body Tilva Ros*

Better solution could be found in the selective excavation of individual mineralized parts, even when applying some variants with the ore caving, which is not discussed here since it propose excavation without caving the upper massif. Figure 8 presents two parts of the ore body where the selective mining was proposed in the open slope with the sublevel ore caving. The basic problem in this mining is the increased volume of preparation, which corresponds to the situation of mining the ore bodies of relatively less thickness (10 - 20 m), whereas, the existing classification of ore deposits is a high thickness (<20 m).

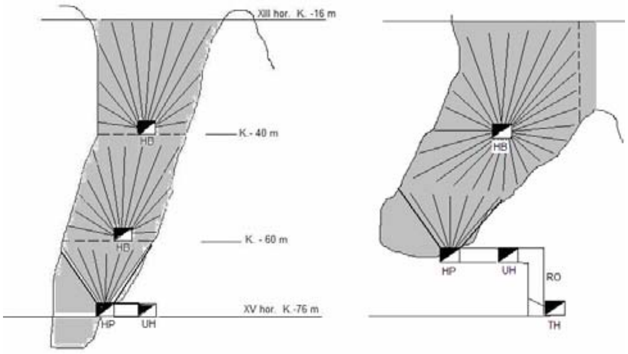

Figure 8. *Proposed method of sublevel mining in the open stopes distributed per direction of the certain parts of the ore body*

As the proposed method shows, the access transport drift (HT) is made for every single part and if the excavation starts from the level of horizon, then this drift is also the loading drift. In the ore body, a drift of cutting (HP) is made in the lowest part of it out of which a "trench" is formed by drilling the fan shaped blast holes and blasting where the short drifts, at distance 8-10 m, are made for ore loading from the cut. At higher levels, depending on the size of mined ore body, the sublevel drifts are made from the preparation slop by development of free space. It is made by previous development of cutting shaft and its extension forms a free space at the beginning of excavation chamber.

Upon completion of slope cutting, the drilling of blast holes is carried out and ore overthrowing by blasting. Blasted ore falls into the block bottom (cut off or trench) where it is loaded by the loading-transport diesel powered machine and transport to the stacker place or mine if the loading drift is above the level of horizon.

For the assumed excavation block, as shown in Figure 10, and assumed width of 15 m and length of 30 m, the amount of ore, obtained in different stages is:

- in making the cut off
  0.5 x (4 +15) x 12 x 2.8 = 319.2 t/m, that is in total 9576 tons,
- in the ore falling from the sublevels
  15 x 30 x 2.8 = 55,440 t.

The stated amount of ore is obtained with minimum losses and practically without depleting. At the shift capacity of ore loading and transport of 300 tons per shift, from the excavated part of ore body, the ore production would be provided in the 216 shifts or 72 days.

3. CONCLUSION

The ore body Tilva Ros ends up between the XIII and XVIII horizon in the form of a number of separate mineralized parts of different shapes and thickness. Non-mineralized parts of the rock massif are between them, and this circumstance makes irrational the use of designed sublevel caving method. On the other hand, there is a limit to the continued application of caving mining method, which led to the proposal for change the mining method that was proposed in this paper. The ore body does not have the remaining large amounts of ore reserves, however, an unfavorable situation with the current state of preparation the other ore bodies for mining seems that by this way, in the certain time, to ensure the ore production with a relatively good quality with respect to the current value of its value caused by the metal prices on the world market.

The newly proposed method of mining is not substantially different, and the proposed excavation of the sublevel open stope had already been applied in the subjected ore body and the possibility of use the equipment for ore drilling, mining and loading from that Jama underground mine has or had in the use. Development the mining project according to the proposed method requires precise definition the limits of the certain parts of ore body, which would be a task of the Geological Service of the mine, and there is a probability of the need for parallel research of ore body in this segment.

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TEHNIČKO – EKONOMSKI POKAZATELJI OTKOPAVANJA SIGURNOSNIH STUBOVA KOD OLOVNO -CINKALNIH LEŽIŠTA NA PRIMERU RUDNIKA ''TREPČA'' – STARI TRG**

Izvod

Osnovna karakteristika sadašnje podzemne eksploatacije je otkopavanje rude na sve većim dubinama i sa sve manjim sadržajem korisnih komponeti u njoj. Sa povećanjem dubine znatno se povećavaju i troškovi eksploatacije. Zato se pristupa iznalaženju najboljih tehničkih i tehnoloških rešenja s ciljem smanjenja troškova eksploatacije.

Sve ove okolnisti zahtevaju iznalaženje rešenja otkopavanja mineralnih sirovina koje su ostale nakon primarne faze eksploatacije, kao npr. sigurnosni stubovi, sigurnosni zidovi, komore itd.

U rudniku "Trepča" – Stari Trg, postoji više od 70 sigurnosnih stubova, uz permanentno ostavljanje novih tokom napredovanja otkopa. Sa sigurnošću se može konstatovati da u stubovima ostaje 15% rudne mase, koja će se eksploatisati u sekundarnoj fazi eksploatacije.

Ključne reči: *ruda, sigurnosni stubovi, eksploatacija, sekundarna faza.*

UVOD

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.

U dosadašnjoj eksploataciji u rudniku "Trepča" – Stari Trg uglavnom je primenjivana metoda krovnog otkopavanja u horizontalnim etažama odozdo na gore sa zasipavanjem, gde visinska razlika između horizonata iznosi 60 m.

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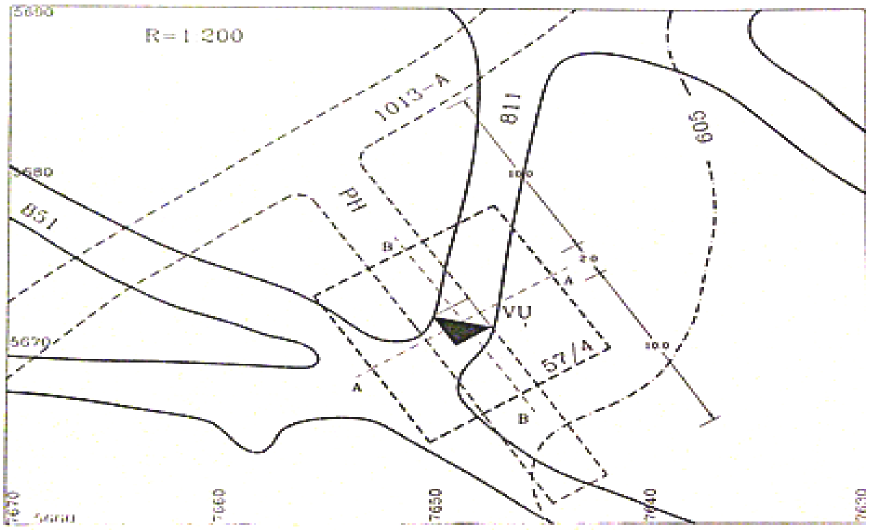
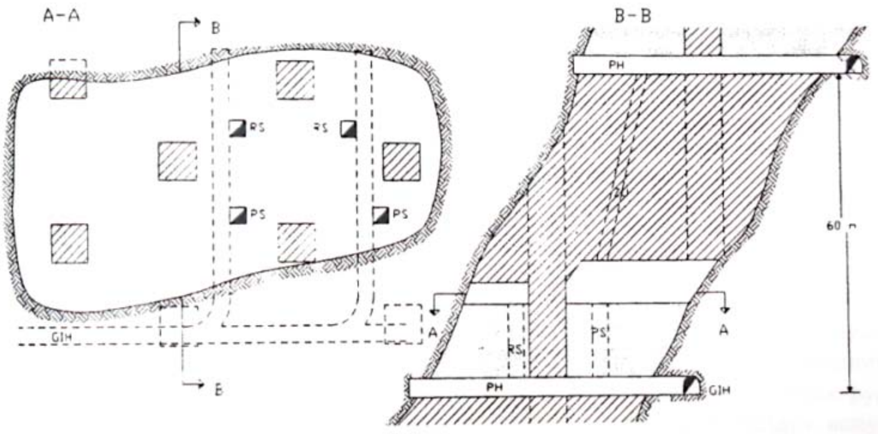
** *Ovaj rad je realizovan u okviru projekta,,Optimizacija procesa primene rude za Pk Prlovi u Flotaciji rudnika Rudnik(33045)koje finansira ministarstvo za prosvetu i nauku Republike Srbije u okviru programa istrazivanja za period od 2011 do 2014.*





Sl. 1. *Trepčanska metoda otkopavanja sa lokacijom sigurnosnog stuba*

Uzimajući u obzir dosadašnje iskustvo na otkopavanju sigurnosnih stubova i karakteristike savremene bušaće opreme i pokretne platforme ALIMAC razrađena je kombinovana uskopno – magazinska metoda otkopavanja sa masovnim obaranjem rude horizontalnim minskim bušotinama u lepezastom rasporedu. Veoma značajna saznanja dobijena su pri probnom otkopavanju sigurnosnih stubova magazinskom metodom uz primenu kratkih minskih bušotina. Pri tome je konstantovano, na nekoliko primera, da nakon otkopavanja i istakanja rude iz magazina okolni zasip usled zbijenosti čvrsto stoji i ne dolazi do nikakvih zarušavanja.

Zbog velikog broja sigurnosnih stubova i sličnih geološko – tehničkih uslova eksploatacije, u radu je izabran jedan karakterističan stub (57/4), koji će služiti kao osnova za eksploataciju ostalih sigurnosnih stubova. Za ovaj stub proračun rudnih rezervi dat je u tabeli br. 1.

Tabela 1. *Rudne rezerve u stubu (57/4)*

Broj Stuba	Površina p. preseka [m^2]	Visina stuba [m]	Zapreminska masa rude [t/m^3]	Rezerve u stubu [t]	Srenji sadržaj metala			Količina metala		
					Pb [%]	Zn [%]	Ag [g/t]	Pb [t]	Zn [t]	Ag [kg]
57/4	100	35	3,7	12.950	5,37	10,38	242	695	1.344	3.133

U cilju povećanja produktivnosti rada kao i sigurnosti pri radu, za otkopavanje sigurnosnog stuba predlaže se magazinska metoda otkopavanja sa izradom horizontalnih minskih bušotina u lepezastom rasporedu. Ova metoda omogućuje masovno miniranje većeg broja pojaseva miniranja, čime se može postići željeni kapacitet otkopavanja.

TEHNIČKO – EKONOMSKI POKAZATELJI OTKOPAVANJA

Tehnološki proces otkopavanja sigurnosnih stubova predstavlja skup određenih radnih operacija kojima se definiše raspored izvršavanja istih, što se prikazuje šemom organizacije rada na otkopavanju. Navedeni proces obuhvata izvođenje sledećih operacija: bušenje, miniranje, istakanje rude, utovar i prevoz rude, osiguranje otkopa, zasipavanje otkopa itd.

Niski gubici rude imaju veliki značaj jer se time produžava vek eksploatacije rudnika, a izgubljena sirovina je nepovratni gubitak prirodnog bogatstva zemlje. Obezbeđenje odgovarajućeg kapaciteta proizvodnje po obimu i kvalitetu ima velikog uticaja na cenu koštanja proizvoda, kao i na obezbeđenje opšteg plana proizvodnje.

Radna snaga u podzemnoj eksploataciji čini jedan od značajnih troškova proizvodnje, radi čega je potrebno pri izboru tehnološkog procesa rada nastojati da se proizvodni procesi što više mehanizuju i da mehanizacija bude što više iskorišćena. Drugim rečima, potrebno je sprovesti efikasnu organizaciju rada. Na sniženje troškova proizvodnje utiče i stepen osiromašenja rude, jer se, kod većeg osiromašenja na višak jalovine koja se dobija pri eksploataciji troši bez potrebe više radne snage, energije i materijala.

Prividna ekonomičnost može se postići kada se u nekom ležištu otkopavaju samo bogatije partije rude, a siromašnije ostavljaju. Takvim radom se u datom momentu mogu postići visoki ekonomski efekti, ali u kasnijoj fazi proizvodnje to može da ima vrlo negativne posledice i dalja eksploatacija se može dovesti u pitanje. Zbog toga bilo kod masovne ili selektivne eksploatacije, otkopavanje ležišta mora da se izvodi planski i sistematski.

POKAZATELJ ISKORIŠĆENJA

Iskorišćenje rude iz rudnog bloka, se po pravilu, izražava koeficijentom iskorišćenja rude (i_r). Ovaj koeficijent se izražava u apsolutnim vrednostima ili procentima. Praktično, ukoliko se ukupne rezerve nekog rudnog bloka označe sa T, gubitak rude koji je nastao usled otkopavanja (neotkopani deo rudnih rezervi) sa T_g, i deo rude koji se proizvede iz bloka sa $T_č$, onda između njih postoji sledeća relacija:

$$T = T_č + T_g\ [t]$$

Koeficijent iskorišćenja rude predstavlja odnos između količine proizvedene rude i ukupne količine rude u rudnom bloku, izražava se u %, odnosno:

$$i_r = \frac{T_č}{T} \times 100\%$$

Koeficijent gubitka rude predstavlja odnos između količine izgubljene rude i ukupne količine rude u rudnom bloku, odnosno:

$$g_r = \frac{T_g}{T} \times 100\%$$

Iz navedenih koeficijenata postoji sledeća zavisnost:

$$i_r + g_r = 1$$

Na osnovu tabele br. 1. geološke rudne rezerve u sigurnosnom stubu *57/4* iznose *12.950 t*. Izvesna količina rude će ostati zarobljena u okolini levkaste rudne sipke što je izazvano izradom iste. Navedena količina rude (T_g) iznosi *260 t*. Na osnovu toga mogu se proračunati sledeće vrednosti:

Količina rude koja se može proizvesti iz ukupnih rudnih rezervi sigurnosnog stuba:

$$T_č = T - T_g = 12.950 - 260 = 12.690\ [t]$$

Koeficijent gubitka rude:

$$g_r = \frac{T_g}{T} \times 100\% = \frac{260}{12.950} \times 100\% = 2\%$$

Koeficijent iskorišćenja rude:

$$i_r = \frac{T_č}{T} \times 100\% = \frac{12.690}{12.690} \times 100\% = 98\%$$

KOEFICIJENT OSIROMAŠENJA

Osiromašenje predstavlja odnos količine jalovine koja je dospela u rovnu rudu prema ukupnoj količini rovne rude, i izražava se koeficijentom osiromašenja rude.

$$O_r = \frac{T_2}{T_1} \times 100\%$$

gde je:

T_2 – količina jalovine koja je dospela u rovnu rudu,

T_1 – ukupna količina rovne rude ($T_1 = T_č + T_2$).

Osiromašenje se takođe može izraziti kao smanjenje sadržaja metala u proizvedenoj rovnoj rudi, u odnosu na sadržaj metala u rudnom bloku. U ovom slučaju radi se o otkopavanju sigurnosnih stubova, i na osnovu dosadašnjeg praćenja i ponašanja zasipnog materijala oko stuba utvrđeno je i da se nakon pražnjenja magazinskog prostora zasip i dalje čvrsto drži bez zarušavanja. Međutim, prilikom istakanja rude, ona povlači za sobom manju količinu zasipnog materijala gde se računa da neće prelaziti vrednost od *3%*, od vrednosti proizvedene rude ($T_č$), tako da će koeficijent osiromašenja iznositi:

$$O_r = \frac{T_2}{T_1} \times 100\% = \frac{381}{13.071} \times 100\% = 3\%$$

KOEFICIJENT PRIPREME

Koeficijent pripreme predstavlja odnos pripremnih radova za izvesnu metodu otkopavanja prema ukupnoj količini rude koja će se proizvesti iz nekog otkopa ili bloka koji se nalazi između dva horizonta.

$$k_p = \frac{1000 \times P_r}{T_1} \left[mm/t \right]$$

gde su:

P_r – obim pripremnih radova [m],

T_1 – količina proizvedene rovne rude [t].

Koeficijent pripreme služi kod projektovanja metode otkopavanja zbog obračuna učešća pripremnih radova u troškovima otkopavanja, a kod aktivnih rudnika i zbog planiranja godišnje količine pripremnih radova u cilju održavanja ili povećanja postojećeg kapaciteta proizvodnje. Obzirom na razlike između pojedinih sigurnosnih stubova u ležištu, koeficijent pripreme izračunat na primeru otkopavanja sigurnosnog stuba *57/4* neće biti relevantan u slučajevima otkopavanja ostalih sigurnosnih stubova. Prilikom otkopavanja ovog sigurnosnog stuba potrebno je izvršiti sledeći obim pripremnih radova:

- izrada *10 m* pristupnog hodnika (na nivou II horizonta),
- izrada *56 m* vertikalnog uskopa,
- izrada komore za smeštaj Alimak platforme koja dužinom i poprečnim presekom odgovara izradi *10 m* pristupnog hodnika,

Izrada levkaste rudne sipke (iskop $160\ m^3$ rude).

Koeficijent pripreme za hodnike i uskope se izražava u [mm/t] proizvedene rude, a za razna proširenja u ovom slušaju levkaste rudne sipke može se izraziti i u [m^3/t] proizvedene rude.

Koeficijent pripreme za hodnike iznosi:

$$k_p = \frac{1000 \times P_r}{T_1} = \frac{1000 \times 20}{13.071} = 1{,}5 \left[mm/t \right]$$

Koeficijent pripreme za uskope iznosi:

$$k_p = \frac{1000 \times P_r}{T_1} = \frac{1000 \times 56}{13.071} = 4{,}3 \left[mm/t \right]$$

Koeficijent pripreme za levkaste rudne sipke iznosi:

$$k_p = \frac{P_r}{T_1} = \frac{160}{13.071} = 0{,}12 \left[m^3/t \right]$$

PRODUKTIVNOST METODE OTKOPAVANJA I KAPACITET PROIZVODNJE

Produktivnost metode otkopavanja najčešće se izražava koeficijentom intenziteta proizvodnje, koji predstavlja odnos proizvedene rude u jednom bloku u toku godine prema jedinici površine otkopavanja. U konkretnom slučaju otkopavanja sigurnosnog stuba, kao kvalitetniji pokazatelj metode otkopavanja nameće se vreme potrebno za njegovo otkopavanje.

Ukupno vreme otkopavanja sigurnosnog stuba sastoji se od vremena za izvođenje pripremnih radova, vremena otkopavanja i vremena potrebnog za montažu i demontažu opreme i druge pomoćne radove.

Vreme izrade pripremnih radova zavisi od položaja sigurnosnog stuba i udaljenosti postojećih rudarskih prostorija. U konkretnom slučaju potrebno je uraditi *20 m* pristupnog hodnika i *56 m* vertikalnog uskopa.

Projektovanom tehnologijm izrade prostorija predviđeno je da se navedeni radovi izvedu tokom sledećeg vremenskog perioda:

- izrada pristupnog hodnika
  14 radnih smena,

- izrada vertikalnog uskopa
37 radnih smena,
- izvođenje pomoćnih operacija
6 radnih smena.

S toga vreme za izvođenje pripremnih radova iznosi:

$$t_p = 14 + 37 + 6 = 57 smena = 19\ radnih\ dana$$

Vreme otkopavanja sastoji se od vremena potrebnog za izvođenje radnih operacija na otkopavanju (bušenje minskih bušotina, miniranje utovar i transport viška oborene rude) i vremena potrebnog za utovar i transport magazionirane rude.

Prema šemi organizacije rada predviđeno je da se tokom radne smene izvrši obaranje jednog pojasa miniranja, što podrazumeva izvođenje sledećih radnih operacija: bušenje minskih bušotina, miniranje, utovar i transport viška oborene rude. Izborom parametara miniranja u sigurnosnom stubu definisano je *17* pojaseva miniranja te će vreme potrebno za otkopavanje istog iznositi:

- otkopavanje
17 x 2 = 34 radne smene,
- izrada levkastog proširenja
4 radne smene,
(što znači da je t_o = *38 radne smene*).

Količina magazionirane rude u sigurnosnom stubu može se proračunati po sledećoj formuli:

$$V_m = (P - p) \times H_s \times k_r - V_u \ \left[m^3\right]$$

$$V_m = (100 - 4) \times 35{,}5 \times 1{,}6 - 1.861 = 3.592 \left[m^3\right]$$

$$Q_m = \frac{3.592 \times 3{,}7}{1{,}6} = 8.306{,}5 [t]$$

gde je:

V_m – zapremina magazionirane rude po završetku otkopavanja stuba,

P – površina sigurnosnog stuba, *100 m^2*,

p – površina vertikalnog uskopa, *4 m^2*,

H_s – visina sigurnosnog stuba, *35,5 m*,

k_r – koeficijent rastresitosti rude, *1,6*,

γ – zapreminska masa rude, *3,7* [t/m^3],

V_u – količina oborene rude koja je utovarena tokom otkopavanja stuba,

v_u – višak zapremine oborene rude, pri jednom pojasu miniranja,

n_s – broj radnih smena u kojima se vrši utovar rude.

$$V_u = v_u \times n_s = 109{,}44 \times 17 = 1.861 \left[m^3\right]$$

Vreme pražnjenja (istakanja) magazionirane rude:

$$t_i = \frac{Q_m}{Q_u} = \frac{8.306{,}5}{400} = 20{,}77 \approx 21 smena$$

Ukupno vreme otkopavanja, utovara i transporta magazionirane rude iznosi:

$$t_u = t_0 + t_i = 38 + 21 = 59\ smena \approx$$
$$\approx 20\ radnih\ dana$$

Vreme potrebno za montažu i demontažu opreme predviđa se *6 radnih smena*, a za pomoćne poslove kao što su produženje cevovoda za vazduh i vodu, čišćenje pristupnih puteva predviđa se *9 radnih smena*. S toga ovo vreme iznosi:

$$t_r = 6 + 9 = 15 smena = 5 radnih\ dana$$

Saglasno usvojenoj šemi organizacije rada, ukupno vreme otkopavanja sigurnosnog stuba iznosi:

$$T_0 = t_p + t_0 + t_r = 57 + 59 + 15 =$$
$$= 131\ smena = 44\ radna\ dana$$

OTKOPNI UČINCI

Ocena produktivnosti rada u podzemnoj eksploataciji može se izraziti učinkom, koji predstavlja količinu proizvoda po izvršenoj osmočasovnoj nadnici. Pri tome se razlikuju sledeći učinci: otkopni učinak, učinak na raznim operacijama ili procesima, jamski učinak, rudnički učinak.

Otkopni učinak predstavlja produktivnost rudnika, a u njega su uključene sve utrošene nadnice na pripremi, bušenju u otkopima, miniranju, utovaru rude, podgrađivanju, zasipavanju i na svim pomoćnim radovima pri otkopavanju.

$$U = \frac{T}{n}\left[t/_{nad}\right]$$

gde su:

- U – učinak pri izvođenju određene radne operacije,
- T – količina rude koja se otkopava angažovanjem n nadnica,
- n – broj nadnica koje su potrbne za otkopavanje T količine rude.

Šemom organizacije rada na otkopavanju sigurnosnog stuba predviđeno je da se za obaranje jednog pojasa miniranja (*675 t rude*) izvrši tokom dve radne smene, pa se shodno tome određuju učinci za operacije bušenja, miniranja i utovara sa transportom.

Za bušenje minskih bušotina jednog pojasa miniranja predviđa se angažovanje *3* radnika u dve smene, odnosno za izvođenje ove operacije potrebno je *6 nadnica*, pa je učinak:

$$U_b = \frac{675}{6\,nad} = 112{,}5\left[\frac{t}{nad}\right]$$

Za izvođenje radne operacije miniranja jednog pojasa biće angažovana tri radnika u smeni koji će vršiti punjenje minskih bušotina, ugradnju pojačivača eksplozije (bustera), začepljenje minskih bušotina, njihovo povezivanje i paljenje. Učinak na miniranju ima sledeću vrednost:

$$U_m = \frac{675}{3\,nad} = 225\left[\frac{t}{nad}\right]$$

Na utovaru i transportu rude biće angažovan po jedan radnik. Navedene radne operacije se izvršavaju tokom jedne radne smene tako što se količina od *253 t* rude utovari i transportuje do centralne rudne sipke. S toga će učinci na utovatu i transportu rude imati iste vrednosti:

$$U_u = 253\left[\frac{t}{nad}\right],\quad U_t = 253\left[\frac{t}{nad}\right]$$

ZAKLJUČAK

Tehničko – ekonomski parametri otkopavanja sigurnosnih stubova u suštini predstavljaju analizu svih faktora i pokazatelja koji su značajni za ostvarenje budućih ekonomskih efekata pri njihovoj eksploataciji.

Može se, naglasiti, da poslednjih godina, shodno opšte prihvaćenim principima savremenog društva, ekonomski parametri i rentabilnost proizvodnje predstavljaju dominantne faktore koji utiču na konačnu odluku o tome da li treba vršiti, ili ne treba vešiti, eksploataciju određenog dela ležišta.

Na osnovu rezultata dobijenih realizacijom projektovanih parametara odrediće se opšti uslovi buduće eksploatacije, kao i metoda otkopavanja. Od vrednosnih pokazatelja približno će se oceniti kapacitet otkopavanja, troškovi eksploatacije po 1 t otkopane mineralne sirovine, kao i ukupna i specifična ulaganja potrebna za otkopavanje.

U radu su prikazavi osnovni parametri i pokazatelji primenjene metode otkopavanja sigurnosnih stubova, kao i njihova analiza i upoređenje.

Za konkretne uslove rada u rudniku ”Trepča”, predloženo je rešenje otkopavanja sigurnosnih stubova magazinskom metodom otkopavanja sa izradom horizontalnih minskih bušotina u lepezastom rasporedu. Predloženo rešenje garantuje manji obim pripremnih radova, veliku sigornost, zadovoljavajući proizvodni kapacitet i znatno smanjenje troškova eksploatacije.

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AN APPROACH REGARDING SELECTION OF MINING TRUCKS

Abstract

The selection of the best possible truck type (model) is of crucial importance for the mine decision-makers. In this paper, a novel approach for the selection of mining truck is proposed. Both the Coefficient of Technical Level (CTL) and Analytic Hierarchy Process (AHP) methods were used in this study. The CTL represents a mathematical function which takes into consideration the relationship between the main technical parameters of mining trucks. A hierarchical structure has been developed where both attribute categories and their importance, i.e. priority or weight in the selection process were determined. For each category, a number of sub-attributes and their priorities were assigned, and a pairwise comparison among them was performed using the AHP method. In the final part of the study, the CTL was integrated into the AHP and represents a holistic approach in the selection of mining trucks. A case study on 180 t truck category offered by five manufacturers was carried out. A sensitivity study of obtained results was also performed indicating a very stable solution. The methodology presented in this

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paper may be used by mining operators to help in the selection of a specific type when acquiring a new truck. It may also be used by equipment manufacturers to help in analysis of benefits of potential modifications concerning an existing truck type and/or development of a new type.
Key words: *Mining Trucks, Multi-attribute Selection Method, Decision-Making, Selection*

1. INTRODUCTION

Trucks are the basic equipment for material haulage in coal, metal and non-metal surface mines. Since the mid-1930s when such trucks were introduced in the mining industry, their size has increased from approximately 13.5 t to today's 360 t of maximum payload (Komljenovic et al., 2003). Large and efficient trucks make mining possible and economically viable. The equipment selection in general, and the truck selection in particular are one of the most important activities that affect surface mine design, production planning, and cost figures. Because of their importance, it is crucial to develop and apply an adequate methodology for their selection.

Various Multi-Attribute Decision Methods (MADM) such as the Analytic Hierarchy Process (AHP) may be used to facilitate decisions that involve multiple competing criteria. MADM methods use multiple criteria rather than relying on a single criterion to make a decision. Thus, MADM techniques are ideally suited to address decision situations that use multiple criteria for selecting the best alternative. The AHP is perhaps the most widely-used of all the MADM methods. The main objective of this study is to develop a comprehensive, holistic selection methodology for mining trucks, which is based on a multi-attribute approach.

The first section of the study provides a bibliography review of previous research related to the selection of mining equipment. The second part focuses on a Coefficient of Technical Level (CTL) developed by Komljenovic et al. (2003) while the third section describes the procedure of mining truck selection using Analytic Hierarchy Process (AHP). Finally, the CTL is integrated into the AHP model, and a case study is carried out to demonstrate the benefits of using this novel approach in truck selection process.

2. LITERATURE REVIEW

Selection of capital equipment is a very important decision-making process in mining industry where many variables, constraints and criteria should be included.

The most comprehensive and detailed analysis on equipment selection for high production low cost (HPLC) mining operations is given by the P&H MinePro (2003). In this study, a specific selection variables and attributes are assigned to hydraulic and cable (electric) shovels in order to help in the selection process. The key selection criteria are classified into eight categories: technical; machine operation; geology and deposit characterization; digging and loading; productivity; maintenance; environmental impact; and commercial considerations. In the term of importance in the selection process, each criterion is ranked as low, high or very high. Specific attributes of both hydraulic and electric shovels, as they relate to the selection criteria, describe the various features and characteristics that make the machine more or less appropriate for a given criterion. This study represents an invaluable practical contribution to the shovel selection in HPLC operations. Moreover, many of the selection attributes, slightly adapted, may be applied in selection of other types of mining equipment.

World-leading mining equipment manufacturer Caterpillar (CAT, 1993) introduces six primary criteria that need to be considered in the truck selection process. These are the type and density of material to be hauled; travel distance; grade; the road conditions; type of dump site; and production requirements. Generally, trucks are divided into two categories where construction/mining trucks are used for materials from soft to very hard, while the quarry trucks can be used for mild to medium hard materials. Travel distance is one of the predominant variables in determining the overall production efficiency. According to the Caterpillar, construction/mining trucks are most efficient for the operations that require longer hauling distance, while quarry trucks are specifically designed for distances from 300 to 2,300 meters. Grade also plays an important role in truck selection. Construction/mining trucks are capable of handling steeper grades (up to 15 %) and will have a better load retention while quarry trucks will be better utilized for roads with less severe grades (8-10 %). Haul road conditions play a significant role in influencing operating costs. Construction/mining trucks are preferable if the significant amount of time will be spent in rough hauling conditions. These trucks have a high rimpull and durable power train in such conditions. Quarry trucks are better suited for the moderate hauls on well-maintained roads. Rough road conditions places an additional stresses on the power train and the frame of these trucks. The construction/mining trucks are well-suited when material needs to be dumped all at once such as fast dumping of overburden on waste dump sites. However, when material needs to flow evenly as it dumps, into crushers and chutes, quarry trucks are desirable. These trucks retain the load in tray longer, feeding crushers without choking them.

Martin et al. (1982) points specifically to the selection considerations for a truck as follows: material characteristics of the mine, loading equipment, haul route requirements, maneuvering space, dumping conditions, capacity, engine power and altitude limitations, final drive gear ratios for mechanical drives, two axle or three axle configuration, mechanical or electrical drive system, tires size, tread and ply rating. According to Lineberry (1986), determination of the truck power is the key in the selection of mining truck. The author indicates that the proper power selection is required in order to ensure safe and efficient operation and to minimize overall handling expenses. In support to truck selection, evaluation and design, Lineberry (1985) developed an optimization model that considers horsepower, capacity, velocity, and total cost.

Over the last couple of decades, a lot of researchers have used different techniques and methods in order to help in the process of equipment selection. According to Burt et al. (2005) these techniques and methods include classical methods (match factor, bunching theory and productivity curves), operations research techniques (integer programming and nonlinear programming) and artificial intelligence techniques (expert systems, knowledge based methods and genetic algorithms). In their study, Burt et al. (2005) generally point-out some of the weaknesses of these approaches such as that integer programming solutions tend to oversimplify the model or rely on excessive assumptions, or that methods of artificial intelligence have been applied to equipment selection with some success. They indicate that common disadvantage among all of these models are fleet homogeneity, i.e. assumption that the truck fleet consists only of one type of truck. They used a new Mixed Integer Linear Programming model that allows for mixed-type fleets and selects the truck and loader types within the solution.

Other researchers also state that operations research optimization techniques currently in use display serious limitations (Haidar et al. 1999). They indicate that these techniques lack flexibility and often are invalidated by their inability to cope with a large number of variables, constraints, and uncertainty, which are a natural part of the mining process. Bascetin (2004) indicates that the most common methods applied in equipment selection are the expert systems and decision support systems. Amirkhanian and Baker (1992) developed a rule-based expert system for selecting earth-moving equipment. This system includes 930 rules interpreting information such as soil conditions, operator performance, and required earth-moving operations. Xie (1997) developed the prototype knowledge-based expert system called EESET (Earthmoving Equipment Selection and Estimation Tool). The system has the capability of selecting appropriate fleets of machines and to estimates their outputs and costs based on the given working conditions. Alkass and Harris (1988) developed an expert system called ESEMPS for earth-moving equipment in road construction. Ganguli and Bandopadhyay (2002) developed an expert system for equipment selection for various unit operations in surface mining. Haidar et al. (1999) developed a decision support system XpertRule for the selection of opencast mine equipment (XSOME). The system was designed using a hybrid knowledge-base system and genetic algorithms. The similar approach with a hybrid knowledge base system and genetic algorithms is used by the same authors in selecting excavation and haulage equipment in surface mining (Naoum and Haidar, 2000).

Bascetin (2003 and 2004) used the method of Analytical Hierarchy Process (AHP) for the selection of loading and hauling equipment for Orhaneli open pit mine in Turkey. Criteria for the equipment selection included capital and operating costs, operating condition, the ground and road conditions and equipment technical parameters. A multi-attribute decision-making process for equipment selection in surface mining was also used by Bimal et al. (2002). Kesimal and Bascetin (2002) used AHP model and fuzzy set theory for equipment selection in both surface and underground mining. Bandopadhyay (1987) used partial ranking of primary stripping equipment in surface mine planning and fuzzy algorithm. The author developed the process of ranking alternatives after determining their rating, and considers that the supports for each alternative are fuzzy sets themselves.

Komljenovic et al. (2003) established the relations among the technical parameters of rear dump mining trucks. The model is based on statistical and correlation analysis carried out on the technical specifications and data provided by the major truck manufacturers worldwide. The established relations in connection with economical parameters (ownership and operating costs) were used to formulate a selection coefficient of mining trucks. The selection methodology is further elaborated at some extent by using multi-attribute decision-making process (Komljenovic and Kecojevic, 2006).

Previous research studies reviewed here indicate that different techniques and methods were used to help in the process of equipment selection. However, this study developed a novel approach by elaborating a hierarchical structure where attribute categories and their importance, i.e. priority/weight in the selection process were determined. For each category a number of sub-attributes and their priorities were assigned, and a pairwise comparison among them was performed. The text that follows explains the selection methodology in details.

3. SELECTION METHODOLOGY

3.1 Coefficient of technical level (CTL) of mining trucks

The Coefficient of Technical Level (CTL) represents a mathematical function which takes into consideration the interrelations among main technical parameters of mining trucks such as gross vehicle weight (*GVW*), vehicle weight (*GV*), maximum payload (*Q*), struck capacity SAE (*V*), heaped capacity SAE (2:1) (*VM*), motor power (*N*), and top vehicle speed (*VT*). This method is based on a statistical and correlation analysis of these parameters and was originally developed by Komljenovic et al. (2003). It is slightly modified for the purpose of this study by omitting its economic part, which is incorporated in the AHP method. In the previous study by Komljenovic et al. (2003), a total of five comparative coefficients were developed

$$k_1 = \frac{Q}{GV} \quad (1)$$

$$k_2 = \frac{Q}{N} \quad (2)$$

$$k_3 = \frac{GVW}{N} \quad (3)$$

$$k_4 = \frac{VM}{GV} \quad (4)$$

$$k_5 = \frac{VT \times Q}{N} \quad (5)$$

The study included technical specifications of 52 different types of mining trucks offered by eight manufacturers worldwide (Caterpillar, O&K, Euclid, Terex, Unit Rig, Komatsu, Liebherr and Belaz) (see Appendix 1). A wide variety of truck sizes was considered including gross vehicle weight ranging from 52 to 555 t, maximum payload from 30 to 328 t, and motor power from 260 to 2 500 kW. Table 1 shows basic statistical values of comparative coefficients obtained through the study. The results of the study reveals the correlation coefficient within the range of r = 0.4133 to r = 0.8382 which indicates dependence of these coefficients on the size of the trucks to be from weak to significant. Only the comparative coefficient k_2 shows a strong correlation with respect to the truck payload Q as k_{2Q} = f(Q) with r = 0.8382. The relationship can be expressed by the following equation (see Figure 1):

$$k_{2Q} = 0.0169 \times \ln(Q) + 0.0433 \quad (6)$$

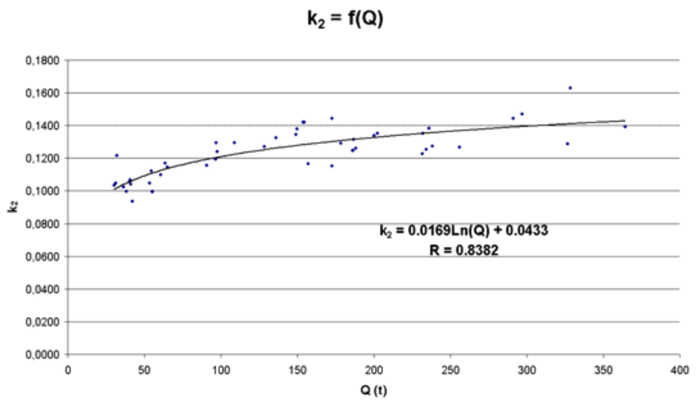

Fig. 1. *Relationship between the coefficient k2 and the maximum truck payload Q*

Table 1. *Statistical characteristics of the comparative coefficients*

	k_1	k_2	k_3	k_4	k_5
Mean	1.3958	0.1222	0.2102	0.8580	7.0582
Standard Deviation	0.1611	0.0152	0.0222	0.0931	1.0756
Minimum	1.0552	0.0938	0.1596	0.6948	5.3297
Maximum	1.9272	0.1630	0.2706	1.1589	10.5930
Confidence Level (95.0%)	0.0449	0.0042	0.0062	0.0259	0.2994
Correlation Coefficient	0.6688	0.8382	0.7028	0.4133	0.6429

In the next step, the least squares method was used to analyze the relationship between the main truck technical parameters. For instance, installed motor power N was considered as a function of gross vehicle weight GVW as N_{GVW} = f (GVW), maximum payload Q as N_Q = f (Q), and heaped capacity VM as N_{VM} = f (VM). The analysis revealed high values of the correlation coefficients ($r > 0.97$). The obtained relationships are as follows:

$$N_{GVW} = 4.24 \times GVW + 83.344; \quad r = 0.9895 \tag{7}$$

$$N_Q = -0.0029 \times Q^2 + 7.8206 \times Q + 71.527; \quad r = 0.9907 \tag{8}$$

$$N_{VM} = -0.0253 \times VM^2 + 16.486 \times \times VM + 16.486; \quad r = 0.9781 \tag{9}$$

A high correlation coefficient (r = 0.9868) was also obtained for relationship between the vehicle weight GV and its maximum payload Q as G_{VQ} = f(Q) (see also Figure 2):

$$GV_Q = -0.0005 \times Q^2 + 0.7846 \times \times Q + 0.916; \quad r = 0.9868 \tag{10}$$

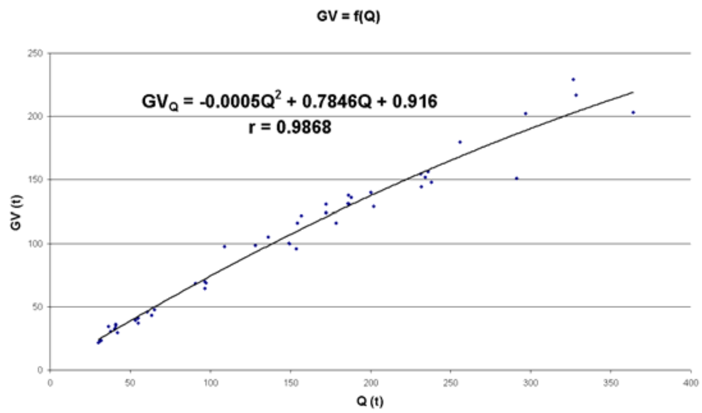

Fig. 2: *Relationship between the truck weight GV and its maximum payload Q*

The highest correlation coefficient (*r = 0.9976*) has been obtained for relationship between the gross vehicle weight GVW and its maximum payload Q as $G_{VWQ} = f(Q)$ (see Figure 3):

$$GVQW_Q = 1.6158 \times Q + 10.326 \quad r = 0.9976 \tag{11}$$

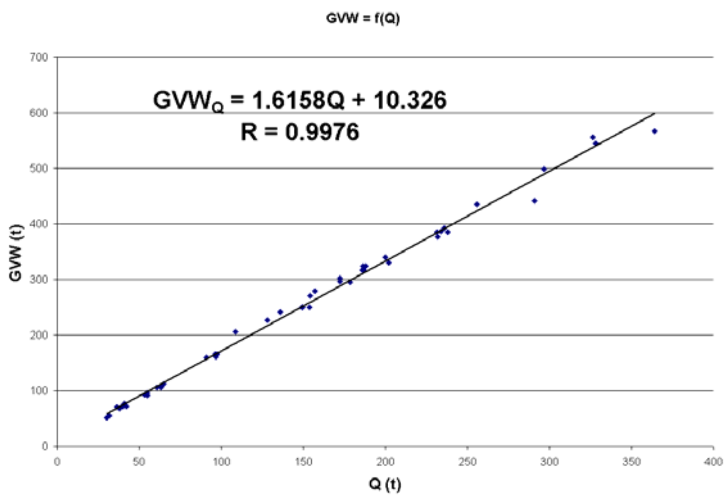

Fig. 3: *Relationship between the gross truck weight GVW and its maximum payload Q*

However, the relationship $G_{VWQ}=f(Q)$ (Equation 11)is not used in further consideration. Instead, the relationship is judged $G_{VQ} = f(Q)$ (Equation 10) more appropriate in this regard given that its contains purely design data.

It is important to note that these relations (Eqs. 6 through 11) are quite universal regardless of the manufacturer. It was observed that there are small variations as a result of different approaches in conception and design philosophy, which characterize various manufacturers.

Generally, there are two groups of technical parameters and comparative coefficients: a) parameters and coefficients whose variability significantly depends on the size of the truck, or which have a certain dependency among them (Group I); and b) comparative coefficients, which do not vary significantly with the size of the machine, or do not reveal any dependency among them (Group II). While formulating the criterion, each of these two groups of analyzed parameters should be considered in a different manner. However, in order to be able to compare numbers expressed in different units, the criterion should be indicated with dimensionless numerical values. This was the rationale for introducing the ratios of these parameters for the further analysis.

The quantification of the Group I of parameters and coefficients $\left(N_{GVW}, N_Q, N_{VM}, G_{VQ}\ and\ k_{2Q}\right)$ was performed as follows:

(i) The deviation between the value of the parameter (coefficient) and its calculated value was determined;
(ii) This deviation was introduced into a ratio with the calculated value of the same parameter (coefficient);
(iii) Positive and negative deviations (differences) have the same significance and importance and should be treated in the same manner;
(iv) Large deviations $\left[\leq(-0.3)\ and\ \geq 0.3\right]$ should be treated as unfavorable, and smaller differences have to be privileged;
(v) The best way to meet the conditions (iii) and (iv) is to bring this ratio to the square; and
(vi) The obtained mathematical expression (ratio) has to tend towards a minimum.

The mathematical interpretation of the described approach is shown below:

$$Z_{1(i)} = \frac{N_{(i)} - N_{GVW(i)}}{N_{GVW(i)}} \quad (12)$$

$$Z_{2(i)} = \frac{N_{(i)} - N_{Q(i)}}{N_{Q(i)}} \quad (13)$$

$$Z_{3(i)} = \frac{N_{(i)} - N_{VM(i)}}{N_{VM(i)}} \quad (14)$$

$$Z_{4(i)} = \frac{GV_{(i)} - GV_{Q(i)}}{GV_{Q(i)}} \quad (15)$$

$$Z_{5(i)} = \frac{k_{2(i)} - k_{2Q(i)}}{k_{2Q(i)}} \quad (16)$$

$$K_{oa(i)} = \sum_{j=1}^{5} Z_{j(i)}^{2} \rightarrow \min \quad (17)$$

where:

$Z_{(i),\ldots,5(i)}$ – intermediate value used to calculate the coefficient of technical level (CTL),

$j = 1, 2, \ldots, 5$ – parameters dependant upon the truck size (N_{GVW}, N_Q, N_{VM}, G_{VQ}, k_{2Q})

N – number of truck types compared ($i = 1, 2, \ldots, N$)

K_{oa} – part of the CTL for the Group I

The calculated values of weight for N_{GVW}, N_Q, N_{VM}, G_{VQ}, and the coefficient k_{2Q} are determined according to the Eqs. (7), (8), (9), (10), and (6), respectively.

By introducing the square of the deviation in Eq. (17), large differences are disadvantaged, and small ones are privileged. This is particularly true for the deviations in the interval of $(-0{,}3 \geq Z_{j(i)} \geq 0{,}3)$. Moreover, with the square of their ratio, the positive and negative deviations (differences) are treated in the same manner.

The comparative coefficients of the Group II such as k_1, k_3, k_4 and k_5 do not reveal a significant dependence on the size of the truck. The coefficients of the Group II also need to be expressed as a ratio. The mean values of each comparative coefficient were used as a reference (benchmark), and deviation from the mean value can be used to quantify these coefficients.

However, the positive and negative deviations from the means do not have the same significance for this analysis, and is advantageous to have small negative deviations. Nevertheless, the large negative deviations have to be disadvantaged. In order to favor small deviations and to make large ones unfavorable, a mathematical function has to be created. This function must always be positive. For the negative deviations, the best results may be obtained by using the following mathematical formulation:

$$y = x^4 \qquad (-1 \leq x < 0) \qquad (18)$$

Positive deviations of the coefficient's value from the means should be disadvantaged as well. For this analysis, they were considered in the same manner as the coefficients of Group I. The mathematical form of this part of the criterion is shown below:

$$K_{ob(i)} = \sum_{j=6}^{9} K_{j(i)} \rightarrow \min \qquad (19)$$

$$K_{6(i)} = \begin{cases} Z_{6(i)}^4; & -1{,}0 < Z_{6(i)} < 0 \\ Z_{6(i)}^2; & Z_{6(i)} \geq 0 \end{cases} \qquad Z_{6(i)} = \frac{k_{1(i)} - k_{1(m)}}{k_{1(m)}} \qquad (20)$$

$$K_{7(i)} = \begin{cases} Z_{7(i)}^4; & -1{,}0 < Z_{7(i)} < 0 \\ Z_{7(i)}^2; & Z_{7(i)} \geq 0 \end{cases} \qquad Z_{7(i)} = \frac{k_{3(i)} - k_{3(m)}}{k_{3(m)}} \qquad (21)$$

$$K_{8(i)} = \begin{cases} Z_{8(i)}^4; & -1{,}0 < Z_{8(i)} < 0 \\ Z_{8(i)}^2; & Z_{8(i)} \geq 0 \end{cases} \qquad Z_{8(i)} = \frac{k_{4(i)} - k_{4(m)}}{k_{4(m)}} \qquad (22)$$

$$K_{9(i)} = \begin{cases} Z_{9(i)}^4; & -1{,}0 < Z_{9(i)} < 0 \\ Z_{9(i)}^2; & Z_{9(i)} \geq 0 \end{cases} \qquad Z_{9(i)} = \frac{k_{5(i)} - k_{5(m)}}{k_{5(m)}} \qquad (23)$$

where:

$k_{1(m)}$ – mean value of the comparative coefficient k_1

$k_{3(m)}$ – mean value of the comparative coefficient k_3

$k_{4(m)}$ – mean value of the comparative coefficient k_4

$k_{5(m)}$ – mean value of the comparative coefficient k_5

$Z_{6,\ldots,9(i)}$ – intermediate value used to calculate the coefficient of technical level

$K_{6,\ldots,9(i)}$ – intermediate variable used to calculate the intermediate values $Z_{6,\ldots,9(i)}$

j = 6, 7, 8, 9 –parameters which are not dependant upon the truck size (k_1, k_3, k_4, k_5)

N – number of truck types compared (i = 1, 2, …, N)

K_{ob} - part of the CTL for the Group II

With these mathematical functions, all elements were combined together to create the technical part of the criterion. Its form is as follows:

$$K_{o(i)} = K_{oa(i)} + K_{ob(i)} \rightarrow \min \quad (24)$$

or

$$K_{o(i)} = \sum_{j=1}^{5} Z_{j(i)}^{2} + \sum_{j=6}^{9} K_{j(i)} \rightarrow \min \quad (25)$$

Equation (25) brings into a relationship main technical parameters of mining trucks enabling to determine their Coefficient of Technical Level (CTL).

3.2. Application of Analytic Hierarchy Process (AHP) in truck selection

The AHP method has a number of desirable attributes which are relevant in a mining truck selection process. These attributes are as follows: (i) it is a structured decision-making method which can be documented and replicated, (ii) it is applicable to decision situations involving multi-criteria, (iii) the AHP is applicable to decision situations involving subjective judgment, (iv) it uses both qualitative and quantitative data, (v) it provides measures of consistency of preference, (vi) there is ample documentation of AHP applications in the academic literature, (vii) commercial AHP software is available with technical and educational support, (viii) the AHP is suitable for group decision-making, and (ix) the AHP facilitates a comprehensive and logical analysis of problems for which considerable uncertainty exists.

The AHP is especially suited for application to problem evaluations in which qualitative factors dominate. It can be characterized as a multi-attribute decision technique that can combine qualitative and quantitative factors in the overall evaluation of alternatives. This method helps to accommodate both the effects of uncertainty on decisions, and a need to clarify decision objectives and carefully formulate decision alternatives (US NRC, 2003). AHP also facilitates a comprehensive and logical analysis of problems for which considerable uncertainty exists. If fact, the power of AHP (and to a large degree its uniqueness) is being able to consider qualitative goals and attributes within its framework.

The AHP method helps determine the priority any alternative has on the overall goal of the problem of interest. The analyst/user creates a model of the problem by developing a hierarchical decomposition representation. At the top of the hierarchy is the overall goal or prime objective one is seeking to fulfill. The succeeding lower levels then represent the progressive decomposition of the problem. The analyst, or other knowledgeable party, completes a pairwise comparison of all elements in each level relative to each of the program elements in the next higher level of the hierarchy. The composition of these judgments fixes the relative priority of elements in the lowest level (usually solution alternatives) relative to achieving the top-most objective. Saaty (1980, 1990) recommends four steps to be used the AHP application: (i) build a decision "hierarchy" by breaking the general problem into individual criteria - User/Analyst Modeling Phase, (ii) gather relational data for the decision criteria and alternatives and encode using the AHP relational scale - User/Analyst pairwise comparison input),

(iii) estimate the relative priorities/weights of the decision criteria and alternatives, and (iv) perform a composition of priorities for the criteria, which gives the rank of the alternatives (usually lowest level of hierarchy) relative to the top-most objective - AHP software or a spreadsheet.

Basic elements of the AHP method are presented below as per Saaty (1980, 1990). For computing priorities of the elements, it is required to develop a judgmental matrix. For the present study, the calculated scoring values ($SV_{A \rightarrow B}$) are used to define the latter as follows:

$$A = \begin{bmatrix} a_{11} & a_{12} & \cdots & a_{1n} \\ a_{21} & a_{22} & \cdots & a_{2n} \\ \cdots & \cdots & \cdots & \cdots \\ a_{n1} & a_{n2} & & a_{nn} \end{bmatrix} \qquad (26)$$

Where a_{ij} represents the pairwise comparison rating between the element i and element j of a level with respect to the upper level. In the current study, the elements a_{ij} correspond to the calculated scoring values ($SV_{A \rightarrow B}$). The entries a_{ij} are governed by the following rules:

$$a_{ij} > 0 \ \{\mathrm{a_{ij}\ takes\ values\ 1,...,9}\};$$
$$a_{ij} = \frac{1}{a_{ji}}; \ a_{ii} = 1 \forall i \qquad (27)$$

The priorities of the elements can be estimated by finding the principal ***eigenvector*** $\vec{W}$ of the matrix A:

$$A \cdot \vec{W} = \lambda_{\max} \cdot \vec{W} \qquad (28)$$

When the weight vector $\vec{W} = [w_1,...,w_n]^T$ is normalized, it becomes the vector of priorities of elements of one level with respect to the upper level. The parameter λ***max*** is the largest ***eigenvalue*** of the matrix A. In other words, the weight vector $\vec{W}$ is the eigenvector of A corresponding to its maximum eigenvalue λ***max***.

In cases where the pairwise comparison matrix satisfies transitivity for all pairwise comparisons it is said to be consistent and it verifies the following relation:

$$a_{ij} = a_{ik} \times a_{kj} \quad \forall i,j,k \qquad (29)$$

Table 2 gives the generic comparison scale used in the AHP method.

In the text that follows, four AHP generic steps listed above are adapted and applied in the process of truck selection. The overall objective is a selection of the best truck type.

Step 1. A hierarchical structure has been developed where attribute categories, and their importance (priority/weight in the selection process) are determined. The major truck selection categories include:

- Technical;
- Machine Operation,
- Impact of Material Properties,
- Haulage Productivity,
- Maintenance,
- Environmental Impact,
- Commercial/Cost Considerations and
- Other Factors.

For each category a number of sub-attributes and their priorities are assigned (Figure 4). It should be noted that the proposed priorities may be changed as a function of specific needs and local conditions.

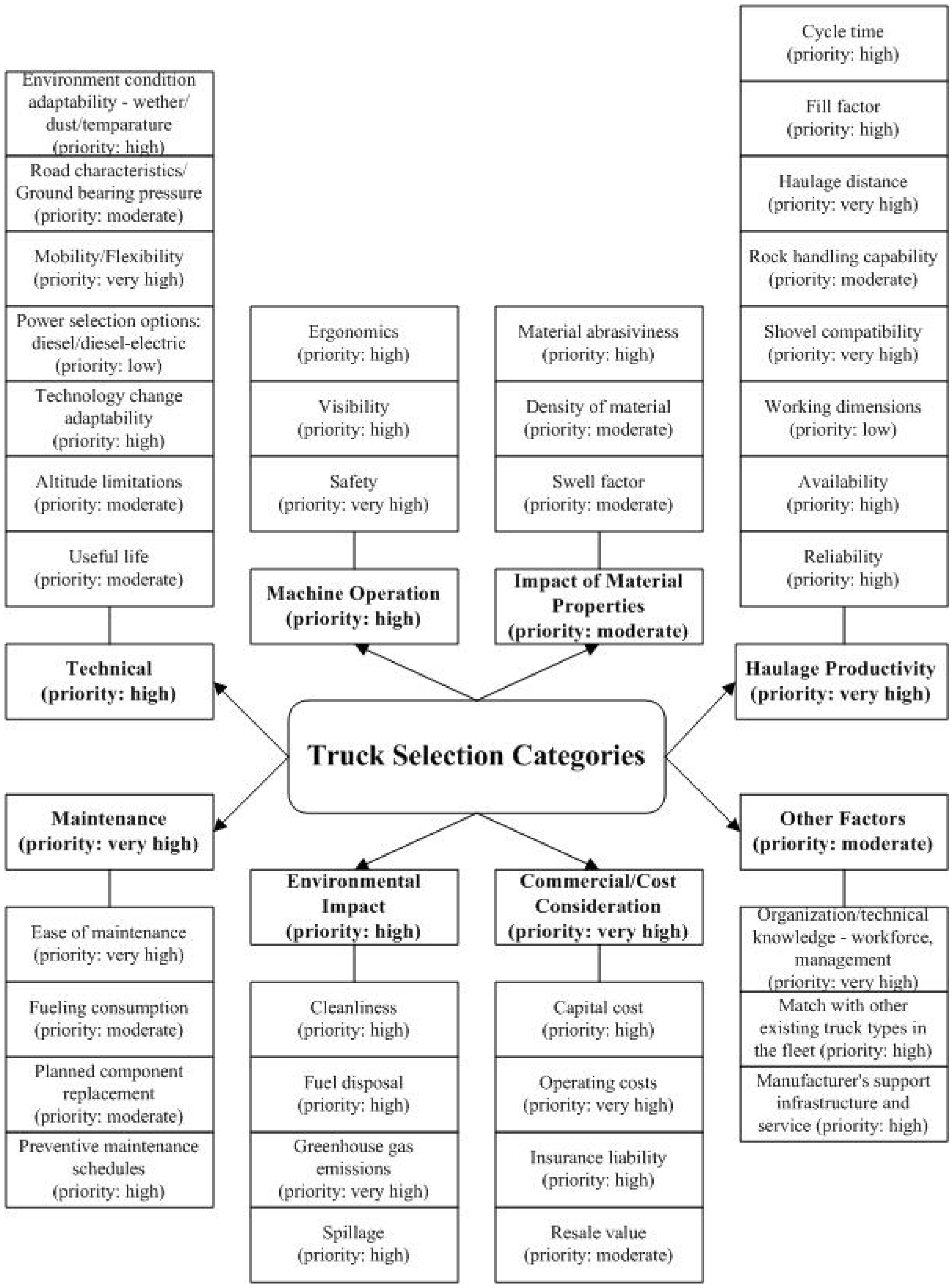

Fig. 4. *A hierarchical structure of truck selection attributes and priority ranks*

Step 2. The relational scale data for comparing the alternatives need to be generated. The analyst (knowledgeable party, decision-maker) performs pairwise comparisons of elements at each level relative to each activity at the next higher level in the hierarchy. In the importance of each criterion relative to system acceptance also needs to be established. In the AHP method, the relational scale of real numbers from 1 to 9 is used to systematically assign preferences. When comparing two attributes, or alternatives, *A* and *B*, with respect to an attribute *U*, in a higher level, the numerical relational scale is used (Saaty, 1980). This scale is shown in Table 2.

Table 2. *Numerical rational scale used in AHP method*

Numerical value	Description
1	*A* has the same importance as *B* with respect to *U*
3	*A* has slightly more importance than *B* with respect to *U*
5	*A* has more importance than *B* with respect to *U*
7	*A* has a lot more importance than *B* with respect to *U*
9	*A* totally dominates *B* with respect to *U*
1/3	*B* has slightly more importance than *A* with respect to *U*
1/5	*B* has more importance than *A* with respect to *U*
1/7	*B* has a lot more importance than *A* with respect to *U*
1/9	*B* totally dominates *A* with respect to *U*

Commonly, the intermediate numbers are used for finer resolution. The priorities/weights previously given for each category (attribute) and sub-category have to be translated into numerical AHP scales for a pairwise comparison. Table 3 shows the relationship among priorities and the numerical AHP scale to be used in this study.

Table 3. *Relationship among priorities and the numerical AHP scale for pairwise comparison*

Relationship between priorities	Numerical AHP scale
Very High to Very High	Default 1 (variation: 1 – 2)
Very High to High	Default 3 (variation: 2 – 4)
Very High to Moderate	Default 5 (variation: 4 – 6)
Very High to Low	Default 7 (variation: 6 – 8)
High to High	Default 1 (variation: 1 – 2)
High to Moderate	Default 3 (variation: 2 – 4)
High to Low	Default 5 (variation: 4 – 6)
Moderate to Moderate	Default 1 (variation: 1 – 2)
Moderate to Low	Default 3 (variation: 2 – 4)
Low to Low	Default 1 (variation: 1 – 2)

For the inverse priority relationship, a reciprocal numerical value is used as per Table 2

For the numerical analysis of priorities/weights, the Expert Choice® software is used. It should be noted that the software is used for the research purposes only. Table 4 depicts pairwise comparison among the main attributes with respect to the main objective, i.e. selection of the best truck type. In compliance with the AHP method, the lower half of the pairwise matrix always contains the reciprocal values, which are not directly presented in Tables. All the attribute values used in this research work are not prescriptive, and may be subject of change in order to adequately reflect specific circumstances. The pairwise comparisons are performed based on attribute ranking given in Figure 4, while numerical values are assigned in accordance with Table 3.

Table 4. *Pairwise comparison between main attributes*

	Technical	Machine Operation	Material Properties	Haulage Production	Maintenance	Environ. Impact	Costs	Other Factors
Technical	1.0	1.0	3.0	1/3	1/2	2.0	1/4	2.0
Machine Operation		1.0	3.0	1/2	1/2	1.0	1/5	2.0
Material Properties			1.0	1/4	1/4	1/4	1/6	1.0
Haulage Production				1.0	2.0	3.0	1/2	4.0
Maintenance					1.0	3.0	1/2	4.0
Environ. Impact						1.0	1/4	2.0
Costs							1.0	5.0
Other Factors								1.0

Inconsistency: 0.03

The AHP method enables calculating the inconsistency, i.e. degree of incoherence in judgments regarding pairwise comparison of the attributes. According to Saaty (1980), small consistency ratios (less than 0.1 is the suggested rule-of-thumb) do not drastically affect the ratings. The user has an option of redoing the comparison matrix if desired. The AHP technique also allows calculations without all completed judgments (attribution of numerical values for all pairwise comparisons), i.e. missing judgments are allowed. Table 5 shows pairwise comparisons for "Technical" category. Other pairwise comparisons related to remained attributes have been performed in a similar way. This is done by using both their assigned ranking given in Figure 1, and the corresponding relationship given in Table 3.

Table 5. *Pairwise comparison among attributes for "Technical" category*

	Environment condition adaptability	Road characteristics Ground pressure	Mobility-Flexibility	Power selection options	Technology change adaptability	Altitude limitations	Useful life
Ambient condition adaptability	1.0	3.0	1/3	5.0	2.0	4.0	3.0
Road characteristics / Ground bearing pressure		1.0	1/5	3.0	1/3	1.0	1/2
Mobility / Flexibility			1.0	7.0	3.0	5.0	5.0
Power selection options:				1.0	1/4	1/2	1/3
Technology change adaptability					1.0	3.0	2.0
Altitude limitations						1.0	1.0
Useful life							1.0

Inconsistency: 0.03

Step 3. An ***eigenvalue*** method is used to determine the relative priority of each attribute to attributes one level up in the hierarchy using the pairwise comparisons of the Step 2 (see equations 26 – 29). In addition, a consistency ratio is calculated and displayed. Figure 5 shows relative priority of main attributes related to the overall objective, i.e. the selection of the best truck type. The priorities are obtained based on the attribute values presented in Table 4. Figure 3 shows relative priority of the sub-attribute related to "Technical category". They are calculated based on the attribute values given in Table 5. The ranking of the other related sub-attribute categories presented in Figure 4 is carried out in a similar manner (see Figure 6).

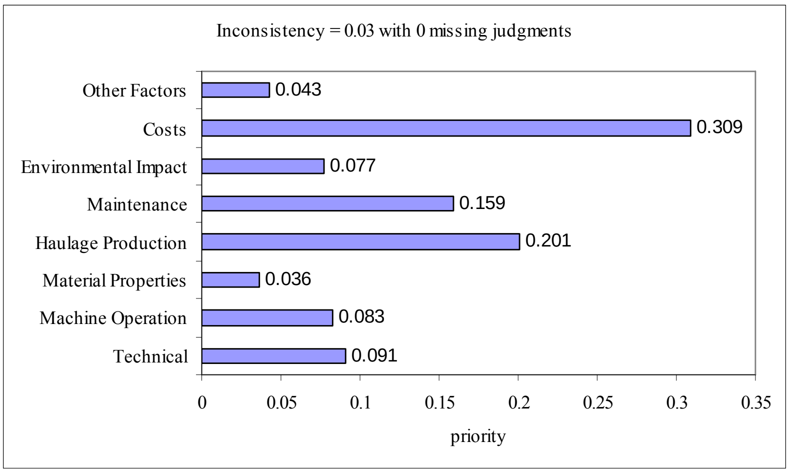

Fig. 5. *Relative priority of main attributes related to main objective*

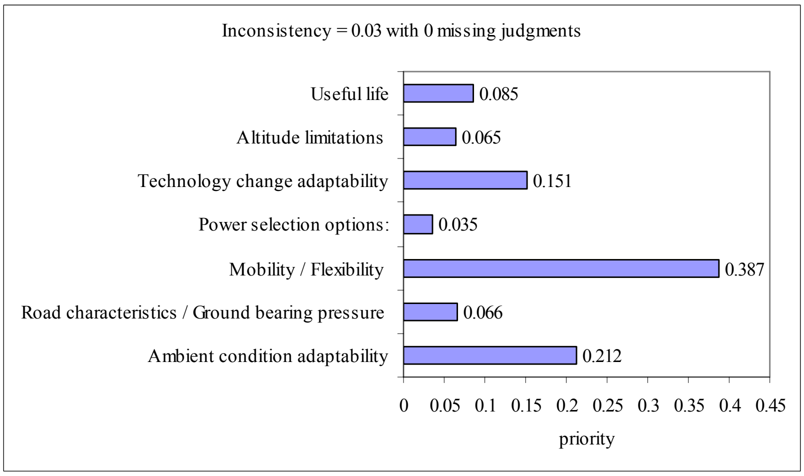

Fig. 6. *Relative priority of main attributes related to "Technical" category*

Step 4. The priorities/weights of the lowest level alternatives relative to the main objective are determined while alternatives are analyzed. This step will be presented through the case study of this paper.

3.3 A holistic selection approach using CTL and AHP

This section describes a holistic selection process which takes into consideration both CTL and AHP methods. The CTL method is integrated into the AHP as an addition and one more attribute to the eight attributes already established and presented in Table 4. Since the CTL puts into a relationship a number of key truck technical parameters, it is decided that this attribute should have the highest rank among the main selection attributes. A new state of pairwise table comparison is shown in Table 6. The numerical values used for the CTL in the pairwise comparison with other attributes may be weighted heavier if the analyst(s) considers that it better reflects actual circumstances. However, the maximal value of nine (9) as per Table 2 may not be exceeded.

Based upon the numerical values given in Table 6, the relative priority/weight of the main selection attributes including the CTL is calculated and results are shown in Figure 7. It should be noted that there are no changes in the priority of the sub-attributes, since their relationship is with the level-up attribute.

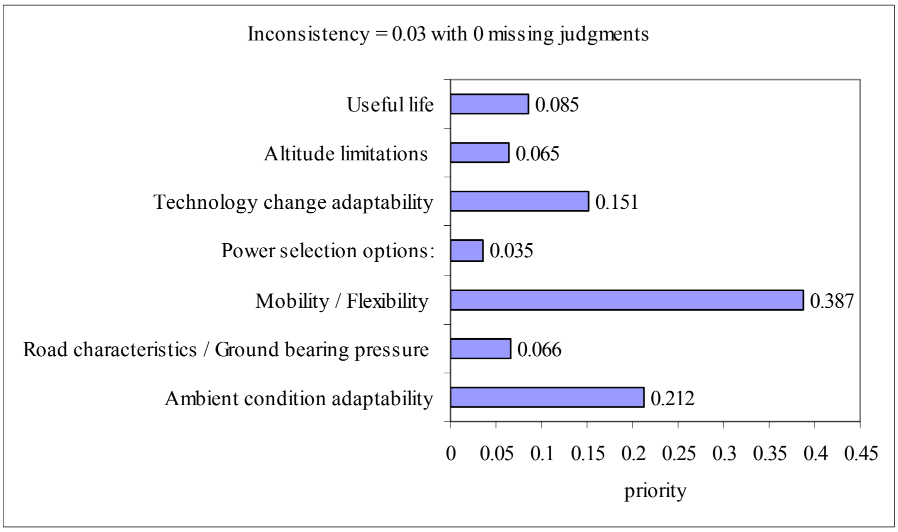

Fig. 7. *Relative priority of the main selection attributes including CTL*

Table 6. *Pairwise comparison table between main attributes including CTL*

	Technical Category	Machine Operation	Material Properties	Haulage Produc-tion	Mainte-nance	Environ. Impact	Costs	Other Factors	CTL
Technical Category	1.0	1.0	3.0	1/3	1/2	2.0	1/4	2.0	1/5
Machine Operation		1.0	3.0	1/2	1/2	1.0	1/5	2.0	1/5
Material Properties			1.0	1/4	1/4	1/4	1/6	1.0	1/7
Haulage Production				1.0	2.0	3.0	1/2	4.0	1/3
Maintenance					1.0	3.0	1/2	4.0	1/2
Environment Impact						1.0	1/4	2.0	1/6
Commercial Costs							1.0	5.0	1/2
Other Factors								1.0	1/7
CTL									1.0

Inconsistency: 0.03

The incorporation of the CTL into the selection attributes through the AHP method completes the development of a holistic selection process related to the mining trucks. The proposed selection methodology offers sufficient flexibility to be tailored in accordance with specific local conditions and needs. It integrates, in a structured and systematic manner, the most dominant tangible factors such as costs, productivity, material properties, maintenance etc, as well as some important intangible influence factors such as organization, level of knowledge, manufacturer's support, environment, etc. Those intangible factors and many others might sometimes have a decisive influence in a final selection. They can hardly be quantified through classical selection approaches, and final decision may not be the most appropriate. ***It is worth emphasizing that the proposed selection process is not prescriptive, but rather a powerful tool helping a decision-maker in making a proper final decision.***

The users of the proposed methodology may determine different weighting factors in accordance with their specific needs/context which may produce different end results. The approach enables such flexibility.

4. CASE STUDY

A case study is presented to illustrate an application of developed methodology for a selection of mining truck. The number of required trucks is previously determined, and is not included in this case study. The number of shovels, as well as their type and size are known, and match with both the number of trucks, and their chosen size. The project life, and project level of risk are not taken into consideration since they are equal for all the competing truck types (models). The selection process is carried out among the actual truck types and manufactures. However, the names of the latter have been changed to arbitrary names such as Manufacturer A, Manufacturer B, etc. A total of five truck types at the 180 t truck category are evaluated in this study. Table 7 presents the main parameters of the analyzed trucks. The coefficient of technical level (CTL) for each truck type is calculated using equations (6) through (25). The obtained results are presented in Table 8. Details of the CTL calculations are presented in Appendix 2.

Table 7. *Main design characteristics of mining trucks (180 t category)*

Truck type (Manufacturer)	Gross vehicle weight GVW (t)	Vehicle weight GV (t)	Maximum payload Q (t)	Struck Capacity V (m^3)	Heaped Capacity VM (m^3)	Motor power N (kW)	Top speed VT (km/h)
Manufacturer -A	318	131	187	73.4	105.0	1417	52.6
Manufacturer -B	324	136	188	79.9	115.1	1492	55.4
Manufacturer -C	317	130	186	92.0	123.0	1491	55.0
Manufacturer -D	324	138	186	77.7	110.9	1492	55.7
Manufacturer -E	331	129	202	76.5	107.8	1492	51.0

Table 8. *Coefficient of Technical Level (CTL) for analyzed trucks*

Truck type (Manufacturer)	Value of $K_{o(i)}$
Manufacturer -A	**0.00679 (minimum)**
Manufacturer -B	0.00704
Manufacturer -C	0.01568
Manufacturer -D	0.01336
Manufacturer -E	0.00738

The CTL favors the minimal values of analyzed alternatives (Eq. 25), while the AHP method prefers the maximum values using a numerical scale for a pairwise comparison (Table 2). Incorporation of the CTL into the AHP method starts with an attribution of scale numerical values to $K_{o(i)}$ for the analyzed alternatives. The AHP numerical pairwise scale is used, where numerical value of 9 represents a maximal value (Table 2). The conversion of the $K_{o(i)}$ values into the AHP numerical scale is developed and presented as follows:

- Step value (h) in the range from $K_{o(min)}$ to $K_{o(max)}$

$$h = \frac{K_{o(\max)} - K_{o(\min)}}{9} \qquad (30)$$

where:

$K_{o(max)}$, $K_{o(min)}$ – maximum and minimum Criterion values among the analyzed alternatives

- Rank number of an alternative (***RNA$_{(i)}$***) in accordance to the AHP numerical scale (integer values are used as per Table 2)

$$RNA_{(i)} = INT\left(9 - \frac{K_{o(i)} - K_{o(\min)}}{h}\right) \qquad (31)$$

or

$$RNA_{(i)} = INT\left[\frac{9 \times \left(K_{o(\max)} - K_{o(i)}\right)}{K_{o(\max)} - K_{o(\min)}}\right] \qquad (32)$$

The minimum CTL value $K_{o(min)}$ gives the maximum $RNA_{(max)}$. The minimum allowed value for RNA is $RNA_{(min)}$ = 1. In the case where the $RNA_{(i)}$ value is calculated as zero according to Eq. (301, (the case of $K_{o(i)}$ = $K_{o(max)}$), then the value of 1 has to be assigned to $RNA_{(i)}$, i.e. $RNA_{(i)}$ = 1.

Scoring values ($SV_{A\to B}$) for a pairwise comparison between two alternatives

$$SV_{A\to B} = \begin{cases} RNA_A - RNA_B + 1 \ \ for \ \ RNA_A - RNA_B \geq 0 \\ \dfrac{1}{RNA_B - RNA_A + 1} \ \ for \ \ RNA_A - RNA_B < 0 \end{cases} \qquad (33)$$

The obtained values through the Eq. (33) regarding rank numbers of the analyzed alternatives are presented in Table 9.

Table 9. *Rank numbers of the analyzed alternatives*

Manufacturer	$K_{o(i)}$	$RNA_{(i)}$
Manufacturer -A	0.00679	9
Manufacturer -B	0.00704	8
Manufacturer -C	0.01568	1
Manufacturer -D	0.01336	2
Manufacturer -E	0.00738	8

With adapting and incorporating the CTL approach into the AHP method, all the required elements for performing an alternative comparison against priority criteria are defined (Step 4 of the AHP method). The pairwise comparison of the analyzed alternatives against the lowest level attributes is carried out for all previously listed subcategories. As an example, Table 10 depicts a pairwise comparison for the CTL attribute, and Figure 8 presents the corresponding weight/priority of the alternatives against the CTL attribute. The values in Table 10 were calculated using Eq. (33).

Table 10. *Pairwise comparison of alternatives for the CTL attribute*

	Manufacturer A	**Manufacturer B**	**Manufacturer C**	**Manufacturer D**	**Manufacturer E**
Manufacturer A	1.0	2.0	9.0	8.0	2.0
Manufacturer B		1.0	8.0	7.0	1.0
Manufacturer C			1.0	1/2	1/8
Manufacturer D				1.0	1/7
Manufacturer E					1.0

Inconsistency: 0.02

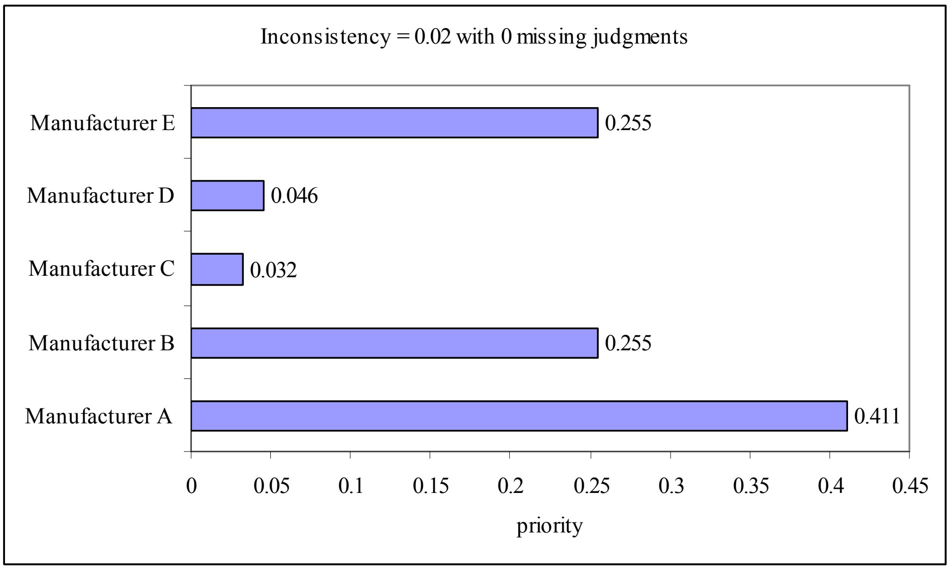

Fig. 8. *Priority/weight of the alternatives against the CTL attribute*

Final ranking of the analyzed alternatives is obtained through their pairwise comparison against all the attributes. Figures 9 and 10 show the ranks of the alternatives against the selection attributes.

A sensitivity analysis for the final results is performed in order to test robustness of the solution. No change has been observed on the final manufacturer's rating within a variation range of +/- 20% regarding the main attribute importance. Thus, the proposed choice of a mining truck type may be considered as stable. The obtained results clearly show that the truck offered by the Manufacturer A is to be recommended for a final selection.

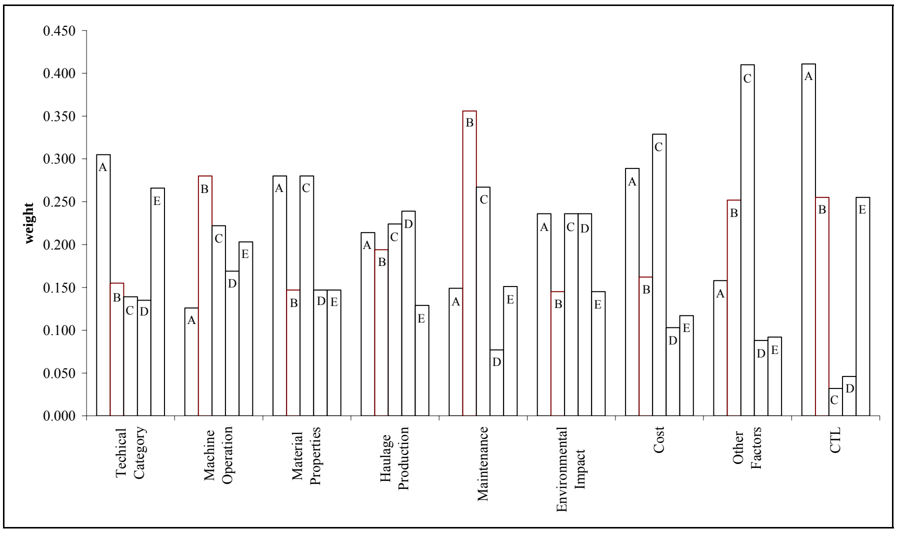

Fig. 9. *Individual ranks of the alternatives against the selection attributes*

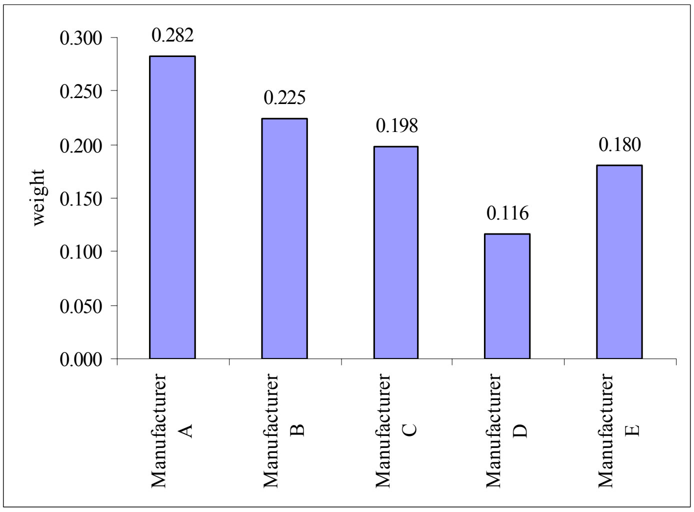

Fig. 10. *Overall ranks of the alternatives against the selection attributes*

Appendix 1. *Design data regarding analyzed trucks*

No	TRUCK	GROSS VEHICLE WEIGHT GVW (t)	VEHICLE WEIGHT GV (t)	MAXIMUM PAYLOAD Q (t)	CAPACITY SAE V (m3)	CAPACITY SAE (2:1) VM (m3)	MOTOR POWER N (kW)	TOP SPEED VT (km/h)
1	CAT 769D	68.18	30.28	37.9	17	24.2	380	75
2	CAT 771D	73.97	33.3	40.67	20.2	27.5	380	56
3	CAT 773D	92.53	39.19	53.34	26.6	35.2	509	66
4	CAT 775D	106.59	43.22	63.37	31.4	41.5	541	66
5	CAT 777D	161.03	64.36	96.67	42.1	60.1	746	60
6	CAT 785C	249.48	95.72	153.76	56.9	78.2	1082	54
7	CAT 789C	317.52	130.94	186.58	73.4	105	1417	52.57
8	CAT 793C	376.49	144.65	231.84	96	129	1715	53.6
9	CAT 797	555.99	229.46	326.53	173	220	2535	64
10	OK-K35	53.86	22.86	31	15.2	19.4	296	55
11	OK-K40	70.7	34.4	36.3	17.2	23.9	354	60
12	OK-K45	77.1	36.29	40.82	19.6	26	392	67
13	OK-K60	95.43	41	54.43	26	35	485	60
14	OK-K100	159.34	68.62	90.72	41.7	57	783	46
15	EUCLID R130B	226.8	98.6	128.2	54.1	78.4	1007	49.3
16	EUCLID R170C	278.96	122.06	156.9	72.21	101.95	1343	55.4
17	EUCLID R190C	324.32	136.22	188.1	79.9	115.1	1492	55.4
18	EUCLID R260	385.92	147.92	238	92.9	131.9	1864	48.8
19	EUCLID R280	435.46	179.76	255.7	106.1	148.2	2013	62
20	TEREX TR35	55	23.25	31.75	15.3	19.4	261	55.5
21	TEREX TR40	70.7	34.4	36.3	17.2	23.9	354	58
22	TEREX TR45	76.11	35.28	40.83	19.6	26	392	60
23	TEREX TR60	95.68	41.25	54.43	26	35	485	57
24	TEREX TR70	112.69	47.69	65	29	41.5	567	57
25	TEREX TR100	159.34	68.62	90.72	41.6	57	783	47.6
26	UNIT RIG MT-3000	206.3	97.45	108.85	42.8	69.6	840	50
27	UNIT RIG MT-3300	241.22	105.22	136	63	87	1025	53
28	UNIT RIG MT-3600B_1	270.52	116.35	154.17	62	98	1084	56.4
29	UNIT RIG MT-3600B_2	294.79	116.35	178.44	77	111	1382	56.4
30	UNIT RIG MT-3700B_1	302.92	130.59	172.33	77	111	1193	55
31	UNIT RIG MT-3700B_2	317	131.07	185.93	92	123	1491	55
32	UNIT RIG MT-4400	392.29	156.29	236	95	139	1706	48.3
33	UNIT RIG MT-5500	545	216.78	328.22	158	218	2014	65
34	KOMATSU HD325-6	72.13	32.05	40.08	18	24	379	70
35	KOMATSU HD465-5	96.1	41.1	55	25	34.2	551	70
36	KOMATSU HD605-5	106.68	46	60.68	29	40	551	70
37	KOMATSU HD785-5	166.4	69.93	96.47	38.6	60.1	807	61.9
38	KOMATSU HD1500-5	249.48	100.46	149.02	54	78	1108	58
39	KOMATSU 630E	296.36	124.03	172.33	72.31	103.3	1492	51
40	KOMATSU 330M	166.4	69.05	97.35	38.6	60.1	783	61.9
41	KOMATSU 530M	249.48	100	149.48	54	78	1082	58
42	KOMATSU 730E	324.32	138.03	186.29	77.7	110.93	1492	55.7
43	KOMATSU 830E	385.85	154.4	231.45	117	147	1887	56.9
44	KOMATSU 930E-2	498.96	202.32	296.64	148	211.2	2014	64.5
45	LIEBHERR T252	331	129	202	76.5	107.8	1492	51
46	LIEBHERR T262	386	152	234	84	119	1864	51
47	LIEBHERR T272	442	151	291	164	175	2014	68
48	LIEBHERR T282	567	203	364	122	183.5	2610	64
49	BELAZ 7540D	51.75	21.75	30	15	18.5	290	60
50	BELAZ 7548C	71.48	29.48	42	27.5	33	448	60
51	BELAZ 75553	92	37	55	25	34.2	552	60
52	BELAZ 75305	340	140	200	80	114	1492	43

Appendix 2. *Details of CTL calculations for analyzed 180t trucks*

Manufacturer	GROSS VEHICLE WEIGHT GVW (t)	VEHICLE WEIGHT GV (t)	MAXIMUM PAYLOAD Q (t)	CAPACITY SAE V (m^3)	CAPACITY SAE (2:1) VM (m^3)	MOTOR POWER N (kW)	TOP SPEED VT (km/h)	k_1 = Q/GV	k_2 = Q/N	k_3 = GVW/N	k_4 = VM/GV
Manuf-A	318	131	186.6	73.4	105.0	1417	52.57	1.4249	0.1317	0.2241	0.8019
Manuf-B	324	136	188.1	79.9	115.1	1492	55.40	1.3809	0.1261	0.2174	0.8450
Manuf-C	317	131	185.9	92.0	123.0	1491	55.00	1.4186	0.1247	0.2126	0.9384
Manuf-D	324	138	186.3	77.7	110.9	1492	55.70	1.3496	0.1249	0.2174	0.8037
Manuf-E	331	129	202.0	76.5	107.8	1492	51.00	1.5659	0.1354	0.2218	0.8357
							Mean:	***1.3958***	***0.1222***	***0.2102***	***0.8580***

Manufacturer	NGVW	Z1	NQ	Z2	NVM	Z3	GVQ	Z4	Koa
Manuf-A	1429.63	-8.834E-03	1 429.74	-8.910E-03	1468.58	-3.512E-02	129.90	8.001E-03	***0.00146***
Manuf-B	1458.46	2.300E-02	1 439.98	3.613E-02	1578.85	-5.501E-02	130.81	4.137E-02	***0.00657***
Manuf-C	1427.42	4.454E-02	1 425.36	4.605E-02	1661.50	-1.026E-01	129.51	1.203E-02	***0.01478***
Manuf-D	1458.46	2.300E-02	1 427.79	4.498E-02	1533.95	-2.735E-02	129.73	6.400E-02	***0.00740***
Manuf-E	1486.78	3.508E-03	1 532.96	-2.672E-02	1499.67	-5.114E-03	139.00	-7.196E-02	***0.00593***

Manufacturer	Z5	K5	Z6	K6	Z7	K7	Z8	K8	Z9	K9	Kob
Manuf-A	-2.046E-02	1.751E-07	-7.202E-02	2.690E-05	-6.208E-02	1.485E-05	7.001E-02	4.902E-03	1.967E-02	3.870E-04	***0.0053***
Manuf-B	1.081E-02	1.168E-04	-3.080E-02	8.993E-07	-3.314E-02	1.207E-06	1.548E-02	2.396E-04	1.056E-02	1.116E-04	***0.0005***
Manuf-C	-1.605E-02	6.644E-08	-2.014E-02	1.645E-07	-1.148E-02	1.737E-08	-8.567E-02	5.386E-05	2.910E-02	8.470E-04	***0.0009***
Manuf-D	3.419E-02	1.169E-03	-2.138E-02	2.089E-07	-3.314E-02	1.207E-06	6.765E-02	4.577E-03	1.489E-02	2.216E-04	***0.0060***
Manuf-E	-1.086E-01	1.393E-04	-9.749E-02	9.032E-05	-5.266E-02	7.687E-06	2.678E-02	7.170E-04	2.221E-02	4.934E-04	***0.0014***

Manufacturer	Ko		Ko(i)/Ko(min)	Ko(min)/Ko(i)	Manufacturer
Manuf-A	***0.00679***	***MIN***	**1.0000**	**1.0000**	**Manuf-A**
Manuf-B	***0.00704***		**1.0377**	**0.9636**	**Manuf-B**
Manuf-C	***0.01568***		**2.3109**	**0.4327**	**Manuf-C**
Manuf-D	***0.01336***		**1.9695**	**0.5077**	**Manuf-D**
Manuf-E	***0.00738***		**1.0874**	**0.9196**	**Manuf-E**

5. CONCLUSIONS

This paper presented the research results on a novel approach for the truck selection in surface mining operations. Both the coefficient of technical level (CTL), and Analytic Hierarchy Process (AHP) multi-attribute decision-making method were used in the study. The CTL was expressed through a mathematical function which takes into consideration the relationships among truck parameters such as gross vehicle weight, total vehicle weight, maximum payload, struck capacity, heaped capacity, motor power, and top vehicle speed. In order to select a haul truck with the most favorable technical characteristics, this mathematical function was set to tend towards a minimum. The second part of the analysis focused on various attributes such as: technical, machine operation, material properties productivity, maintenance, environment, commercial/cost consideration and miscellaneous factors. Each of these attributes was subdivided in many sub-categories.

Both the attributes and their sub-categories were assigned with a relative priority, and a pairwise comparison between them was performed. The Analytic Hierarchy Process (AHP) technique was also used in both quantifying their impact, and in proposing a final solution. The third part of the study included an integration of CTL into the AHP where CTL attribute was assigned with the highest rank among the main truck selection attributes. A case study was performed in order to demonstrate the usefulness of the suggested methodology. The proposed selection methodology integrated, in a structured and systematic manner, the most dominant factors such as costs, productivity, material properties, maintenance, etc, as well as some important intangible influence factors such as organization, level of knowledge, manufacturer's support, environment, etc. This methodology can be used as a powerful tool helping mine operators to examine strengths and weaknesses of certain types of the trucks by comparing them according to appropriate criteria. It also can be used by mine decision-makers in pre-selection of a specific model when buying a new mining truck, or by the truck manufacturers to analyze the potential benefits of envisaged modifications concerning an existing truck model and/or development of a new truck model. It is worth mentioning that a similar approach may also be developed and applied to other types of mining and material handling equipment.

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TECHNICAL - ECONOMIC INDICATORS IN EXCAVATION THE SAFETY PILLARS OF THE LEAD ZINC DEPOSITS IN AN EXAMPLE OF THE MINE TREPCA – STARI TRG**

Abstract

The basic characteristic of the current underground mining is the ore mining at increasing depths and less content of useful components in it. With increasing depth, significantly increase the operating costs. Therefore, the best approach is to find the technical and technological solutions aimed to reduce the operating costs.

All of these circumstances require finding a solution of mining the mineral resources that remain after the primary stage of exploitation, e.g. security pillars, security walls, chambers, etc.

In the mine Trepca - Stari Trg, there are more than 70 safety pillars with permanent leave the new ones in the progress of excavations. It can be stated with certainty that the pillars include 15% of ore mass, which will b mined in the secondary stage of exploitation.

Key words: *mining, safety pillars, mining, secondary phase.*

INTRODUCTION

For ore bodies in the mine Trepca of large areas and spanning from the floor to the hanging wall, as temporary safety means of stope in the primary stage of operation, the safety pillars are left arranged in a chess order, size 10 x 10 m. The distance between rows is from 12 - 16 m and the distance between pillars in the order is 16 - 20 m.

In the previous exploitation of the mine Trepca - Stari Trg, the method of roof excavation in horizontal floors with the leveling and filling was mainly applied, where the height difference between the horizons is 60 m.

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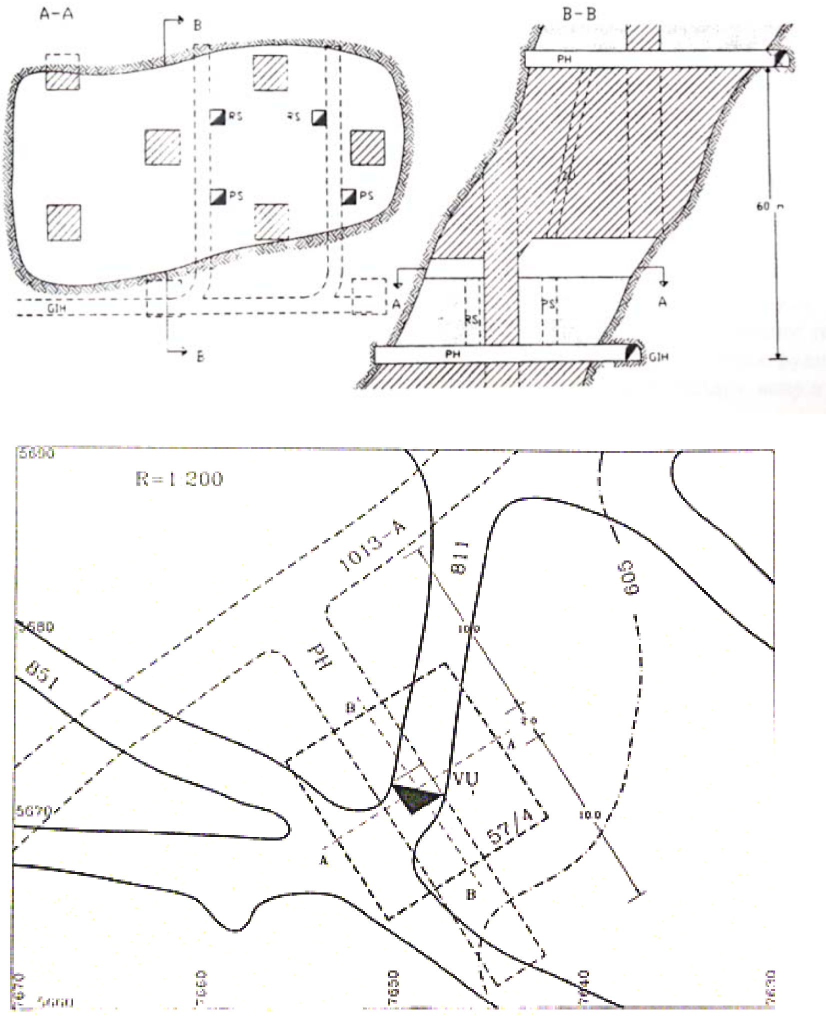

Figure 1. *The Trepca mining method with the location of safety pillar*

Taking into account the previous experience in excavation the safety pillars and characteristics of modern drilling equipment and mobile platforms *ALIMAC* , the block caving mining method with massive folding of ore was developed by horizontal blast holes in the fan-like arrangement. Very important information was obtained during the trial excavation of safety pillar by the block caving method with the use of short blast holes. It was stated during this, in several instances, that after excavation and unloading of ore from magazine, the surrounding stowing is strong and there is any caving.

Due to a large number of safety pillars and similar geological - technical conditions of exploitation, a distinctive pillar (57 / 4) is chosen in this paper, which will serve as the basis for exploitation of other safety pillars. Calculation of the ore reserves for this pillar is given in Table 1.

Table 1. *Ore reserves in a pillar (57/4)*

Pillar No.	Area of cross cut [m^2]	Pillar height [m]	Bulk density of ore [t/m^3]	Reserves in pillar [t]	Average metal content			Metal quantity		
					Pb [%]	Zn [%]	Ag [g/t]	Pb [t]	Zn [t]	Ag [kg]
57/4	100	35	3.7	12,950	5.37	10.38	242	695	1.344	3.133

In order to increase the productivity and safety at work, the block caving method with development of horizontal blast holes in the fan-like arrangement is proposed for excavation the safety pillar. This method allows the mass blasting of a number of blasting belts, what can achieve the desired capacity of excavation.

TECHNICAL – ECONOMIC INDICATORS OF EXCAVATION

The technological process of excavation the safety pillar is a set of certain working operations that define the execution schedule of the same, what is shown by the scheme of work organization on excavation. The above process involves performing the following operations: drilling, blasting, ore unloading, loading and transportation of ore, stope insurance, backfilling of stope, etc.

Low ore losses are significant because they extend the service life of mine, and the lost raw material is an irreversible loss of natural wealth of the country. Provision of adequate production capacity per volume and quality has great impact on the cost of products, as well as providing the general production plan.

Labor force in the underground mining makes a significant production cost, why it is necessary in the choice of t technological process of work to try mechanization of many manufacturing processes and to use as much as possible the machinery. In the other words, it is necessary to implement an efficient organization of work. The reduction in production costs also affects the degree of ore dilution, because with higher dilution on the excess of waste rock that is obtained during exploitation, more labor, energy and materials are spent without any need.

Apparent efficiency can be achieved when only high grade parties are mined in some deposit, and low grade parties are left. Such work is at given time can achieve high economic effects, but in later stage of production that can have very negative consequences and further exploitation can be questioned. Therefore, for any mass or selective exploitation, mining of deposit must be planned and carried out systematically.

RECOVERY INDICATORS

Ore recovery from the mine block, as a rule, is expressed by the coefficient of ore recovery minerals *(i_r)*. This ratio is expressed in absolute values or percentages. Practically, if the total ore reserve of some block are marked with *T*, the loss of ore, which was created by the excavation (unexcavated part of the ore reserves) with T_g, and part of the ore that is produced from the block with $T_{\check{c}}$, then between them there is the following relation:

$$T = T_{\check{c}} + T_g \, [t]$$

Coefficient of ore recovery is the ratio between the quantity of produced ore and total quantity of ore in mineral block, expressed in %, i.e.:

$$i_r = \frac{T_{\check{c}}}{T} \times 100\%$$

Ore loss coefficient is the ratio between the quantity of lost ore and total quantity of ore in the mineral block, i.e.:

$$g_r = \frac{T_g}{T} \times 100\%$$

The following dependency is from the given coefficients:

$$i_r + g_r = 1$$

Based on Table 1, the geological ore reserves in the safety pillar *57/4* amounted to *12,950 t.* A certain amount of ore will remain trapped in the vicinity of ore hopper bars caused by manufacturing the same. The specified amount of ore (T_g) is *260 t.* Based on this, the following values can be calculated:

The amount of ore that can be produced from the total ore reserves of security pillar:

$$T_{\check{c}} = T - T_g = 12.950 - 260 = 12.690 \; [t]$$

Coefficient of ore loss:

$$g_r = \frac{T_g}{T} \times 100\% = \frac{260}{12.950} \times 100\% = 2\%$$

Coefficient of ore recovery:

$$i_r = \frac{T_{\check{c}}}{T} \times 100\% = \frac{12.690}{12.690} \times 100\% = 98\%$$

DEPLETION COEFFICIENT

Depletion presents the ratio of waste amount that is in the run-of-mine ore according to the total amount of run-of-mine ore and it is expressed by the ore depletion coefficient.

$$O_r = \frac{T_2}{T_1} \times 100\%$$

Where:

T_2 – quantity of waste in the run-of-mine ore,

T_1 – total quantity of run-of-mine ore ($T_1 = T_{\check{c}} + T_2$).

Depletion can also be expressed as the reduction of metal content in the produced run-of-mine ore, compared to the metal content in the ore block. In this case it was the uncovering of the pillars of security, and on the basis of previous monitoring and behavior filling material around the pillar and it was found that after emptying the magazine space, the filling material continued to hold without demolition. But, during ore unloading, it entails a smaller amount of filling material where it is accounted that it will not exceed a value of *3%* of the value of ore produced ($T_{\check{c}}$), so that the coefficient of depletion will be:

$$O_r = \frac{T_2}{T_1} \times 100\% = \frac{381}{13.071} \times 100\% = 3\%$$

COEFFICIENT OF PREPARATION

Coefficient of preparation is the ratio of preparation works for the certain mining method to the total amount of ore that will be produced from a stope or block which is located between two horizons.

$$k_p = \frac{1000 \times P_r}{T_1} \; [mm/t]$$

where:

P_r – volume of preparation works [m],

T_1 – quantity of the produced run-of-mine ore [t]

Coefficient of preparation is used in design of mining method for calculation the participation of preparation works in the mining costs, and with active mines for planning the annual amount of preparatory works in order to maintain or increase the current production capacity. Due to the differences between some safety pillars in the deposit, the preparation coefficient, calculated in the example of excavation the safety pillar *57/4* , will not be relevant in the case of mining the other security pillars. In excavation of this safety pillar, it is necessary to do the next volume of preparation works:

- Drifting of 10 m access drift (at the II level of horizon),
- Raising of 56 *m* vertical raise,
- Development of chamber for accommodation the ALIMAK platform with length and cross section corresponding to drifting of *10 m* access corridor,
- Development of funnel ore chute (excavation of $160\ m^3$ ore),

Coefficient of preparation for corridors and raises is expressed in [*mm/t*] of produced ore, and for various extensions, in this case funnel ore chute, it can be expressed in [m^3/t] of produced ore.

Coefficient of preparation for corridors is:

$$k_p = \frac{1000 \times P_r}{T_1} = \frac{1000 \times 20}{13.071} = 1{,}5\ [mm/t]$$

Coefficient of preparation for raises is:

$$k_p = \frac{1000 \times P_r}{T_1} = \frac{1000 \times 56}{13.071} = 4{,}3\ [mm/t]$$

Coefficient of preparation for funnel ore chutes is:

$$k_p = \frac{P_r}{T_1} = \frac{160}{13.071} = 0{,}12\ \left[m^3/t\right]$$

PRODUCTIVITY OF MINING METHOD AND PRODUCTION CAPACITY

Productivity of mining method is usually expressed by the coefficient of production intensity, which represents the ratio of produced ore in one block during the year per unit of mining surface. In the case, the excavation of safety pillar, the time required for its excavation is imposed as better indicator of mining method.

Total time for excavation the safety pillar consists on time for development the preparation works, excavation time and the time needed for assembly and disassembly of equipment and other ancillary works.

Time for development the preparation works depends on a position of safety pillar and distances of the existing mining facilities. In this case, it is necessary to develop *20 m* of access corridor and *56 m* of vertical raise.

Designed technology of room development predicted that the specified works will be carried out in the next period:

- Drifting of access corridor
  14 working shifts
- Raising of vertical raise
  37 working shifts
- Performing of auxiliary operations
  6 working shifts

Therefore, realization of preparation works is:

$$t_p = 14 + 37 + 6 = 57\,shifts = 19\ working\ days$$

Time of excavation consists of the time required to perform the work operations on excavation (drilling of blast holes, blasting, loading and transport of the excess lowered ore) and the time needed for loading and transport of block caved ore.

According to the scheme of work organization, it is planned to perform the breaking of a belt of blasting, which involves performing the following work operations: drilling of blast holes, blasting, loading and transport of the excess lowered ore. By choosing the parameters in the safety pillar, 17 blasting belts are defined, so the time required for excavation of the same will be:

- Excavation
  17 x 2 = 34 working shifts,

Development of funnel extension 4 working shifts
(*what means that to = 38 working shifts*)

Quantity of block caved ore in the safety pillar could be calculated by the following formula:

$$V_m = (P - p) \times H_s \times k_r - V_u \ \left[m^3\right]$$

$$V_m = (100 - 4) \times 35{,}5 \times 1{,}6 - 1.861 = 3.592 \left[m^3\right]$$

$$Q_m = \frac{3.592 \times 3{,}7}{1{,}6} = 8.306{,}5 [t]$$

where:

V_m – volume of ore after finalization of pillar excavation

P – surface of safety pillar, *100* [m^2]

p – surface of vertical raise, *4* [m^2]

H_s – height of safety pillar, *3.5* [m]

k_r – coefficient of ore looseness, [*1.6*]

γ – bulk density of ore, *3.7* [t/m^3]

V_u – quantity of lowered ore that is loaded during excavation of pillars

v_u – excess of lowered ore volume in a belt of blasting

n_s – number of working shifts for ore loading

$$V_u = v_u \times n_s = 109{,}44 \times 17 = 1.861 \left[m^3\right]$$

Discharge time (unloading) of block caved ore:

$$t_i = \frac{Q_m}{Q_u} = \frac{8.306{,}5}{400} = 20{,}77 \approx 21 \, shifts$$

Total time of excavation, loading and transport of block caved ore is:

$$t_u = t_0 + t_i = 38 + 21 = 59 \ shifts \approx$$
$$\approx 20 \ working \ days$$

The required time for assembly and disassembly of equipment - 6 working shifts are anticipated, and for ancillary activities such as the extension of pipelines for air and water, cleaning of access roads - 9 working shifts are anticipated. Therefore, this time is:

$$t_r = 6 + 9 = 15 \, shifts = 5 \, working \ days$$

In accordance with the adopted scheme of work organization, total time of excavation the safety pillar is:

$$T_0 = t_p + t_0 + t_r = 57 + 59 + 15 =$$
$$= 131 \ shifts = 44 \ working \ days$$

EXCAVATION EFFECTS

Evaluation the work productivity in the underground mining can be expressed by the effect that represents the amount of products by realized eight hour wage. During this, there are the following effects: the effect of excavation, effect of various operations or processes, pit effect and the effect of excavation.

The effect of excavation represents the mine productivity and it includes all spent wages on preparation, drilling in stopes, blasting, ore loading, support, backfilling and all ancillary works at the excavation.

$$U = \frac{T}{n} \left[{}^{t}\!/\!_{wage} \right]$$

where:

U – effect in carrying out the certain work operation,

T – ore quantity that is excavated hiring n wages,

n – number of wages required for excavation T quantity of ore

Scheme of work organization on excavation the safety pillar provides breaking of a blasting belt (675 t of ore) during two work shifts and, accordingly to this, the effects of drilling, blasting and loading operations with transport are determined.

For drilling the blast holes of a blasting belt, hiring of 3 workers in two shifts is anticipated, or 6 wages are necessary to perform this operation, and the effect is:

$$U_b = \frac{675}{6\,wage} = 112{,}5\left[\frac{t}{wage}\right]$$

To perform the work operation of a belt blasting, three workers will be engaged in a shift workers who will carry out filling of blast holes, install the explosion boosters, blockage of blast holes, their connection and firing. The effect of blasting has the following value:

$$U_m = \frac{675}{3\,wage} = 225\left[\frac{t}{wage}\right]$$

One operator will be engaged on loading and transport of ore. These work operations are performed during one working shift such as the quantity of 253 *t* of ore is loaded and transported to the central ore chute. Therefore, the effects on the ore loading and transport will have the same values:

$$U_u = 253\left[\frac{t}{wage}\right], \quad U_t = 253\left[\frac{t}{wage}\right]$$

CONCLUSION

Technical - economic parameters of excavation the safety pillars are essentially the analysis of all factors and indicators that are important to achieve the future economic effects in their exploitation.

It can be noted that in recent year, according to generally accepted principles of modern society, the economic parameters and profitability of production represent the dominant factors influencing the final decision on whether to perform or not perform the exploitation of a certain part of deposit.

Based on the results obtained by the implementation of designed parameters, the general conditions will be determined for future exploitation as well as the mining method. The value of indicators will be approximately determine the mining capacity, operating costs per 1 t of excavated minerals, as well as the total and specific investments required for excavation.

This work presents the basic parameters and indicators of the used mining method of safety pillars as well as their analysis and comparison.

For specific work conditions in the mine Trepca, a solution was proposed for mining the safety pillars using the block caving method with development of horizontal blast holes in the fan-like arrangement. The proposed solution guarantees smaller volume of preparation works, high safety, satisfactory production capacity and significantly reduction of operating costs.

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DEFINISANJE SISTEMA UTICAJNIH USLOVA KOD IZBORA METODE OTKOPAVANJA KRATKIM MEHANIZOVANIM ČELOM U RUDNICIMA UGLJA

Izvod

U radu se daje prikaz sistema uticajnih uslova kod izbora metode otkopavanja kratkim mehanizovanim čelom sa vertikalnom koncentracijom u rudnicima uglja u Srbiji.

Ova metoda je fleksibilna i prilagodiva složenim uslovima eksploatacije, kakvi su u većini sada aktivnih ležišta. Cilj provedenih istraživanja u okviru ovog rada je da se da nov pristup predmetnoj metodi otkopavanja i njenoj široj primeni.

Ključne reči: *ugalj, podzemna eksploatacija, kratko mehanizovano čelo*

1. UVOD

Uspešno odabran skup tehnoloških operacija u procesu otkopavanja za određenu radnu sredinu i određene prirodno-geološke uslove, koja se definiše kao metoda otkopavanja, obezbeđenjem visokog stepena, koncentracije tehnike i tehnologije proizvodnog rada treba da postigne:

- što veći smenski, odnosno dnevni kapacitet otkopne jedinice;
- što veću produktivnost po zaposlenom radniku na otkopu;
- što manje gubitke uglja;
- što veću sigurnost rada u svim tehnološkim operacijama na dobijanju uglja;
- zadovoljavajuću ekonomičnost.

Osnovni pravci razvoja sistema otkopavanja ležišta uglja podzemnim načinom usmeren je ka unapređenju metoda širokih čela sa kompleksnom mehanizacijom, sa različitim dužinama otkopa i visinama otkopavanja.

Sagledavajući konkretne prirodno-geološke uslove u aktivnim ležištima mrkog uglja u Srbiji došlo se do saznanja o mogućnosti uvođenja kratkih mehanizovanih čela u pojedinim delovima ležišta, te se u ovom radu određuje sistem uticajnih uslova kod izbora navedene metode, koja ima značajne prednosti u odnosu na sada primenjene klasičine stubne metode.

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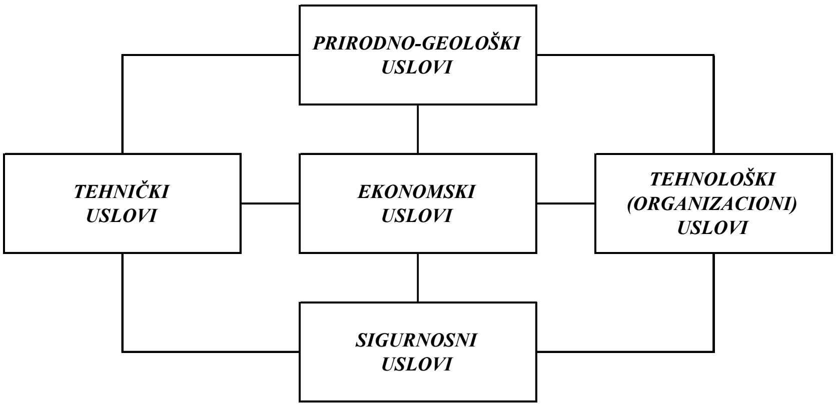
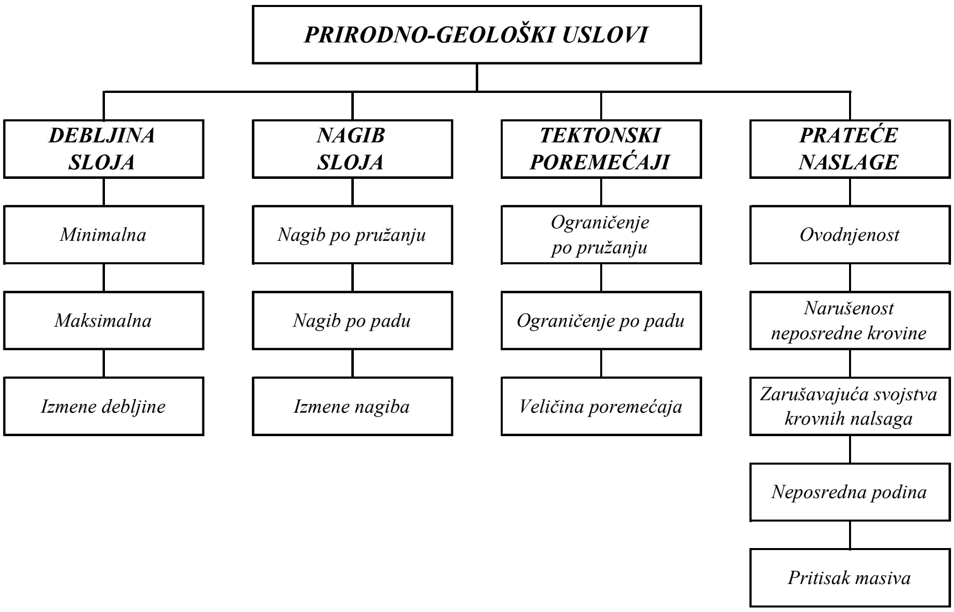
2. OPIS METODE KRATKIH MEHANIZOVANIH ČELA

Kratko mehanizovano čelo za razliku od širokog čela dimenzioniše se sa dužinama otkopa od 30-50 m, i obično se sa jednom transportno-ventilacionom prostorijom veže za ostale prostorije u otkopnom polju.

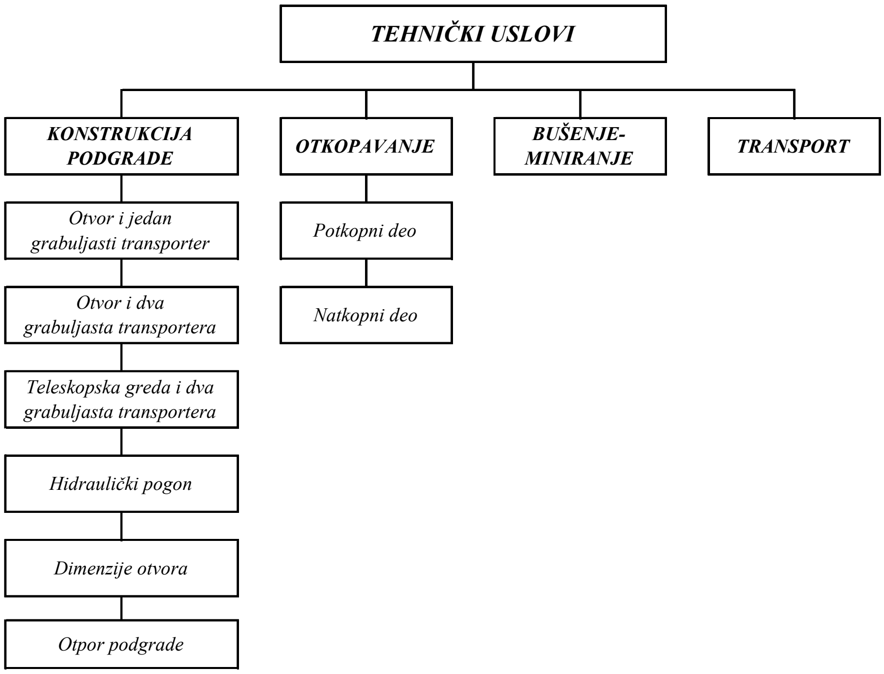
Ovakav model otkopa bitno se razlikuje od klasičnog modela širokog čela, dok su razlike u tehnologiji rada praktično neznatne.



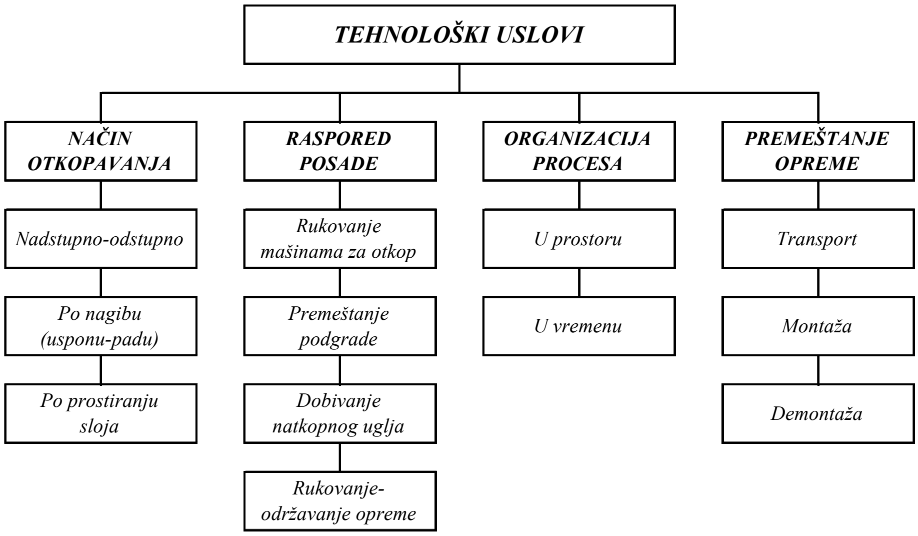
Sl. 1. *Sistem uslova kod izbora metode otkopavanja kratkim mehanizovanim čelom sa VK*



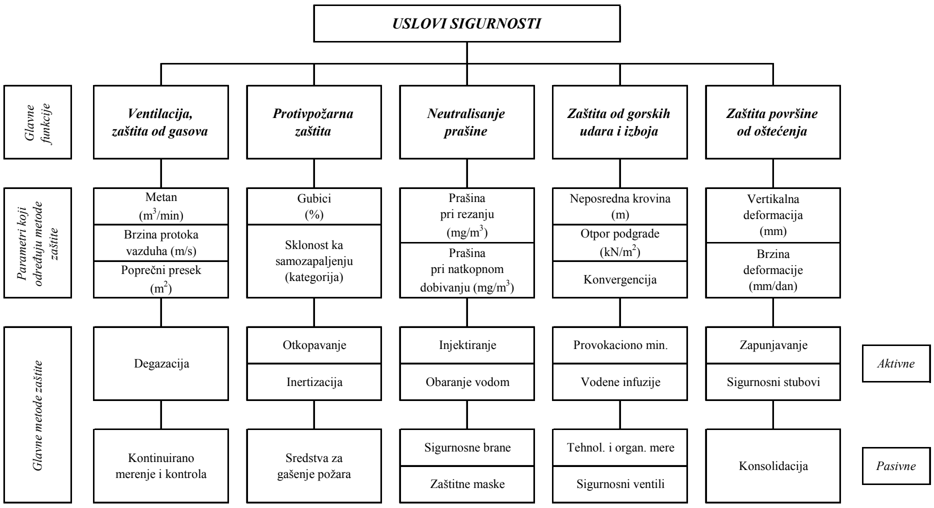
Sl. 2. *Prirodno-geološki uslovi kod izbora metode otkopavanja kratkim mehanizovanim čelom*



Sl. 3. *Tehnički uslovi kod izbora otkopavanja kratkim mehanizovanim čelom sa VK*



Sl. 4. *Tehnološki (organizacioni) uslovi kod izbora metode otkopavanja kratkim mehanizovanim čelom sa VK*



Sl. 5. *Uslovi sigurnosti kod izbora metode otkopavanja kratkim mehanizovanim čelom sa VK*

Model kratkog mehanizovanog čela pogodan je zbog svoje dužine za ležišta nepravilnog oblika. Prednosti primene ove metode i tehnologije otkopavanja sadržane su u sledećem:

- otkopavanju delova ležišta uglja nepravilnog oblika i kraćih dimenzija otkopnih polja;
- u zavisnosti od prirodno-geoloških uslova prilagodiva je brzina napredovanja čime se postiže odgovarajuća proizvodnja;
- malom učešću radne snage i većim učincima u odnosu na stubne otkope;
- većoj fleksibilnosti promenljivim radnim uslovimai lakšim intervencijama u otežanim uslovima otkopavanja.

Kompleksnu mehanizaciju na kratkom mehanizovanom čelu sa natkopnim dobijanjem sačinjava:

- samohodna hidraulična podgrada;
- mašina za otkopavanje;
- hidraulično postrojenje,
- sistem grabuljastih transportera;
- energetsko postrojenje.

Ovi delovi kompleksa međusobno se povezuju mehanički i funkcionalno u cilju sinhronizacije radnih operacija. Samo otkopavanje uglja vrši se dobijanjem iz potkopnog i natkopnog dela, potkopano rezanjem i natkopno „točenje“.

3. SISTEM USLOVA KOD IZBORA METODE OTKOPAVANJA KRATKIM MEHANIZOVANIM ČELOM SA VERTIKALNOM KONCENTRACIJOM (VK)

Uvažavajući konkretne prirodno-geološke uslove u nekom ležištu potrebno je precizirati sistem uslova kako bi se na osnovu toga mogao odabrati adekvatan sistem otkopavanja.

Na slikama 1-5 dat je istraživanjima utvrđen prikaz sistema uslova kod izbora metode otkopavanja kratkim mehanizovanim čelom sa vertikalnom koncentracijom.

ZAKLJUČAK

Na osnovu sagledavanja primenjenih sistema otkopavanja u konkretnim prirodno-geološkim uslovima aktivnih ležišta uglja u Srbiji, očigledno je da se u cilju ekonomičnije i sigurnije eksploatacije mora vršiti modernizovanje i osavremenjavanje sistema otkopavanja.

Rezultatima izvršenih istraživanja utvrđeno je da se racionalizacija sistema otkopavanja treba vršiti u većem obimu uvođenjem kratkih mehanizovanih čela, koncentracije pri otkopavanju debelih ugljenih slojeva.

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DEFINITION OF SYSTEM THE INFLUENTIAL CONDITIONS IN THE SELECTION OF MINING METHOD WITH THE SHORT MECHANIZED FACE IN THE COAL MINES

Abstract

The paper outlines the system of influential conditions in the selection of mining method with the short mechanized face with vertical concentration in the coal mines in Serbia.

This method is flexible and adjustable to the complex mining conditions, such as in the most now active deposits. The aim of realized investigations within this paper is to give a new approach to the subject mining method its wider use.

Key words: *coal, underground mining, short mechanized face*

1. INTRODUCTION

Successfully selected a set of technological operations in the process of mining for a particular work environment and specific natural and geological conditions, which is defined as the mining method providing a high degree, concentration of technique and technology of production work has to achieve:

- as high as possible shift or daily capacity of excavation unit;
- as high as possible productivity per employee at a stope;
- fewer losses of coal;
- as high as possible safety of all technological operations for coal obtaining;
- a satisfactory efficiency.

The main directions of development the system of coal deposit excavation by the underground method is aimed at improving methods of wide face with a complex machinery, with different lengths of stope and heights of excavation.

Taking into consideration the specific natural and geological conditions in the active deposits of brown coal in Serbia, it was came to the knowledge on possibility of introduction the short mechanized faces in some parts of the deposit, and in this paper, the system of influential conditions is determined in the selection of the above method, which has significant advantages over the currently used classical pillar methods.

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2. DESCIPTION OF METHOD OF SHORT MECHANIZED FACES

Short mechanized face as opposed to the wide face is dimensioned with the stope length of 30-50 m, and usually with one transport-ventilation room attached to other facilities in the excavation area.

This model of stope differs substantially on classical model of wide face, until the differences of operation technology are practically negligible.

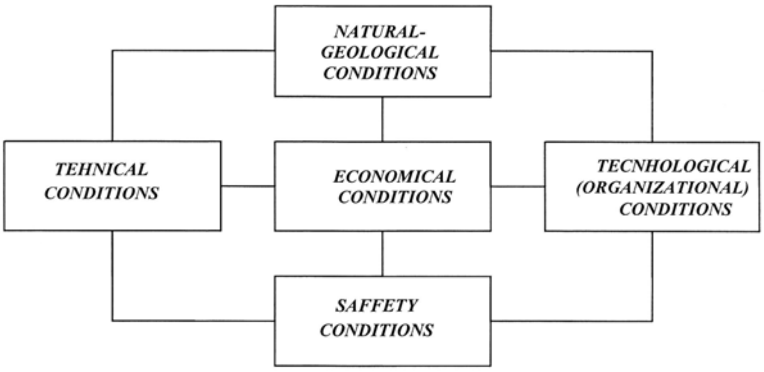

Fig. 1. *System of conditions for selection the mining method with short mechanized face with vertical concentration*

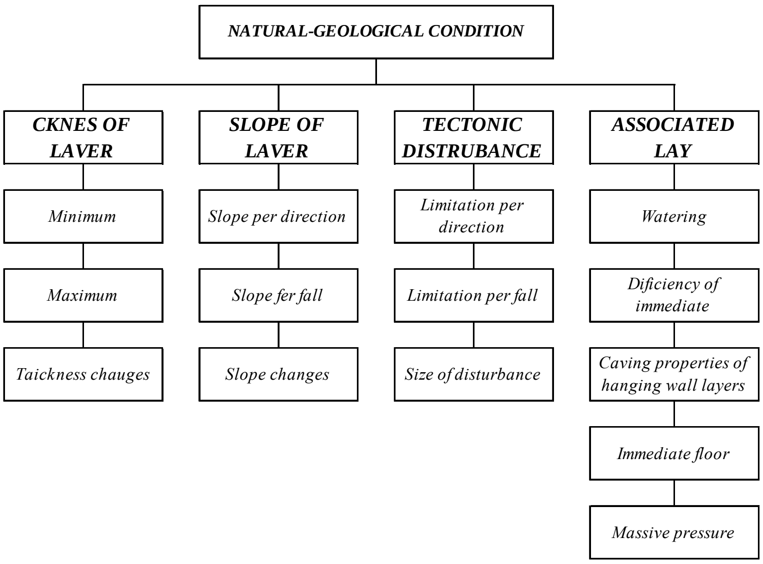

Fig. 2. *Natural geological conditions in the selection of mining method with mechanized short face*

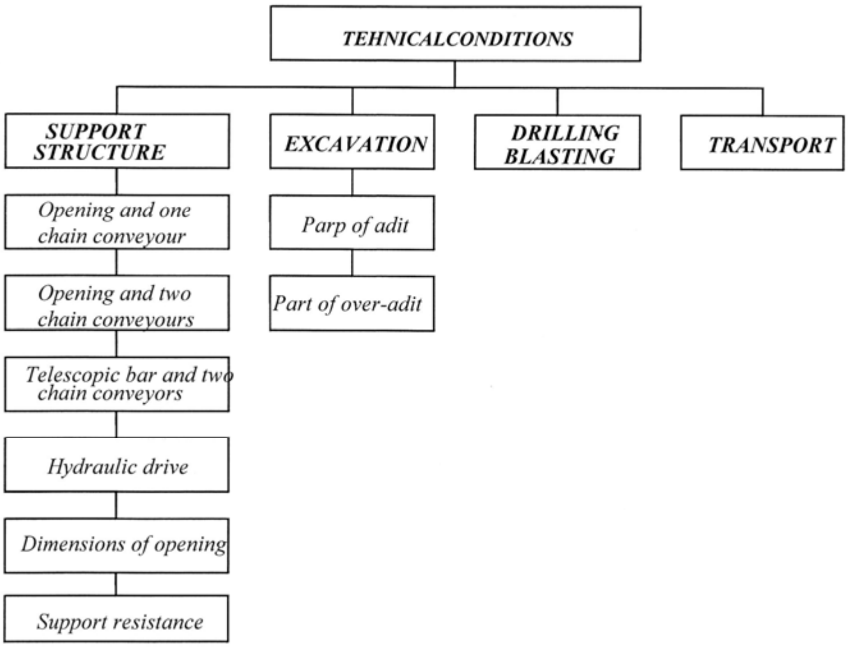

Fig. 3. *Technical conditions in the selection of mining method with mechanized short face with high concentration*

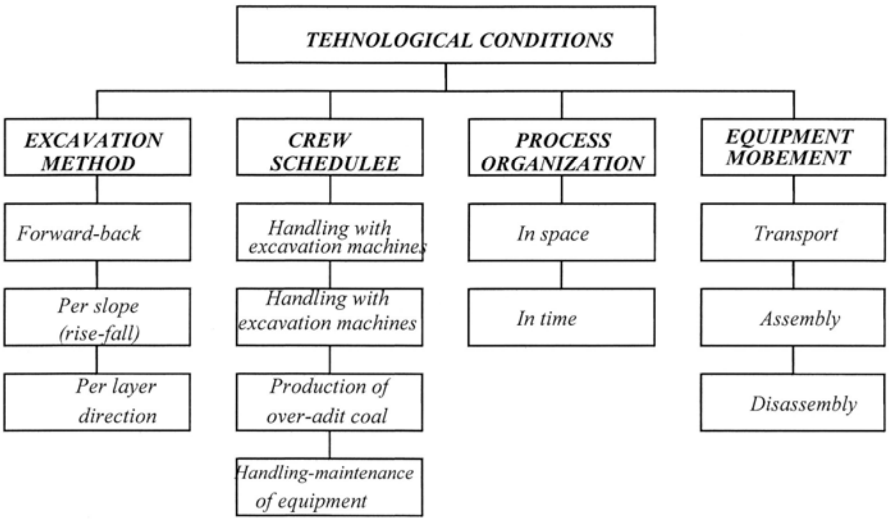

Fig. 4. *Technological (organizational) conditions in the selection of mining method with mechanized short face with high concentration*

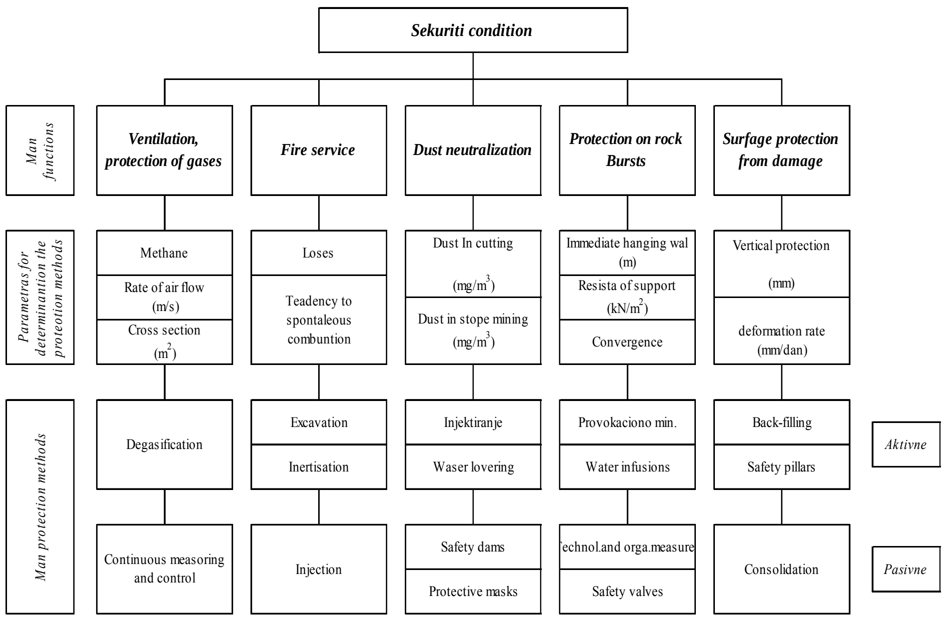

Fig. 5. *Safety conditions in the selection of mining method with short mechanized face with high concentration*

Model of short mechanized face is suitable due to its length for irregular shaped deposits. Advantages of this mining method and technology are contained in the following:

- mining of the coal deposit parts of irregular shape and shorter dimensions of mining fields;
- depending on the natural and geological conditions, the progress rate is adjustable allowing a proper production;
- low participation of labor force and higher effects compared to the pillar stopes;
- higher flexibility to changing work conditions and easier interventions in difficult mining conditions.

Complex mechanized equipment on a short mechanized face with the over-adit production consists of:

- self-propelled hydraulic support;
- excavating machine;
- hydraulic plant;
- system of chain conveyors;
- power plant.

These parts of the complex are connected to each other mechanically and functionally in order to synchronize working operations. The coal excavation is carried out by getting from adit and over-adit part, the adit cutting and over-adit "refilling".

3. SYSTEM OF CONDITIONS IN SELECTION THE MINING METHOD WITH THE SHORT MECHANIZED FACE WITH VERTICAL CONCENTRATION (VC)

Taking into account the specific natural and geological conditions in some deposit, it is necessary to specify the system of conditions as an adequate system of mining could be selected based on this.

Figures 1-5 give a review of system of conditions, determined by investigations, in a selection of mining method with the mechanized face with vertical concentration.

CONCLUSION

Based on the analysis of the applied mining systems in specific natural-geological conditions of active coal deposits in Serbia, it is obvious that in order to more economical and safer mining, the modernization and upgrading of mining system have to be carried out.

The results of realized investigations have determined that the rationalization of mining system should be done by introduction of the short mechanized faces, the concentration in excavation of thick coal layers.

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FORMIRANJE ISTOČNOG ODLAGALIŠTA JALOVINE U BORU I PREGLED DOSADAŠNJIH AKTIVNOSTI NA ZAŠTITI ŽIVOTNE SREDINE**

Izvod

Raskrivka u periodu od 1975 do 1980 godine sa postojećeg površinskog kopa Bor i proširenja do konačnih granica, odlagana je na više lokacija u blizini površinskog kopa. U ovom radu dato je formiranje Istočnog odlagališta – visokog planira odlaganjem 29,614.373 m^3 čvrste mase i pregled dosadašnjih aktivnosti na zaštiti životne sredine. Formiranje Istočnog odlagališta uticalo je na promenu ruže vetrova u Boru, ali istovremeno i na zaštiti životne sredine sela Oštrelj od uticaja gasova iz Topionice Bor.

Ključne reči: *odlagalište jalovine, zaštita životne sredine*

1. UVOD

Raskrivka (jalovina) sa površinskog kopa Bor odlagala se na više lokacija pri čemu su formirana spoljašnja odlagališta: Zmajevo, Severozapadno, Severno, Severoistočno, Istočno odlagalište i unutrašnje odlagalište unutar kopa. Odlaganjem jalovine na spoljašnjim odlagalištima formirane su ravne i kose degradiranane površine:

- Zmajevo i Severozapadno odlagalište sa 12 ha ravne površine i 26 ha kose površine,
- Severno odlagalište sa 38 ha ravne i 27 ha kose površine,
- Severoistočno odlagalište sa 47 ha ravne i 58 ha kose površine,
- Istočno odlagalište sa 27 ha ravne i 40 ha kose površine

Istočno odlagalište jalovine, koje se naziva još Oštreljsko ili Cijanizacija nalazi se na krajnjem istoku od površinskog kopa Bor (slika 1) pored bivšeg pogona Cijanizacije, koje nije više u funkciji jer je deo kosine odlagališta za luženje kliznulo 80 godina prošlog veka i onesposobilo ovo postrojenje. To je ujedno najviše odlagalište, čija završna ravan je na K+475 m nv, a nožica na K+375 m nv. Visina formiranog odlagališta iznosi 100 m sa nagibom visoke kosine 38^0. U podnožju ovog odlagališta na jugoistočnoj strani nalazi se jezero Robule (slika 2.).

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** *Rad je proizašao iz projekta broj 37001 "Uticaj rudarskog otpada iz RTB-a Bor na zagađenje vodotokova sa predlogom mera i postupaka za smanjenje štetnog dejistva na životnu sredinu", koji je finansiran sredstvima Ministarstva za prosvetu i nauku Republike Srbije*

Sl. 1. *Istočno odlagalište jalovine u Boru*

Sl. 2. *Jezero Robule*

2. FORMIRANJE ISTOČNOG ODLAGALIŠTA

U cilju smanjenja dužine transporta koje je uticalo na poskupljenje transportnih troškova i ukupnu cenu dobijanja bakra iz površinskog kopa Bor, raskrivka (jalovina) se odlagala u neposrednoj blizini kopa i na udaljenosti 100 m od gornje ivice kopa.

Formiranja Istočnog odlagališta PK Bor počelo je sa nasipavanjem jalovine sa nivoa 460 m nv i po usponu od 5% prema istoku. Prostor za odlaganje jalovine iznosio je 22,000.000 m^2 sa mogućnošću proširenja odlagališta dalje prema istoku i prema jugoistoku iznad jalovišta za luženje, gde je odlagana siromašna ruda sa 0,1 – 0,3% Cu, koja je lužena i hidrometalurškim putem u Cijanizaciji dobijen je bakar.

Prema dinamici formiranja, Istočno odlagalište bilo je aktivno u:

- 1975 god. sa količinom jalovine 5,923.183 m^3 čv. mase
- 1976 god. Sa količinom jalovine 5,367.950 m^3 čv. mase
- 1977 god. Sa količinom jalovine 7,380 003 m^3 čv. mase
- 1978 god. Sa količinom jalovine 5,339.064 m^3 čv. mase
- 1979. god. Sa količinom jalovine 5,604.173 m^3 čv. mase
- Ukupno odložene čvrste mase 29,614.373_m^3 čv. mase

3. PETROGRAFSKE KARAKTERISTIKE ODLOŽENE JALOVINE

U zahvatu proširenja postojećeg površinskog kopa Bor bile su zastupljene sledeće stenske mase, koje su odlagane i na Istočno odlagalište:

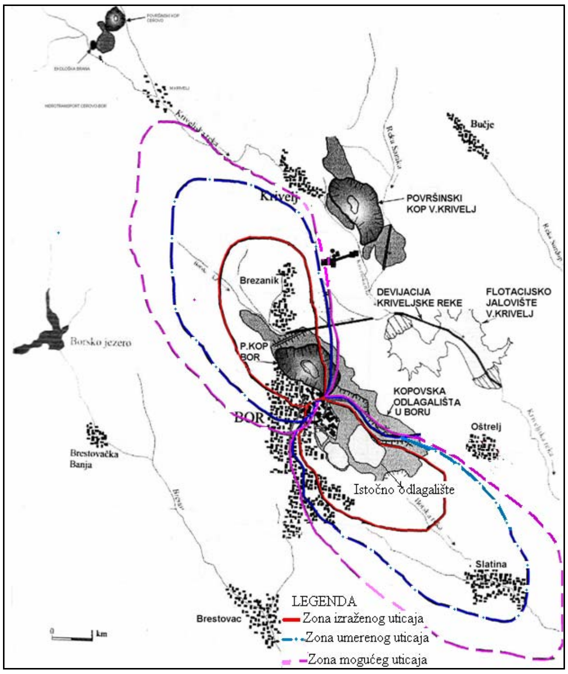
- Piroklastične stene, γ=2,61 t/m^3
- Silifikovani andeziti, γ=2,82 t/m^3
- Hloritisani andeziti, i γ=2,62 t/m^3 i,
- Kaolinisani andeziti γ=2,82 t/m^3

4. OSNOVNE PEDOLOŠKE KARAKTERISTIKE POVRŠINA ISTOČNOG ODLAGALIŠTA

Odlagani jalovinski materijal na Istočnom odlagalištu bio je pomešan sa smeđim kiselim zemljištem. Vremenom jalovina se na odlagalištu stabilizovala pri čemu je pokrenuta i pedogeneza obrazujući smeđe kiselo zemljište oštećeno sumpordioksidom. Mestimično se javlja i smeđe lesivirano zemljište nastalo na andezitu dosta oštećeno delovanjem sumpordioksidom. Hemijske osobine jalovine nisu povoljne jer je izražena jaka kiselost pH=3 i nizak stepen zasićenosti bazama.

5. UTICAJ ISTOČNOG ODLAGALIŠTA NA ŽIVOTNU SREDINU

Položaj Istočnog odlagališta jalovine utiče da se selo Oštrelj štiti od zagađenja koja se emituju preko topioničkog dimnjaka (slika 3). Površine Istočnog odlagališta jalovine leti zbog ekspozicije prema jugu su izložene visokoj temperaturi i insolaciji pa brzo tlo izgubi vlagu i dolazi do sušenja u površinskom sloju. Dominantni vetrovi iz severozapadnog pravca podižu raspadnuti površinski sloj peska i prašine i u vidu oblaka (slika 3) raznosi zagađenje na zemljište u pravcu sela Slatine. Obzirom na veliki nagib kosina 38^0, bujice koje se stvaraju posle kiša i topljenja snega spiraju rastresiti materijal formirajući skeletno tlo na velikim površinama (slika 4). Proces intezivnog spiranja kosine dovelo je do stvaranja brojnih recentnih geomorfoloških oblika kao što su jaruge, rovine, urvine i dr.



Sl. 3. *Zone uticaja zagađenja iz dimnjaka Topionice u Boru*

Klima na području Bora je kontinentalna sa srednjom godišnjom količinom padavina u Boru i okolini 583,0 mm i srednjom godišnjom vrednosti temperature 10,9 ^{0}C. U Boru i okolini u toku godine vetrovito je 38,8% ili 141,62 dana i 61,2% su tišine (C) ili 223,38 dana u godini bez vetra. Borski region spada u neprovetrena područja. Na mikroklimu u Boru veliki uticaj imaju i formirana odlagališta jalovine, a naročito Istočno odlagalište čija je visina 100 m i dosta utiče na neprovetrenost u Boru. Iz tabele 1 se mogu videti brzine i čestine vetrova iz pojedinih pravaca posmatranih za 10 godina.

Tabela 1. *Podaci o vetrovima na području Bora i okoline za period 1998-2008.*

Pravac	N	NNE	NE	ENE	E	ESE	SE	SSE	S	SSW	SW	WSW	W	WNW	NW	NNW	C
Č %	1.0	0.4	0.4	4.6	4.2	0.7	0,.3	0.2	2.7	0.6	0.3	0.7	4.8	6.8	9.8	1.3	61.2
V m/s	1.5	1.7	1.8	1.6	1.6	0.8	0.9	1.0	1.5	1.2	1.1	1.2	1.8	2.3	2.4	1.8	-

Sl. 4. *Jaruge na kosini Istočnog odlagališta*

Sl. 5. *Završna ravan Istočnog odlagališta*

6. DOMET PRAŠINE SA RAVNIH POVRŠINA ISTOČNOG ODLAGALIŠTA

Dominantni vetar iz severozapadnog pravca duva upravno na završnu ravan Istočnog odlagališta, čija površina je 270.000 m^2. Za prognozu dometa koncentracije prašine iznad MDK u životnoj sredini Slatine korišćena je formula:

$$X = \frac{k \cdot E}{\psi \cdot L_g W_s (C_{MDK} - C_0)}, m$$

gde su:

- X - rastojanje (domet) od linijskog izvora štetnosti na osi smera duvanja vetra, m
- k - eksperimentalni koeficijent koji zavisi od položaja izvora,
- E- emisija prašine sa suvih površina deponije, mg/s
- ψ - bezdimenzioni koeficijent koji karakteriše turbulentnost toka vazdušne struje
- L_g - dužina linijskog izvora
- W_s - srednja brzina vetra
- C_o - prirodni fon koncentracije prašine,
- C_{MDK} - maksimalna dozvoljena koncentracija prašine.

Emisija prašine sa otvorenih površina na pravcu vetra F (m^2) određuje se na osnovu specifičnog podizanja prašine po formuli:

$$E = g \times F,\ mg/s$$

gde su:

- g - vrednost specifičnih podizanja prašine za različite brzine vetra preko otvorenih površina,
- F - otvorena površina preko koje duva vetar brzinom W

$$E = 108.000\ mg/s$$

Domet čestica prašine sa završne ravni Istočnog odlagališta prema selu Slatina u pravcu severozapadnog vetra iznosi: X=4613,80 m. Taloženje čestica iz vazduha na zemljište u zavisnosti od veličine čestica vrši se na prostoru perjanice čija osa je dužine 4613,80 m. Prema stepenu disperznosti razlikuje se: prašina sa česticama većim od 10 milimikrona (taloži se u blizini izvora emisije povećanom brzinom kad vazduh miruje), prašina sa česticama od 10 do 0,1 milimikrona (taloži se na udaljenosti od uzvora emisije konstantnom brzinom kad vazduh miruje), prašina sa česticama ispod 0,1 milimikrona (ne taloži se). Prašina sa česticama većim od 10 milimikrona utiče direktno na zdravlje ljudi i to na bolest kože i bolest očiju. Prašina ispod 10 milimikrona izaziva pneumokoniozna oboljenja. Prašina sadrži sulfide, minerale bakra i ostale teške metale i kada dospe u zemljište podstiče proces zakišeljavanja zemljišta pri čemu se pojačava pokretljivost jona teških metala u zemljištu i sposobnost njihovog akumuliranja u biljkama. Prema literaturnim podacima, ako je pH<6,5 u zemljištu se jako povećava pokretljivost kadmijuma, ako je pH<5,5 povećava se pokretljivost i kod Ni, Mn, Zn, Co, Al, a sa pH<4 i kod Pb i Cu.

7. DOSADAŠNJE AKTIVNOSTI NA ZAŠTITI ŽIVOTNE SREDINE

U cilju zaštite životne sredine od podizanja prašine sa suvih površina odlagališta raskrivke u Boru vršena su istraživanja na rekultivaciji površina (slike 6. i 7).

Sl. 6. *Rekultivacija odlagališta jalovine u Boru*

Sl. 7. *Pokušaj rekult. na Istoč. odlag.*

8. REZULTATI IZVEDENIH RADOVA

U jesen 1979 godine zasađeno je 5.000 sadnica bagrema i izvršena setva bagrema u brazdice direktno na supstratu. Juna 1980 godine ustanovljeno je da je oko 40% bagremovog semena izniklo, a oko 30% sadnica je preživelo.

U 1981. godini se pratio tok razvoja zasađenih sadnica i ustanovljeno je da je od 40% primljenih ostalo da se razvija samo 10% (slika 5. i 7.) i to isključivo na onim delovima odlagališta gde je bilo samonikle trave, ali mestimično tamo gde se vlaga više zadržava (zaklonjena i senovita mesta). Isti slučaj je i sa isklijalim sadnicama iz semena.

Poučeni iskustvom iz prvog pokušaja, sadnja je izvršena u proleće 1982 godine i tom prilikom je zasađeno 120.000 sadnica bagrema sa dodatkom humusa. Procenat prijema zasađenih sadnica iznosio je 60%.

1983 godine zasađeno je 80.000 sadnica bagrema, 1984 godine 96.000 sadnica bagrema, 1985 godine 198.000 sadnica bagrema i 1986 godine 100.000 sadnica bagrema. Procenat prijema iznosio je 70%. Na razvoj sadnica uticala je suša u letnjem periodu i nepovoljna podloga.

ZAKLJUČAK

U cilju sprečavanja daljeg zagađivanja okoline prašinom sa Istočnog odlagališta potrebno je nastaviti sa rekultivacijom i primeniti iskustva u sađenju bagrema sa odlagališta u Velikom Krivelju i Cerovu, gde je prijem 95%. Vode jezera Robule treba tretirati prema programima koji postoje u Institutu za rudarstvo i metalurgiju. Rekultivacijom ravnih i kosih površina Istočnog odlagališta doprineće zaštiti životne sredine i boljoj mikroklimi Bora.i sela Slatine.

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Ružica Lekovski[*], *Radmilo Rajković*[*], *Ljubiša Obradović*[*]

FORMATION OF THE EAST WASTE DUMP IN BOR AND REVIEW OF PREVIOUS ACTIVITIES ON THE ENVIRONMENTAL PROTECTION[**]

Abstract

Overburden, in the period from 1975 to 1980 from the existing open pit Bor and expansion to the final borders, was delayed at several locations near the open pit. This paper gives the formation of the East Waste Dump - high landfill of 29,614.373 m^3 solid mass and review of the previous activities on the environmental protection. Formation of the East Waste Dump has resulted into a change in a wind rose in Bor, but at the same time to the environmental protection of the village Oštrelj on gases from the Smelter Bor.

Key words: *tailing dump, environmental protection*

1. INTRODUCTION

Overburden (waste) from the open pit Bor was delayed at several locations where the outside waste dumps were formed: Zmajevo, Northwest, North, Northeast, east and Inner waste dump within the open pit. Disposal of waste on outside waste dump has formed the flat and slope degraded areas:

- Zmajevo and Northwest waste dump with 12 ha of flat area and 26 ha of slope area,
- North waste dump with 38 ha of flat and 27 ha of slope area,
- Northeast waste dump with 47 ha of flat and 58 ha of slope area,
- East waste dump with 27 ha of flat and 40 ha of slope area.

East waste dump which is also called the Oštrelj waste dump or Cijanizacija is located on the far east from the open pit Bor (Figure 1) near the former plant Cijanizacija, which is no longer in a function because a part of dump slope for leaching slipped in the 80s of the last century and knocked this plant. This is also the highest waste dump with the final plane at K +475 m above sea level, and the foot at K +375 m above sea level. Height of formed waste dump is 100 m with a gradient of high slope high slope with a gradient 38^o. On the foot of this waste dump, the Robule Lake is located on the southeast side (Figure 2).

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Fig. 1. *East Waste Dump in Bor*

Fig. 2. *Robule Lake*

2. FORMATION OF THE EAST WASTE DUMP

In order to reduce transport distances, which resulted in the price increase of transport costs and total cost of copper obtaining from the open pit Bor, the overburden (waste) was delayed in the immediate vicinity of the open pit and at distance of 100 m from the top of pit.

Formation of the East waste dump of the open pit Bor began with filling the waste from the level of 460 m above sea level and the rise of 5% to the east. The area for waste dumping amounted to 22,000.000 m^2 with the possibility of expanding the waste dump further on the east and southeast over the tailing dump for leaching, where the low-grade ore with 0.1 - 0.3% Cu was delayed, which was leached and copper was obtained by hydrometallurgical method in cianization.

According to the dynamics of formation, the East waste dump was active in:

- 1975 with the waste amount 5,923.183 m^3 solid mass
- 1976 with the waste amount 5,367.950 m^3 solid mass
- 1977 with the waste amount 380 003 m^3 solid mass
- 1978 with the waste amount 5,339.064 m^3 solid mass
- 1979 with the waste amount 5,604.173 m^3 solid mass
- Total delayed solid mass 29,614.373 m^3 solid mass

3. PETROGRAPHIC CHARACTERISTICS OF DEPOSITED WASTE

The following rock masses, also disposed on the East waste dump, were present in the expansion of the existing open pit Bor:

- pyroclastic rocks, γ=2.61 t/m^3,
- silicified andesites, γ=2.82 t/m^3,
- chloritised andesites, γ=2.62 t/m^3, and
- kaolinited andesites, γ=2.82 t/m^3.

4. BASIC PEDOLOGICAL CHARACTERISTICS OF THE SURFACES ON THE EAST WASTE DUMP

Delayed waste material on the East waste dump was mixed with the brown acid soil. By the time, the waste on the dump was stabilized where the pedogenesis started forming the brown acid soil damaged by sulfur dioxide. Brown loosed soil, formed on andesite, occurs sporadically and it is damaged a lot by action of sulfur dioxide. Chemical properties of waste are not favorable because strong acidity pH = 3 and low degree of saturation with bases is expressed.

5. THE EFFECT OF THE EAST WASTE DUMP ON ENVIRONMENT

Position of the East waste dump has effects of protection against pollution the village Oštrelj that is emitted through the Smelter stack (Figure 3). Surfaces of the East waste dump, due to the exposition to the south, are exposed to high temperature and insolation in summer, and the soil loses moisture quickly and the surface layer is dried. Prevailing winds from the northwest direction raise the decomposed surface layer of sand and dust in the form of a cloud (Figure 3), deliver the pollution on land towards the village of Slatina. Regarding to a high incline of slopes 38^0, the stream that is generated after the rains and melting of snow wash the loose material forming a skeletal soil over large areas (Figure 4). The process of intensive wash-out of slope led to creation of a number of recent geomorphological features such as gullies, grabens, cliffs, etc.

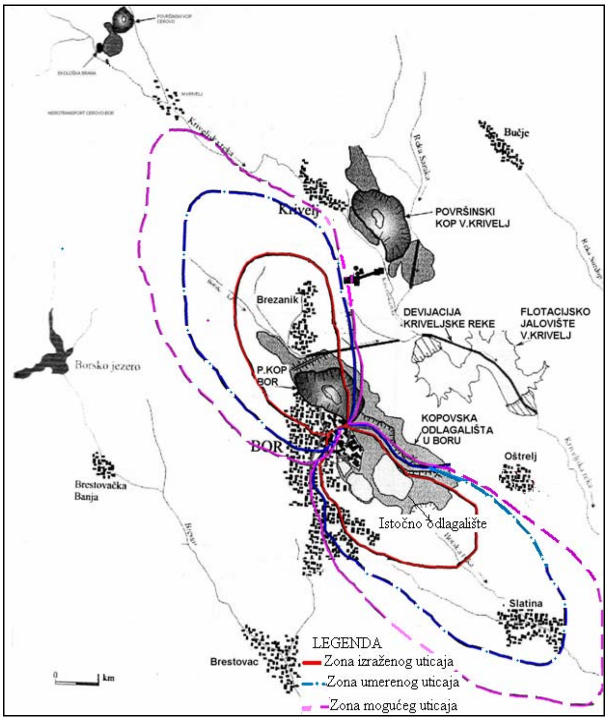

Fig. 3. *Zone of pollution influence from the Smelter stack in Bor*

Climate in the area of Bor is continental with the average annual precipitation in Bor and its surroundings 583.0 mm and average annual temperature of 10.9°C. In Bor and its surroundings during a year is windy 38.8% or 141.62 days, and 61.2% are silence (C), or 223.38 days a year with no wind. Bor region belongs to the non ventilated area. The microclimate in Bor is under a high impact of formed waste dumps and in particular the East wqaste dump at an altitude of 100 m, and non-ventilation also has a lot of influence in Bor. Table 1 gives the speeds and frequencies of winds from different directions observed for 10 years.

Table 1. *Data on winds in the area of Bor and surroundings for the period 1998-2008*

Direction	N	NNE	NE	ENE	E	ESE	SE	SSE	S	SSW	SW	WSW	W	WNW	NW	NNW	C
S %	1.0	0.4	0.4	4.6	4.2	0.7	0,.3	0.2	2.7	0.6	0.3	0.7	4.8	6.8	9.8	1.3	61.2
V m/s	1.5	1.7	1.8	1.6	1.6	0.8	0.9	1.0	1.5	1.2	1.1	1.2	1.8	2.3	2.4	1.8	-

Fig. 4. *Ravines on the slope of the East waste dump*

Fig. 5. *Final plane of the East waste dump*

6. THE RANGE OF DUST FROM FLAT SURFACES OF THE EAST WASTE DUMP

The dominant wind blows from the northwest direction perpendicular to the plane of the East waste dump, whose surface is 270,000 m^2. For the forecast of range of dust concentrations above the MDK in the environment of Slatina, the formula was used:

$$X = \frac{k \cdot E}{\psi \cdot L_g W_s (C - C_0)}, m$$

Where:

- X - distance (range) from line source of hazard on the axis of wind blowing direction, m
- K - experimental coefficient which depends on the position of source,
- E- dust emission from dry areas of waste dump, mg/s
- dimensionless coefficient that characterizes the flow of air flow turbulence
- W_i – wind speed of certain direction,
- L – length of line source,
- W_s – average wind speed
- C_o – natural font of dust concentration,
- C_{MDK} – maximum acceptable dust concentration

Dust emission from open areas on the wind direction F (m^2) is determined by the specific dust raising by the formula:

E=g x F, mg/s

Where:

- g – value of specific dust raising for different wind speeds over open areas,
- F – open area over which wind blows in speed W

E=108000 mg/s

The range of dust particles from the final plane of the East waste dump towards the village of Slatina in the northwest wind direction is: X = 4613.80 m. Deposition of particles from the air on the land depending on the particle size is done in the area of a plume, whose axis is the length of 4613.80 m. According to the dispersion degree, there are: dust with particles larger than 10 milli microns (deposited near the emission source by the increased speed when the air is at rest), dust with particles of 10 to 0.1 milli microns (deposited at a distance from the emission source at a constant speed when the air is at rest), and dust with particles below 0.1 milli microns (not deposited). Dust with particles larger than 10 milli microns directly impact human health and the skin and eye disease. Dust under 10 milli microns causes pneumoconious diseases. Dust contains sulfides, copper minerals and other heavy metals and when it comes into soil, it encourages the process of soil acidification in which the mobility of heavy metal ions is increased in soil and the ability of their accumulation in plants. According to literature data, if pH<6.5 in the soil, the mobility also increases of Ni, Mn, Zn, Co, Al, and with pH<4 also of Pb and Cu.

7. PREVIOUS ACTIVITIES OF ENVIRONMENTAL PROTECTION

In order to protect the environment on dust raising from dry surfaces of the waste dump in Bor, the investigations were carried out on reclamation of the areas (Figures 6 and 7).

Fig. 6. *Reclamation of the waste dump in Bor*

Fig. 7. *An attempt of reclamation on the East waste dump*

8. THE RESULTS OF REALIZED WORKS

In the autumn in 1979, 5000 seedlings of acacia were planted and acacia was sowed in furrows directly on the substrate. In June 1980, it was found that about 40% acacia seeds grew, and about 30% of seedlings survived.

In 1981, the course of development of planted seedlings was followed and it was found that 40% was planted and only 10% (Figures 5 and 7) has to be grown and only in those parts of the landfill where there were wild grasses, but in some places where moisture is more reserved (sheltered and shady places). The same case is with germinated seedlings from seed.

Drawing on experience from the first attempt, planting was done in the spring of 1982 and on that occasion. 120,000 seedlings of acacia were planted with the addition of humus. The percentage of receiving the planted seedlings was 60%.

In 1983, 80,000 seedlings of acacia were planted; in 1984, 96,000 seedlings of acacia; in 1985, 198,000 seedlings of acacia and in 1986, 100,000 seedlings of acacia. The percentage of receiving was 70%. The development of seedlings was influenced by drought in summer, and unfavorable substrate.

CONCLUSION

In order to prevent further environmental pollution with dust from the East waste dump, it is necessary to continue with the remediation and apply experiences in planting acacia from the waste dump in Veliki Krivelj and Cerovo, where a reception was 95%. The water of the Robule lake should be treated according to the programs that exist in the Mining and Metallurgy Institute. Remediation of flat and sloped surfaces of the East waste dump will contribute to the environmental protection and better microclimate of Bor and the village of Slatina.

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ANALIZA OSTVARENIH GODIŠNJIH KAPACITETA OTKOPAVANJA I UTICAJNIH FAKTORA NA POVRŠINSKOM KOPU VELIKI KRIVELJ KOJI POSLUJE U OKVIRU RTB BOR-GRUPA, SRBIJA***

Izvod

Ovim radom dat je pregled i izvršena analiza ostvarenih godišnjih kapaciteta otkopavanja na površinskom kopu Veliki Krivelj, koji posluje u okviru RTB BOR-Grupe u Srbiji. Analizom su obuhvaćeni i glavni uticajni faktori.

Period koji se analizira počinje 1979. godine, kada su otpočeli radovi na otkopavanju otkrivke, a završava se 2010. godine.

Analiza je pokazala da su ostvarivani manji godišnji kapaciteti iskopina od projektovanih, ali prvenstveno na otkopavanju otkrivke s obzirom da je kapacitet na rudi morao da bude konstantan zbog nesmetanog rada flotacije.

Analiza je takođe pokazala da je kapacitet otkopavanja u direktnoj zavisnosti od broja aktivnih kamiona, pri čemu je ta zavisnost izrazita u slučaju otkrivke.

***Ključne reči: p**ovršinski kop, Veliki Krivelj, kapacitet, otkopavanje*

1. UVOD

Površinski kop Veliki Krivelj, u daljem tekstu PK, na istoku Republike Srbije, posluje u okviru DOO Rudnici bakra Bor, u daljem tekstu RBB, odnosno u okviru Rudarsko-topioničarskog basena Bor-Grupe, u daljem tekstu RTB.

Proizvodnja koncentrata bakra iz rude sa PK čini oko 75% od ukupne proizvodnje RBB-a, dok je učešće rude sa PK oko 90% ukupno otkopane rude, sa trendom daljeg povećanja [1].

Glavnim rudarskim pojektom otkopavanja i prerade rude bakra u ležištu Veliki Krivelj za kapacitet od 8 x 10^6 tona vlažne rude godišnje [2] iz 1978. godine obrađeno je otvaranje PK i njegova dalja eksploatacija do 2000. godine, sa pomenutim kapacitetom od 8 miliona tona vlažne rude godišnje.

Otkrivanje ležišta, odnosno otkopavanje otkrivke započelo je 1979. godine, pri čemu su se paralelno odvijali i radovi na izgradnji

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kapaciteta za njenu preradu, što je podrazumevalo primarno, sekundarno i tercijalno drobljenje i prosejavanje, mlevenje i flotiranje.

Krajem 1982. godine počelo je i otkopavanje rude, kao i njena prerada do finalnog proizvoda, tj. koncentrata bakra.

U analiziranom periodu do 2010. godine projektovani godišnji kapaciteti oktkopavanja bili su ostvarivani sa različitim uspehom pri čemu je to zavisilo od mnogo faktora, a ovim radom su obuhvaćeni samo glavni.

Nakon sprovedene analize može da se zaključi da je PK uglavnom ostvarivao manje godišnje kapacitete otkopavanja rude i otkrivke u odnosu na projektovane, s tim da je u nekim periodima ta razlika bila značajna.

Međutim, važno je napomenuti da su faktori koji su uticali na to često bili ''van'' PK, odnosno nalazili su se u sferi globalnih trendova u politici i ekonomiji i strategiji države Srbije, koja je neminovno uslovljavala i strategiju razvoja RTB-a, samim tim i PK-a.

U toku 2009. i 2010. godine došlo je do određenih ulaganja sredstava u nabavku dela osnovne i pomoćne opreme, što je rezultiralo delimičnim povećanjem kapaciteta otkopavanja, koji je još uvek ispod potrebnog nivoa, s obzirom na višegodišnji zaostatak u raskrivanju ležišta. Novi razvojni planovi zahtevali su i izradu novog Dopunskog rudarskog projekta otkopavanja i prerade rude bakra u ležištu Veliki Krivelj za kapacitet 10,6 x 10^6 tona vlažne rude godišnje. Posebna pažnja usmerena je na otkopavanje otkrivke jer je kapacitet na otkrivci do sada uvek bilo manji od planiranog zbog čega se javljalo višegodišnje kašnjenje, koje ovim Projektom treba nadoknaditi.

2. PREGLED PROJEKTOVANIH I OSTVARENIH GODIŠNJIH KAPACITETA OTKOPAVANJA

Analiza ostvarenih godišnjih kapaciteta podrazumeva da se oni uporede sa planskim koji su definisani odgovarajućim projektima, pa se u tom smislu najpre daje pregled rudarskih projekata koji su izrađeni za potrebe PK:

1. Glavni rudarski pojekat otkopavanja i prerade rude bakra u ležištu Veliki Krivelj za kapacitet od 8 x 10^6 tona vlažne rude godišnje (izrađen 1978. godine). Po ovom Projektu otkopavanje je trebalo da se vrši do 2000. godine, a vršeno je do 1985. godine.
2. Dopunski rudarski pojekat otkopavanja rude u ležištu Veliki Krivelj za kapacitet od 10,3 x 10^6 tona vlažne rude godišnje (izrađen 1985. godine). Po ovom Projektu otkopavanje je vršeno do 1991. godine.
3. Dopunski rudarski projekat otkopavanja rude u ležištu Veliki Krivelj za kapacitet od 12,9 x 10^6 tona vlažne rude godišnje (izrađen 1991. godine). Po ovom Projektu otkopavanje je vršeno do 1995. godine.
4. Dopunski rudarski pojekat otkopavanja rude rudnika Veliki Krivelj za kapacitet od 10,6 miliona tona vlažne rude (izrađen 1995. godine). Po ovom Projektu otkopavanje je vršeno do juna 2006. godine.
5. Dopunski rudarski pojekat otkopavanja i prerade rude u ležištu Veliki Krivelj za kapacitet od 8,5 x 10^6 tona vlažne rude godišnje (izrađen 2006. godine). Po ovom Projektu otkopavanje se vrši i danas.

U posmatranom periodu rađeni su još neki projekti koji se nisu primenjivali, s obzirom na činjenicu da nisu dobili saglasnost resornog Ministarstva rudarstva. Zbog toga su u prethodnom pregledu prikazani samo oni projekti po kojima su izvođeni radovi, uz navođenje period operativne primene.

S obzorom na to da se obrađuje relativno dug period (1979-2010.) analizirani su samo godišnji kapaciteti iskopina, tj. rude i otkrivke, planirani i ostvareni. Zbog toga se pojavio problem u vezi planiranih kapaciteta u godinama u kojima je dolazilo do njihove promene. On je prevaziđen tako što je kao merodavan usvojen onaj planirani godišnji kapacitet koji je definisan novim projektima.

Kako je zapisano, u toku je izrada Dopunskog rudarskog pojekta otkopavanja i prerade rude bakra u ležištu Veliki Krivelj za kapacitet od 10,6 x 10^6 tona vlažne rude godišnje pri čemu se planira da on bude završen u toku 2011. godine i da u istoj godini počne i njegova primena.

Menadžment RBB-a doneo je odluku o povećanju kapaciteta otkopavanja kako bi se povećao obim proizvodenje koncentrata bakra što je preduslov za upošljavanje kapaciteta rekonstruisane topionice čija izgradnja treba da se završi do novembra 2013. godine. U skladu sa tom odlukom izvršena je i nabavka dela osnovne i pomoćne opreme za potrebe PK.

3. UTICAJNI FAKTORI NA KAPACITET OTKOPAVANJA

Bakar, kao krajnji proizvod RTB Bor-Grupe, predstavlja berzansku robu koja se prodaje na nekoliko svetskih ''pijaca'', koje su pod direktnim uticajem globalnih geopolitičkih odnosa i interesa krupnog kapitala. Zahvaljujući takvim odnosima berzanska cena bakra se menjala u neverovatnom opsegu od 1 319 $/t koja je važila 07. 11. 2001. godine do 10 350 $/t koja je važila 22. 08. 2010. godine. Najviša u odnosu na najnižu cenu je veća za 785%, što je podatak koji govori sam za sebe.

Na kretanje svetskih cena bakra prilike u našoj Državi, odnosno RTB-u nemaju značajan uticaj s obzirom da je proizvodnja relativno mala.

Za potrebe ovog rada analizirane su cene bakra koje su važile na berzi u Londonu (LME) [3] s obzirom na to da je prodaja borskog bakra ugovarana po tim cenama.

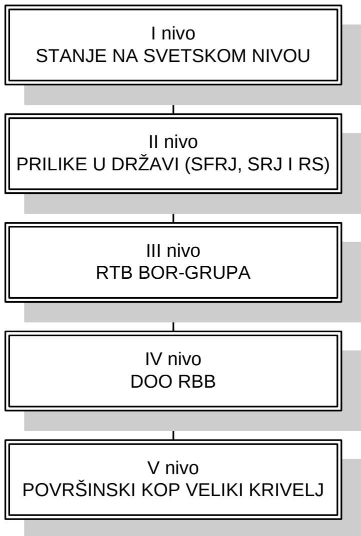
Zbog toga se stanje geopolitičkih odnosa na svetskom nivou, koje direktno utiče na cenu bakra, smatra prvim i najvišim nivoom uticajnih faktora, posredno i na prilike na PK.

Drugi nivo uticaja ostvaruje se u okviru Države koja donosi presudne odluke o strategiji svog razvoja, samim tim i o strategiji razvoja RTB-a jer je on njena značajna karika. Značaj se ogleda pre svega u tome što se radi o robi koja uvek može da se proda na stranom tržištu i da se tako ostvare stabilni prihodi.

Treći nivo uticaja ostvaruje se u okviru samog RTB-a koji je podeljena na zavisna preduzeća organizovana kao društva sa ograničenom odgovornošću.

Jedno od društava sa ograničenom odgovornošću je RBB, pa se ono smata četvrtim nivoom uticaja. Društvo ima više pogona pri čemu je pogon Jama Bor jedini koji sada proizvodi rudu bakra pored pogona PK.

I na kraju uticajni faktori koji deliju u okviru samog PK predstavljaju peti nivo, **slika 1**.



Sl. 1. *Nivoi uticajnih faktora na kapacitete otkopavanja*

Hijararhija uticajnih faktora, prikazana na **slici 1**, pokazuje da na kapacitete otkopavanja najmanje utiču prilike koje vladaju na samom PK, što je apsurdno.

Prvih deset godina proizvodnog rada PK (1982-1992.) realizovane su u relativno dobrim uslovima tako da je stečeno bogato iskustvo koje se koristi i danas. To iskustvo odigralo je odlučujuću ulogu u opstanku PK u periodima kada je njegov rad bio pod znakom pitanja.

Rukovodstvo PK uvek je povlačilo racionalne poteze u okviru skromnog ili nikakvog manevarskog prostora pa se tako proizvodnja odvijala kada je u jednoj smeni bilo samo nekoliko kamiona.

Uticajni faktori, na nivou samog PK, koji opredeljuju kapacitet otkopavanja u realnim uslovima podrazumevaju sledeće:

- inoviranje postojećeg kamionskog transporta otkrivke uvođenjem kombinovanog transporta,
- uhodanu organizaciju rada,
- racionalne planove otkopavanja,
- optimalno vođenje radova u realnim uslovima,
- dobro iskorišćenje raspoložive osnovne i pomoćne opreme,
- stručnu i iskusnu radnu snagu,
- solidno održavanje u otežanim uslovima,
- zadovoljavajuću radnu disciplinu i
- činjenicu da nije bilo velikh havarija koje su posledica više sile.

Imajući u vidu prethodna objašnjenja konstatuje se da su glavni uticajni faktori na kapacitet otkopavanja na PK ustvari bili u velikoj meri izvan njega.

Sa aspekta ostvarenih godišnjih kapaciteta otkopavanja u okviru samog PK najvažniji uticajni faktori bili su broj raspoložive osnovne i pomoćne mehanizacije i njeno iskorišćenje, kao i uvođenje kombinovanog sistema za transport otkrivke do zatvorenog površinskog kopa Bor, i njeno odlaganje u otkopani prostor tog kopa.

U tom smislu dalja analiza podrazumeva uticajne faktore koji su razvrstani na sledeći način:

- svetska geopolitička situacija i cena bakra,
- geopolitičko stanje u Državi,
- karakteristični periodi u poslovanju RTB-a
- karakteristični periodi u poslovanju RBB-a i
- stanje opreme na PK.

4. ANALIZA OSTVARENIH KAPACITET I UTICAJNIH FAKTORA

Uobičajeni uticajni faktori koji se odnose na najviši nivo, odnosno na globalne geopolitičke odnose i interes krupnog kapitala, ustvari se manifestuju preko jednog pokazatelja, odnosno preko svetske cene bakra.

Kao što se sa **dijagrama broj 1 i 2** vidi, cena bakra na LME gotovo da nije imala uticaja na proizvodnju rude na PK, izuzev u periodu 2000 – 2005. godine, kada je dostignut istorijski minimum cene bakra, posmatrano na konstantnu vrednost dolara iz 2005. godine [4]. Uticaj izrazito niske cene bakra ogledao se u nemogućnosti obnavljanja rudarske mehanizacije, usled smanjenog prihoda od prodaje bakra.

Međutim, u konkretnom slučaju globalni geopolitički odnosi i interes krupnog kapitala iskazali su se i u spefifičnom obliku koji se nije odnosio direktno na cenu bakra već na odnos prema državi u okviru koje je vršena eksploatacija na PK.

Podsećanja radi u analiziranom periodu eksploatacija na PK obavljana je u tri zvanične države, koje su nasleđivale jedna drugu, pri čemu je svaka naredna bila manja, i to:

- Socijalistička Federativna Republika Jugoslavija (SFRJ) do 1992. godine,

- Savezna Republika Jugoslavija (SRJ) 1992-2006. godine,

- Republika Srbija (RS) 2006. nadalje.

Promena iz SFRJ u SRJ podrazumevala je građanski rat koje u mnogome promenio svetski odnos prema tim državama, i to u negativnom smislu.

Zbog stanja u Državi, koje je ocenjeno kao loše, Ujedinjene nacije su uvele ekonomske sankcije koje su direktno uticale na rad PK s obzirom na to da je sva oprema uvozna. Formalno sankcije su trajale do 1992. do 1996. godine.

S druge strane, za vreme egzistiranja SRJ NATO je izvršeno bombardovanje u okviru koga su gađani i ciljevi u RTB-u (Trafo stanica 3) što je prekinulo rad topionice, odnosno nije bilo proizvodnje bakra. U smanjenom obimu PK je ipak radio.

Svi prethodno nabrojani faktori uticali su negativno na rad PK, tj. povećavali razliku između planiranog i ostvarenog obima otkopavanja, posebno u slučaju otkrivke.

Nestabilno geopolitičko stanje u Državi najbolje ilustruje pomenuta činjenica da je ona u analiziranom periodu imala tri oblika. Pored promena država taj period može da se rasčlani i na manje periode koji su bili karakteristični, i to:

Period od početka rada površinskog kopa (1979.) do početka raspada SFRJ (1992.).

Period od raspada SFRJ i formiranja SRJ (1992.) do raspada SRJ i proglašenja RS (2006).

U okviru prethodnog period desilo se i sledeće:

- uvedene su sankcije (1992-1996.),
- izvršeno bombardovanje (24. 03. - 09. 07. 1999.),
- došlo do političkih promena u Srbiji (05. 10. 2000.).

Nakon 2000. godine, proizvodnja na PK izrazito pada, pošto u periodu devedesetih nije ulagano dovoljno u obnavljanje i zamenu osnovne i pomoćne rudarske mehanizacije. Veliko zakašnjenje u raskrivanju ležišta, nastalo kao posledica neregularnih uslova rada PK tokom devedesetih, uz vrlo nisku cenu bakra na LME, nije davalo mogućnosti za veću proizvodnju rude, odnosno bakra, i samim tim nije bilo mogućnosti za obnavljanje zastarele i dotrajale rudarske mehanizacije. Kako ni država Srbija nije imala sredstava za obnovu rudarske mehanizacije, u saradnji sa Svetskom Bankom, procenjeno je da je najbolje rešenje privatizacija, zbog čega je započeta priprema, koja je pre svega podrazumevala smanjenje broja radnika i privatizaciju pojedinih celina.

Karakteristični periodi u poslovanju RTB-a vezani su za sledeće događaje:

- prvi pokušaja privatizacije-tender mart 2007.,
- drugi pokušaja privatizacije-tender februar 2008.,
- prvi pokušaja nalaženja strateškog partnera-tender novembar 2008. i,
- drugi pokušaja nalaženja strateškog partnera-tender februar 2009.

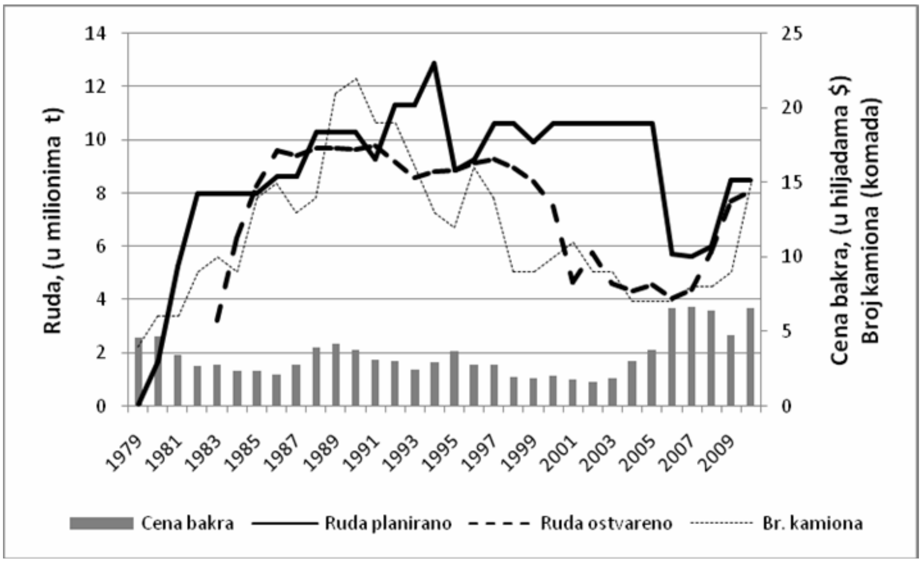
Nakon neuspelog pokušaja nalaženja strateškog partnera, na Državnom nivou, doneta je odluka da se investira u RTB kako bi se on osposobio za samostalno poslovanje, a opet sa ciljem da se privatizuje ili zainteresuju strateški partneri. To je omogućilo nabavku nove osnovne i pomoćne mehanizacije i oživljavanje proizvodnje, pri čemu je taj trend nastavljen i u narednim godinama.

Stanje opreme na analiziranom PK najbolje se reprezentuje se preko stanja transportne opreme, tj. broja kamiona u eksploataciji.

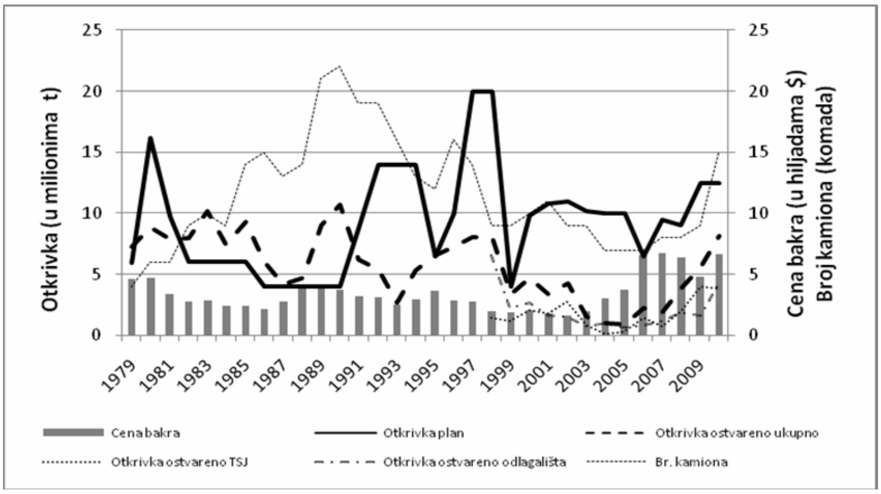
Sa istim ciljem posebno se ističe period od kada je za transport jalovine korišćen i kombinovani sistem transporta, odnosno period od 1998. godine. Za njega je karakteristična činjenica da još uvek nije izgrađen u potpunosti, odnosno da je od projektovane dve drobilice za sada ugrađena samo jedna, pa tako u odnosu na projektovani kapacitet (4 700 t/h), može da radi samo sa polovinom (2 350 t/h).

Pored prethodnog karakteristično je i to da je njegovo iskorišćenje bilo izrazito malo, prosečno oko 10% (od polovine projektovanog kapaciteta), pre svega zbog nedostatka kamiona. Situacija se popravila u 2009. i 2010. godioni kada su u eksploataciju uvedeni novi kamioni, odnosno kada se povećao njihov broj, pa se iskorišćenje sistema povećalo na oko 30%.

Pošto su prethodno obrazloženi glavni uticaji faktori na godišnji kapacitet otkopavanja na PK na **dijagramima broj 1 i 2** dati su odgovarajući pregledi za rudu i otkrivku, zajedno sa pregledom cene bakra i broja aktivnih kamiona.



Dijagram br. 1. *Pregled godišnjih kapaciteta otkopavanja rude, cene bakra i broja aktivnih kamiona za period od 1979. do 2010. godine*



Dijagram br. 1. *Pregled godišnjih kapaciteta otkopavanja rude, cene bakra i broja aktivnih kamiona za period od 1979. do 2010. godine*

Karakteristično za date preglede je sledeće:

- ostvarivani su manji godišnji kapaciteti iskopina od projektovanih, prvenstveno na otkopavanju otkrivke s obzirom da je kapacitet na rudi morao da bude konstantan zbog nesmetanog rada flotacije,
- kapacitet otkopavanja bio je u direktnoj zavisnosti od broja aktivnih kamiona, pri čemu je ta zavisnost izrazita u slučaju otkrivke,
- uvođenje kombinovanog sisetma transport nije značajnije uticalo na povećanje kapaciteta otkopavanja otkrivke, s obzirom da on nije kompletno izgrađen,
- cena bakra značajno je uticala na povećanje kapacitet otkopavanja samo u periodu posle 2006. godine i
- glavni uticajni faktori na kapacitet otkopavanja bili su izvan PK, odnosno u sferi globalne svetske i državne politike

5. ZAKLJUČAK

Otkopavanje rude bakra na površinskom kopu Veliki Krivelj, koji posluje u okviru RTB BOR-Grupe u Srbiji, čini glavnonu ukupne proizvodnje, u koju su sada uključeni i jama Bor i Rudnik bakra Majdanpek.

Da bi se sagledala dosadašnja eksploatacija na ovom PK izvršena je analiza ostvarenih godišnjih kapaciteta i uticajnih faktora za period od otvaranja 1979. do 2010. godine.

Analiza podrazumeva upoređivanje projektovanih godišnjih kapaciteta otkopavanja sa ostvarenim, zbog čega je dat odgovarajući pregled važećih rudarskih projekata, po kojima je vršeno otkopa-vanje, i podataka o otkopanim količinama rude i otkrivke.

Godišnji kapaciteti otkopavanja pokazuju sledeće trendove:

- povećanje od otvaranja do 1990. godine
- stagnacija od 1990. do 2005. godine
- povećanje od 2005. nadalje

U poglavlju broj 3 navedeni su glavni uticajni faktori na kapacitet otkopavanja koji su razvrstani na sledeći način:

- svetska geopolitička situacija (cena bakra),
- geopolitičko stanje u Državi,
- karakteristični periodi u poslovanju RTB-a
- karakteristični periodi u poslovanju RBB-a i
- stanje opreme na PK.

S obzirom na to da je RBB zavisno preduzeće u okviru RTB-a logično je da je njegovo poslovanje pod direktnim uticajem RTB-a.

Nakon što su prezentovani projektovani i ostvareni godišnji kapaciteti otkopavanja i glavni uticajni faktori izvršena je odgovarajuća analiza, koja je pokazala:

- da su ostvarivani manji godišnji kapaciteti iskopina od projektovanih, ali prvenstveno na otkopavanju otkrivke s obzirom da je kapacitet na rudi morao da bude konstantan zbog nesmetanog rada flotacije,
- da je kapacitet otkopavanja u direktnoj zavisnosti od broja aktivnih kamiona, pri čemu je ta zavisnost izrazita u slučaju otkrivke ,
- da uvođenje kombinovanog sisetma transport nije značajnije uticalo na povećanje kapaciteta otkopavanja otkrivke,
- da je cena bakra značajno je uticala na povećanje kapacitet otkopavanja samo u periodu posle 2006. godine i
- da su glavni uticajni faktori na kapacitet otkopavanja bili izvan PK.

Imajući u vidu da je cena bakra izuzetno visoka i da se prognozira duže zadržavanje takvog trenda otkopavanje rude bakra na površinskom kopu Veliki Krivelj dobija na značaju, kao i ukupno poslovanje Rudarsko topioničarskog baseba Bor-Grupe.

Takvo stanje omogućava dobijanje kredita za obnavljanje opreme i dalje povećanje proizvodnje koja će biti rentabilna.

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ANALYSIS OF REALIZED ANNUAL CAPACITIES OF EXCAVATION AND INFLUENCING FACTORS AT THE OPEN PIT VELIKI KRIVELJ OPERATING WITHIN RTB BOR-GROUP, SERBIA***

Abstract

This paper provides an overview and analysis of realized annual excavation capacities at the open pit Veliki Krivelj, which operates within RTB BOR-Group in Serbia. The analysis included the main influencing factors.

The analyzed period started in 1979 when the works started work on excavation of overburden, and finished in 2010.

The analysis showed that lower annual capacities of overburden were realized than designed capacities, but primarily on the excavation of overburden due to a fact that the capacity of ore had to be constant for the smooth operation of flotation.

The analysis also showed that the capacity of excavation directly depends on number of active trucks, where this dependence is distinctive in the case of overburden.

***Key words**: open pit, Veliki Krivelj, capacity, excavation*

1. INTRODUCTION

The open pit Veliki Krivelj, hereinafter referred to as OP, in the east of Serbia, has been operating within the Copper Mines Bor, hereinafter referred to RBB that is within the Mining and Smelting Basin - Group, hereinafter referred to as RTB.

Production of copper concentrate from ore from OP represents about 75% of the total production of RBB, while participation of ore from OP is about 90% of total ore excavated with a trend of further increasing [1].

Main Mining Project of excavation and processing of copper ore in the deposit Veliki Krivelj for capacity of 8 x 10^6 tones of wet ore per year [2] from 1978 covered the opening of OP and its further exploitation until 2000 with the aforementioned capacity of 8 million tones of wet ore per year.

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Disclosure of the deposit that is excavation of overburden began in 1979 with simultaneously works on building capacity for its processing, which included the primary, secondary and tertiary crushing and sieving, grinding and flotation.

At the end of the 1982, excavation of ore began and its processing to the final product, i.e. copper concentrate.

During the analyzed period until 2010, the designed annual capacities of excavation were accomplished with varying degrees of success where it is dependent on many factors, and this paper covers only the main ones.

After realized analysis, it can be concluded that OP mainly realized less annual capacities of ore and overburden excavation capacities in relation to the designed, but that in some periods, the difference was significant.

However, it is important to note that the factors that influenced this were often "out" of OP or in the sphere of global trends in politics and economy and strategy of the Serbian state, which inevitably dictated the strategy for RTB, therefore the OP.

In 2009 and 2010 there was a certain investments in the acquisition a part of basic and auxiliary equipment, which resulted in a partial increase in mining capacity, which is still below the required level, considering the multi-year backlog in the overburden of deposit. New development plans required a development of a new Additional mining project on excavation and processing of copper ore in the deposit Veliki Krivelj for capacity of 10.6 x 10^6 tones of wet ore per year. Special attention is focused on the excavation of overburden because the capacity of overburden was until now lower than the planned due to this many years delay occured, which should be compensated by this Project.

2. REVIEW OF DESIGNED AND REALIZED ANNUAL CAPACITIES OF EXCAVATION

Analysis of the annual capacities means that they are compared with the planned ones, defined by appropriate projects are defined, and in that sense, first gives an overview of mining projects that are developed for the needs of OP:

1. Main Mining Design for excavation and processing of copper ore in the deposit Veliki Krivelj for capacity of 8 x 10^6 tones of wet ore per year (developed in 1978). Under this Project, the excavation should be carried out until 2000, and it was realized until 1985.
2. Additional Mining Design for ore mining in the deposit Veliki Krivelj for capacity of 10.3 x 10^6 tones of wet ore per year (developed in 1985). Under this project, the excavation was carried out until 1991.
3. Additional Mining Design for ore mining in the deposit Veliki Krivelj for capacity of 12,9 x 10^6 tones of wet ore per year (developed in 1991). Under this project, the excavation was carried out until 1995.
4. Additional Mining Design for ore mining in the deposit Veliki Krivelj for capacity of 10.6 million tones of wet ore per year (developed in 1995). Under this project, the excavation was carried out until June 2006.
5. Additional Mining Design for excavation and processing of ore in the deposit Veliki Krivelj for capacity of 8 x 10^6 tones of wet ore per year (developed in 2006). Under this Project, the excavation is carried out nowadays.

During the observed period, some other projects were carried out that were not used, due to the fact that they did not get the approval of the competent Ministry of Mining. Therefore, in the previous review, were present only projects for which the works were done with stating the period of operational implementation.

Considering the relatively long period (1979-2010) of processing, only the annual capacities of excavation were analyzed, i.e. ore and overburden, planned and realized. Therefore, the problem occurred regarding to the planned capacities in the years of their changes. It was overcome so as the planned annual capacity is adopted that is the authoritative and defined by the new projects.

As it is written, the Additional Mining Design of excavation and processing of copper ore in the deposit Veliki Krivelj is ongoing for capacity of 10.6 x 10^6 tones of wet ore per year where it is planned to be completed during 2011 and to begin its implementation in the same year.

RBB management has decided to increase the mining capacity to increase the volume of copper concentrate production which is a prerequisite for the employment of capacity of the reconstructed smelter which should be completed by November 2013. In accordance with that decision, the procurement of basic and auxiliary equipment for OP was made.

3. INFLUENCING FACTORS ON THE CAPACITY OF MINING

Copper, as the final product RTB Bor Group, is the stock exchange goods that are sold in several world "markets" which are directly affected by the global geopolitical relations and interests of big business. Thanks to these relations, the exchange copper price was changed in an incredible range of 1,319 $/t which was valid to 07.11.2001 to 10,350 $/t which is valid to 22.08. 2010. The highest regarding to the lowest price is higher for 785%, which is data that speaks for itself.

The conditions in our state and RTB have no significant impact to the movement of the world copper prices because the production is relatively small.

For the purposes of this study, the copper prices valid on the London Stock Exchange (LME) [3] were analyzed due to the fact that the sale of the Bor copper was contracted at those prices.

Therefore, the state of geopolitical relations at the global level, which directly affects the price of copper, regarded as the first and highest level of influencing factors, and indirectly on the situation on OP.

The second level of influence is realized within the state that makes crucial decisions about its development strategy, and therefore the development strategy of RTB because it is an important element of it. The importance is reflected primarily in the fact that it is a commodity which can always be sold on foreign markets and thus achieve the stable incomes.

The third level of impact is achieved within RTB, which is divided into subsidiary companies organized as limited liability companies.

One of the limited liability companies is RBB, and it is deemed as the fourth level of influence. The company has several sites where the site Jama Bor is the only site that now produces copper ore in addition to OP.

Finally, the influential factors within OP represent the fifth level, **Figure 1.**

Figure 1. *Levels of influential factors on the capacity of mining*

Hierarchy of influential factors, shown in **Figure 1**, shows that the conditions existing at OP have the least effect on the mining capacities what is an absurd.

The first ten years of production work of OP (1982-1992) were realized in relatively good conditions so that the acquired wealth is an experience which is still used today. This experience played a decisive role in the survival of OP at times when its work was in doubt.

Management of OP had always rational actions entailed in the modest or without any maneuver so the production took place in one shift with only a few trucks.

Influential factors at the OP level, determining the capacity of mining in real conditions include the followings:

- innovation the current truck-transport of overburden by introduction of combined transport,
- organization of well established work,
- rational-mining plans,
- optimum management of works in real terms,
- well utilization of the main and auxiliary equipment,
- expert and experienced workforce,
- solid maintenance in difficult conditions,
- satisfactory work discipline, and
- the fact that there were no large-scale disasters that are a consequence of force majeure.

Considering the above explanations, it is stated that the main influential factors in the capacity of excavation at OP had actually been largely beyond it.

From the point of realized annual mining capacities within OP, the most influential factors were the number of main and auxiliary machinery and its utilization, as well as the introduction of combined system for transport of overburden to the closed open pit Bor and its disposal in the mined area of this open pit.

- In this sense, further analysis involves the influential factors that are classified as follows:
- the world geopolitical situation and copper price,
- geopolitical situation in the country,
- typical business periods in RTB,
- typical business periods in RBB,
- condition of equipment at OP.

4. ANALYSIS OF REALIZED AND INFLUENTIAL CAPACITIES

Common influential factors related to the highest level, and the global geopolitical relations and interests of big business, in fact are manifested by one indicator, or through the world price of copper.

As it is seen from **diagrams 1 and 2**, the copper price on the LME had almost no impact on ore production at OP, except during the period 2000 - 2005 when the historic minimum of copper price was reached regarded to the constant value of dollar in 2005 [4]. Impact of very low copper price was reflected in the inability of renewal the mining equipment due to reduced revenues from the sale of copper.

However, in this case the global geopolitical relations and interests of big business were expressed in a specific form that is not directly related to the price of copper, but the relationship to the state in which the exploitation was carried out at OP.

Let us remind that in the analyzed period, the exploitation at OP was carried out in three official states, which inherited one another, with each following a minor, as follows:

- Socialistic Federal Republic of Yugoslavia (SFRY) until 1992
- Federal Republic of Yugoslavia (FRY) 1992 – 2006
- Republic of Serbia (RS) 2006 - further

Change from SFRY in SRY meant the civil war that greatly changed the world attitude towards these countries, in a negative sense.

Because of the situation in the state, which was assessed as poor, the UN imposed the economic sanctions that directly influenced the work of OP due to the fact that all equipment is imported. Formal sanctions lasted from 1992 until 1996.

On the other hand, during the existence of SRY, the NATO carried out bombing carried in which the targeted goals were in RTB (Substation 3) what sopped the work of the Smelter Plant and there was no production of copper. OP was still working with reduced levels.

All above listed factors had negative impact on the work of OP, i.e. they increased the difference between planned and realized volume of excavation, especially in the case of overburden.

Unstable geopolitical situation in the country is best illustrated by the mentioned fact that it had three forms in the analyzed period. In addition to the changes of states, this period of time period may be broken down into smaller periods that were characteristic as follows:

The period from the start of open pit mine (1979) until disintegration of SFRY (1992).

The period since disintegration of SFRY and establishment of the FRY (1992) and the proclamation of the Republic of Serbia (2006).

Within the previous period, the following happened:

- imposed sanctions (1992-1996),
- carried out bombing (24.03.–09.06.1999),
- political changes in Serbia (5.10.2000).

After 2000, the production droped markedly at OP because during the nineties it was not invested enough in the renewal and replacement of primary and ancillary mining equipment. The considerable delay in stripping the deposit, occurred as the result of irregular working conditions of OP in the nineties, with a very low price of copper on the LME, did not give opportunities for higher production of ore, that is copper, and therefore it was not able to renewal the outdated and worn-out mining equipment. Neither the Serbian state did not have the funds for the restoration of mining equipment, in cooperation with the World Bank, estimated that the best solution is privatization, which began in preparation, what primarily meant reduction in number of workers and privatization of individual units.

The characteristic periods in the business of RTB are related to the following events:

- the first attempt of privatization – tender March 2007,
- the second attempt of privatization-tender February 2008,
- the first attempt to find a strategic partner-tender November 2008, and
- the second attempt to find a strategic partner-tender February 2009.

After the unsuccessful attempt to find a strategic partner, at the state level, it was decided to invest in RTB in order to equip it for its own business and again in order to privatize or interested a strategic partner. This enabled the purchase of new primary and auxiliary machinery and the revival of production, with this trend continued in coming years.

The condition of equipment at analyzed OP is the most represented through the condition of t equipment, i.e. the number of trucks in service.

With the same aim in particular stands the period when the combined transport system was used for transport of waste, i.e. the period since 1998. It is characterized by the fact that it is still not completely built, or that of the designed two crushers currently was installed only one, so compared to the designed capacity (4.700 t/ h), it can work only with half (2.350 t/ h).

In addition to the previous, it is characteristic that its usage was extremely small, average about 10% (half of designed capacity), primarily due to a lack of trucks. The situation improved in 2009 and 2010 when the new trucks were introduced in the exploitation, or when their number increased, and the utilization of the system increased to about 30%.

As previously explained the main influential factors to the annual mining capacity at OP; **diagrams 1 and 2** present the appropriate reviews for ore and overburden, along with an overview of copper prices and the number of trucks.

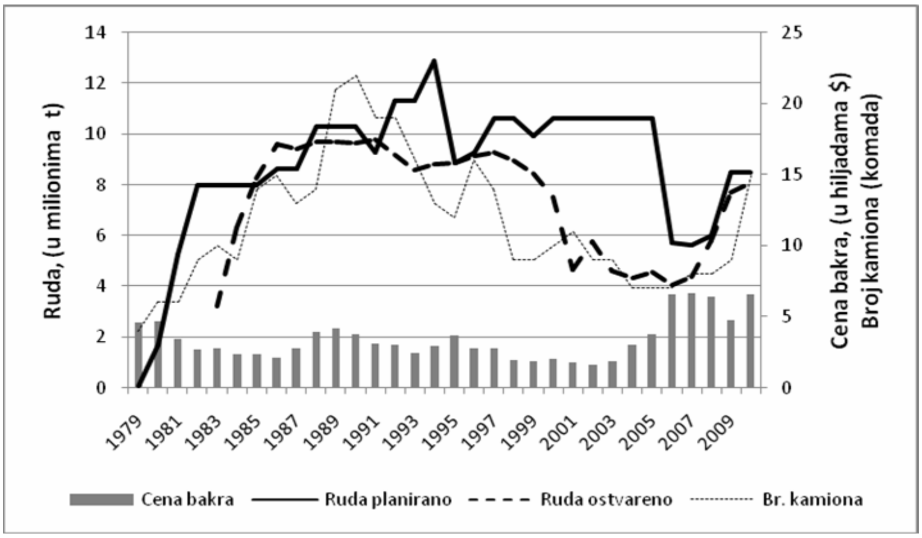

Diagram 1. *Review of annual capacity of ore mining, copper price and number of active trucks for the period from 1979 to 2010*

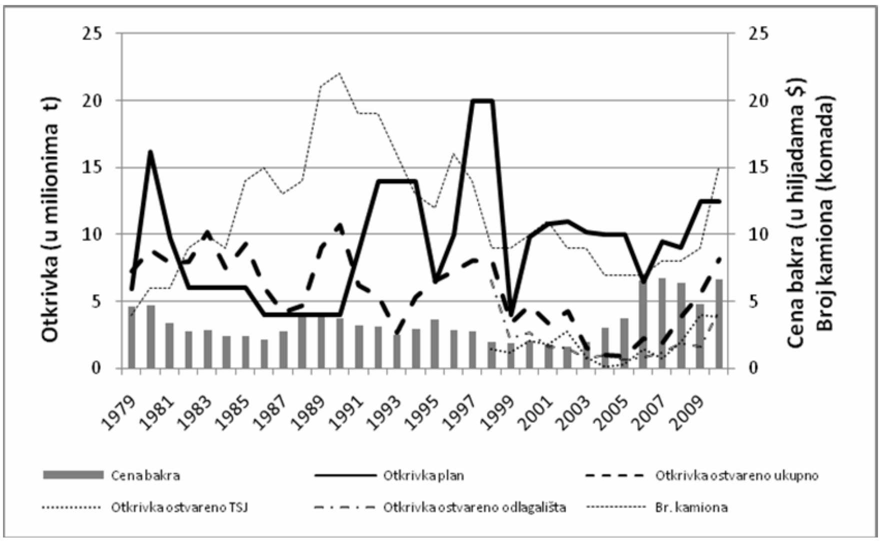

Diagram 1. *Review of annual capacity of overnurdenmining, copper price and number of active trucks for the period from 1979 to 2010*

Characteristics for given reviews are the followings:

- lower annual capacities of excavation were realized than designed ones, primarily in overburden mining regarding that the capacity of the ore had to be constant for the smooth operation of flotation,
- mining capacity directly depended on the number of active trucks where this dependence was explicit in the case of overburden,
- introduction of combined transport system did not significantly affect the capacity increase in excavation of overburden, as it is not completely built,
- copper price significantly increased the capacity of mining only in the period after 2006, and
- main influential factors to the mining capacity were outside of OP, or in the field of global and national policy.

5. CONCLUSION

Excavation of copper ore at the open pit Veliki Krivelj, which operates within RTB BOR Group in Serbia, is the main part of total production, which now also includes the underground mine Jama Bor and Copper Mine Majdanpek.

In order to recognize the current exploitation at this OP, an analysis of realized annual capacities and influential factors was performed for the period from opening 1979 to 2010.

The analysis involves comparing the designed annual capacities of excavation with the realized ones, due to this the appropriate overview of the current mining projects is given, according to which the excavation was done, and data on the excavated quantities of ore and overburden.

Annual capacities of excavation show the following trends:

- an increase from opening to 1990,
- stagnation from 1990 to 2005,
- increase from 2005 - further
- Chapter 3 lists the major influential factors on the capacity of mining that are classified as follows:
- the world geopolitical situation (copper price),
- geopolitical situation in the country,
- typical business periods in RTB,
- typical business periods in RBB,
- condition of equipment at OP.

Considering that RBB is a subsidiary within RTB, it is logical that its business is directly affected by RTB.

After presentation of projects and realization of annual capacities of mining and the main influential factors, a comprehensive study was realized, which showed that:

- lower annual projected capacities of excavation were realized than designed ones, but primarily on the excavation of overburden due to the capacity of the ore had to be constant for the smooth operation of flotation,
- the capacity of mining directly depends on the number of active trucks, with the explicit dependence in the case of overburden,
- the introduction of combined transport system did not significantly affect the increase the capacity of overburden excavation,
- the copper price significantly increased the capacity of mining only in the period after 2006, and
- the main influential factors were outside the excavation capacity at OP.

Considering that the copper price is extremely high and that the long time of this trend is predicted, the copper ore mining at the open pit Veliki Krivelj gaining importance, as well as a total business of the Mining and Smelting Basin Bor-Group.

This situation allows the loan to renew the equipment and further increase the production that will be profitable.

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TECHNOLOGICAL METHODS FOR USE THE GEOTHERMAL ENERGY BY DRILL HOLES

Abstract

Geothermal energy is the natural heat of the ground and involves accumulated heat in fluids and granite rocks. Geothermal energy can be used to generate electric and heat energy. In the first case it is a marginal source of energy, which can be used only in limited number of locations. In the second case, it is a geothermal energy with low enthalpy values, which is based on the use of the natural heat of the ground using the heat pumps. Process of acceptance the technologies that use the geothermal energy is very slow and the main problem is high price of installation the new plant. This paper presents some of technologies, based on the use of geothermal energy, as well as the investment and exploitation costs of plants whose operation is based on the use this form of energy.
***Key words**: geothermal energy, geothermal power plant, heat pump, costs*

1. INTRODUCTION

Geothermal energy is so called internal heat energy (resulting from formation of the planet Earth (before more than four billion years), which is the most partly regenerated by slow but continuous, natural decay of radioactive elements (primarily U - uranium, Th - thorium, and K - potassium), which are located in the granite rocks of the Earth crust, depth up to 30 km. Except by the radioactive decay, heat in the Earth crust is also created by crystallization and solidification of molten masses, exothermic chemical reactions, or friction in the movement of tectonic masses. Since that the time period is very long in which these processes take place, it is not possible to determine whether the Earth temperature rises or decreases. The potential of geothermal energy is enormous and estimated to 5000 EJ, out of which 10% is considered as exploitable in the next 100 years (*Cataldi,* 1999).

The energy transferred by heat during the day from deeper layers (core temperature exceeds 4000°C) to the surface of the Earth crust is about 3.3 to 7.5 kJ/m^2.

The Earth crust is relatively thin 5-50 km, medium density $\rho = \sim 2700$ kg/m^3, specific heat capacity $c = \sim 1000$ *J/kgK* and

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geothermal flux density 0.065 W/m^2 [1]. Geothermal heat flux density in Serbia is in average of about 0.06 W/m^2. The highest values are in the Pannonia basin, central Serbia and in the central part of Southern Serbia and the amount over 0.1 W/m^2.

The Earth temperature on the surface is mainly the consequence of solar radiation. However, the effect of solar radiation is observed only to depth of 30 m (where the temperature is almost constant), after what the effect of solar radiation is lost. Then, temperature of the Earth crust increases with depth increase. Temperature gradient is the highest close to the ground and is 30 K/km and decreases with the Earth depth increase (Table 1).

The values of temperature gradient given in Table 1 are not strict - in areas where tectonic touch plates - areas of strong seismic activity, active volcanoes and areas with good potential geothermal energy (hyperthermal areas) - gradient temperature can have value of over 100 K / km. These areas are of great importance for the use of geothermal energy and the geothermal power plants are mainly built in them (Indonesia, Philippines, Japan, New Zealand, Central America and the West Coast of USA).

Table 1. *Temperature gradient, depending on the depth of the country* [2]

Depth below the earth's rust $[km]$	**Temperature gradient** $[K/km]$	**Temperature T**
10	18-22	493-523
20	12-18	633-723
30	10-12	963-1133

Areas far from the boundaries of tectonic plates also can have values of temperature gradient about 50 K/km. This phenomenon is mainly associated with changes in geology and favorable hydrological characteristics of terrain.

METHOD OF USING THE GEOTHERMAL ENERGY

"For practical use geothermal energy it is necessary to take advantage of the natural circulation of water or create the conditions for such circulation. Essentially, principle of creating heat, is to so that brings water from the surface to deeper layers, in they are so warm and reheated again appears to surface. Water or water vapor that appears in sources of hot water or steam is due to deeper layers from the surface through permeable layers. Since in greater depths less porous rocks, and since that increases with depth and temperature of rocks can be count to greater depths in the crust there large mass of dry rocks where they accumulated substantial amounts of energy". [2] Earth's crust consists mainly of rocks, magma, and water so as to distinguish these groups of energy sources:

- petro thermal energy - energy from hot and dry rocks,
- hydro energy hot water in large depths,
- hydro energy sources of water vapor, and
- hydro energy sources of hot water.

Until a way for use the energy accumulated in the hot, dry rocks with hightemperature (700 - 1200°C) would be found, as an energy source would be only heat and water vapor that occurs at shallow depths. The use of energy accumulated in the dry rock would require a complex technology (enough deep drill holes), what is not yet profitable or technological developed, because at the present stage of technical development the drilling depth is limited to 7 km. [1] To use the energy accumulated in the hot rocks, it is necessary to crush the dry rocks at great depths below the earth surface as to get enough large surface area for heat transfer between the rocks and water that would be created by mechanical drill hole of crushed rocks. Thus heated water would be transferred to the surface through the other drill hole, which would be used to drive the turbine (Figure 1).

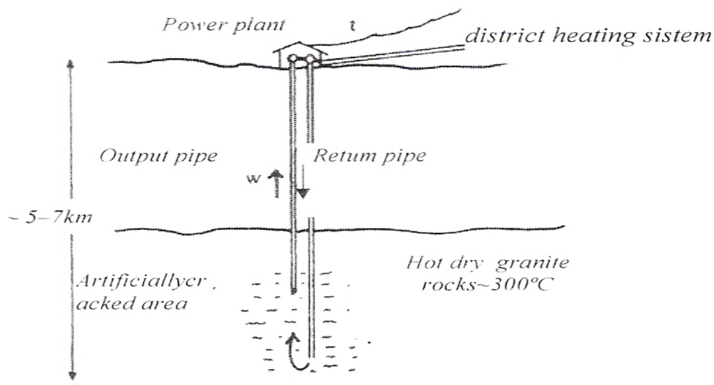

Figure 1. *System for use the energy accumulated in dry rocks* [1]

Water or water vapor that appears in the sources of hot water and water vapor reached through permeable layers into the aquifers (accumulations). Water accumulates heat of hot rocks and heat that comes from the larger depth and reaches a temperature of ~ 400°C. If water find a path to the surface either through wells or through the top layer of impermeable rock, it occurs in form of hot or boiling water (fumaroles), or the form of steam (geysers). When the water temperature is high enough when coming up on the earth's surface, it turns into steam which can be used to drive steam turbine.

POWER GENERATION

The first geothermal power plant (250 kW) operating at principle of dry steam commissioned in 1913 in Larderello near Siena (Italy). This plant is still in operation and the electricity supplies about one million households that is production of nearly 5000 GWh/year, representing about 10% of the total world production of electricity from geothermal sources. There are few examples of plants that use source dry steam, so it should be mentioned the biggest power plant "The Geysers" in the U.S. (Figure 2), As well as power plants "Matsukawa" in Japan, "Wairakei" in New Zealand and "Kamojang" in Java. Although the principle of dry steam is the simplest technology (used in dry steam temperatures above 235°C, which directly drives the turbines of generators, except when necessary removed: solid matter - the gravitational separator, or non condensing gases - in the system for venting), due to geographical constraints, most modern geothermal power plant uses *Flash* process.

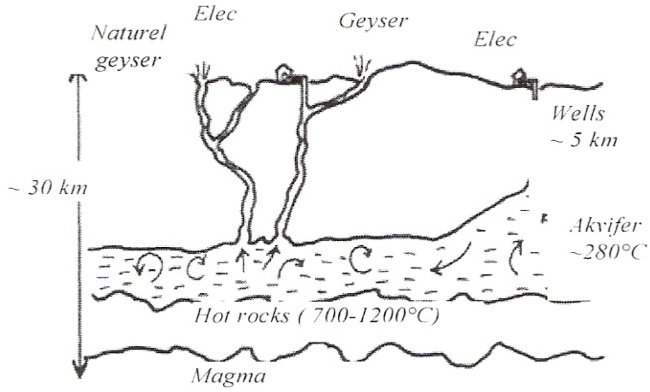

Figure 2. *Geothermal power plant in the hyper thermal field (Geysers geothermal field, California)* [1]

Flash process is based on the use of wet vapor of temperature above 180°C, which enters the separator where the pressure is much lower than the inlet fluid pressure, causing vapor steam that drives a steam turbine, and resulting in water trough the injection well returns to the aquifer. Figure 3 shows the simplified scheme of the flash single-phase process. Performance of geothermal plants with single-phase flash process can be improved by 37% using two-phase flash process that is for another 6% for the three-phase flash process. Most modern power plants use geothermal *Flash* multiphase process.

"In Serbia, in spite of the favorable opportunities for exploitation of thermal energy and other geothermal from geothermal water resources, there are favorable opportunities for exploitation of geothermal energy dry rocks. The exploitation of energy from this resource will start in due course taking into account the currently the minimum use of natural hot springs and mineral water. In Serbia outside of the Pannonian basin is 160 natural sources of geothermal water with a temperature higher than 15°C. The highest temperature of the water source in Vranjska Spa (96°C), then in Josanicka Spa (78°C), Sijerinskoj Spa (72°C) and others were studied".[3] The sources of geothermal water in Serbia with the highest temperatures could be used exclusively for use the most expensive and least efficient plants to convert geothermal energy into electricity with ***binary*** **process** (Figure 4).

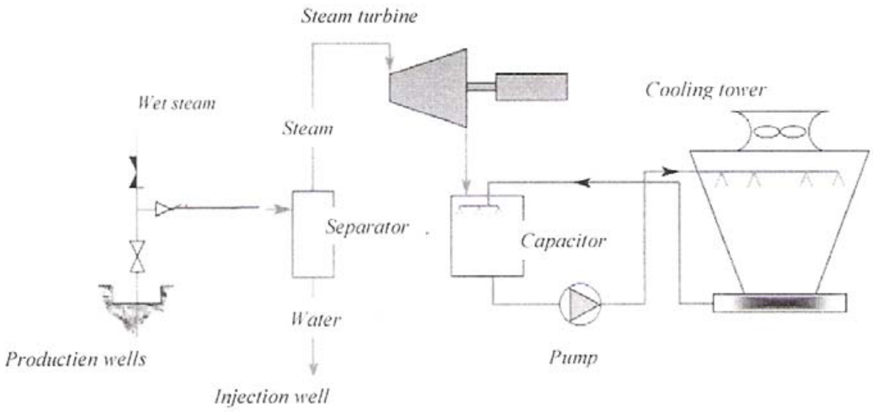

Figure 3. *Schematic view of the flash conversion process the geothermal energy into electricity*

Hydro fluid (primary fluid) temperature below 16ºC in the heat exchanger gives off heat secondary working medium (secondary or binary fluid), much lower evaporation temperature of 100ºC - mostly from hydrocarbons (typically isobutene or isopropane), which evaporates and starts the turbine blades of generator, and geothermal fluid is again returned to aquifer through the injection well). Advantage of the the principle is of greater efficiency and higher availability of respective tanks. An additional advantage is complete containment system, because the used water back to source, and thus the loss of water reduced. Most geothermal power plants, whose construction is planed, will use this principle [5].

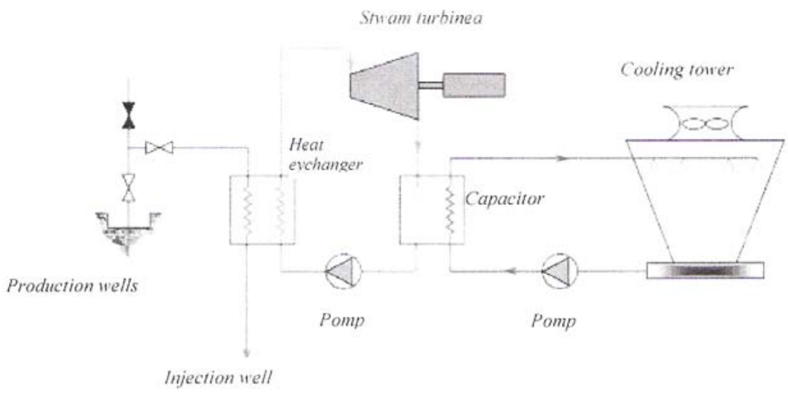

Figure 4. *Schematic presentation of a binary geothermal conversion process into electricity*

Using geothermal energy does not depend on weather conditions, season or time of day, and only limitation is that it can be used only in the place where the site, causing the current level of use of geothermal energy in the world at low compared to the existing potential. According to data of the International Geothermal Association, total installed capacity of geothermal power plants in the world during 2010 amounted to 10.52 GW, and is predicted to by the end of 2015, reaching as much as 16 GW[4]. Main barriers to more intensive use of geothermal energy are relatively high investments with only investment in research and geothermal sources are over 20% of the total cost. In addition to the adverse trend for t use of geothermal energy affects significantly strict conditions for the release of used geothermal water in the environment. New generation geothermal power plants emit only about 0.136 kg CO_2/ kWh, compared with the issue of CO_2 from power plants is enormous advantage in of the geothermal energy.

COSTS OF GEOTHERMAL POWER PLANT CONSTRUCTION

Cost of geothermal power plants depends on many factors: the location, the chemical properties of geothermal water, temperature of fluids, drilling rates, technology used, power plants, etc. World experience suggest that the most influential factor in determining prices geothermal power, price well, which in turn depends the depth of penetration. To the known prices of existing wells, the current price for example wells to a depth of 600 - 1100 m is between 220000 to 500000 €. Na (Figure 5) given the approximate value of investment costs geothermal power plants, depending on the technology (Principles) that are used in them. Operating and maintenance costs depend on installed capacity of geothermal power, and range from 0.37 to 0.6 c € / kW (power installed capacity of less than 5 MW) and from 0.2 to 0.37 c € / kW (installed power above 30 MW). Electricity prices obtained in geothermal power plants in the world is from 6 - 9 c € / kWh. The electricity production in Serbia is subsidized by the geothermal power plants from 7.5 c € / kWh.

Century exploitation of geothermal power plants is 30-40 years, and the time of construction is between 1-3 years.[5] Given that the energy sales contracts usually last from 0-20 years, a repayment period of the project to be incorporated in period of the contract of sale of energy, since the time of investment profitability is estimated at about 10 years.

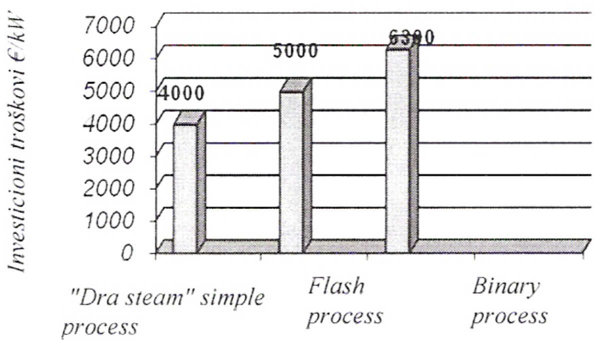

Figure 5. *The investment cost of geothermal power plants, depending on the production process* [4]

DIRECT USE OF GEOTHERMAL ENERGY

Republic of Serbia will not solve the problem production of electricity using geothermal power plants, primarily due to low temperature of geothermal water, but can seek opportunity for additional economic effects especially in direct application of geothermal resources in their immediate vicinity, for the purposes of: heating and hot water treatment, recreation, industry and Agriculture (Grain drying, heating of greenhouses etc.). Direct use of geothermal energy heating is much easier and cheaper, but because aggressive chemical composition of water, usually impracticable. Heating geothermal energy was recorded in Hungary, Japan, New Zealand, France, etc. In Iceland, Reykjavik, is the largest geothermal heating system, where 89% population is heated by this way, and we should mention the system to melt snow and ice in the settlements. Price geothermal energy for direct use in industry the world was 2-3 c € / kWh. If that energy is used for sports and recreational purposes, for balneology, cooling in summer energy prices in the world amounts of 2-5 c € / kWh. Cost of geothermal energy is used for heating is 1-6 c € / kWh.[6]

GEOTHERMAL HEAT PUMPS

If hydro resources are not available allow direct use of geothermal energy possible use of geothermal heat pumps. Geothermal heat pumps and heat pumps are devices that work on the thermodynamic principle of raising heat, i.e. heat from the body lower temperature transferred to a body of higher temperature without wasting mechanical work, which is several times smaller than the transferred energy, using a circular process left movement appropriate fluid. It is the devices that use geothermal energy accumulated in the soil and ground water **for air conditioning. Systems using geothermal pumps can be to:**

- open circuit (after the heat transfer secondary medium, the geothermal fluid is returned to or injection well or discharge into the canals, waterways or sewage,
- closed-circuit (heat is transferred to the country pipe system).

Geothermal heat pump system with horizontal tube called "collectors in the country" (Figure 6) are placed in a grid at a depth of between 1.5m and use the fact Sun's energy that is accumulated in the soil during summer. Lack of above system lies in the fact that during the winter of accumulated energy consumption, at the end winter soil temperature is about 5ºC. For larger objects recommended a system with a vertically mounted tube (probes) at a depth of 40 -100 m It is a relatively expensive systems, but the reliability, ease of handling, long-term solution for air conditioning facilities at any location, it becomes the solution.

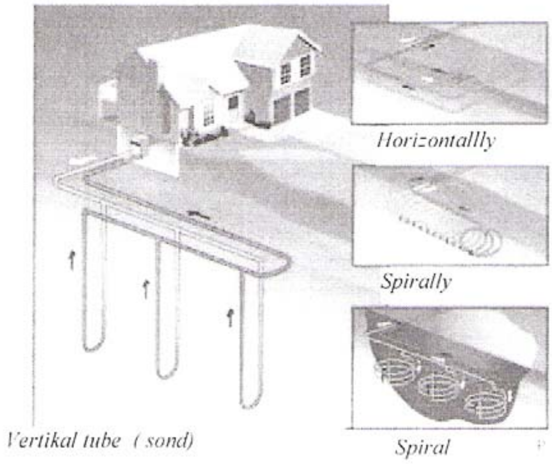

Figure 6. *Heating systems with geothermal heat pumps (closed-circuit process)*

Compared to the "collectors in the country" and the use of water from wells for heating and cooling of buildings, installation of the probes, is slightly more expensive but attractive variant heat pumps (including the account reliability, ease of handling and long term troubleshooting heating).

Geothermal heat pumps do not produce noise and almost that do not pollute the environment. Their use reduces the cost of air conditioning in residential buildings and office buildings for about 50%, and the life they longer than conventional systems, due to adequate protection from weather conditions. Several European countries have in recent years passed the legal provisions under which each building a brand new facility must be built heat pump for space heating. Only during 2008 in eight European countries the total number new installed heat pumps has increased by 50%. In Sweden, for example, there is mounting legal obligation heat pumps in each new building. Thanks to the support of this in the same state is mass is replacing the existing heating system heat pumps. The EU Council decided that in 2020 each newly built business and residential building must be energy independent, which will anyway contribute to increasing demand for heat pumps.

INVESTMENT AND OPERATION COSTS

Investment costs, as well as the time interval repayment Heat pump systems vary, depending on installed elements. A good representative for the analysis application of heat pump is a residential facility, area of 3800 m^2 in the street Dimitrija Tucovića 77, in Belgrade, where the parallel analysis of investment cost variations of the geothermal pump and geothermal energy split systems (Table 2) and variants with remote heating and air conditioners was made (Table 3). Specific thermal load of the building is $\dot{q} = 50 \frac{W}{m^2}$, total installed thermal capacity of 190 kW, power evaporator heat pump is 143 kW, and total installed cooling capacity of 200 kW.

Table 2. *Investment cost variations with geothermal heat pumps and air conditioners*

Variants Geothermia + split systems	**Price €**
Geosonda - the estimated geothermal probe PE-Xa, 25kom. by25m (effectively 115m)	40 000
Well drilling and placing probes for full bentonite (drilling cost 24 € / m)	75 000
Heat pump with a substation (max. electric power 48 kW +10 kW circulation pumps)	70 000
Installation of internal wall heating and cooling for the performance (1900 m^2)	130 000
Installation of air conditioners (30 pcs. at a cost of 400 € / piece)	12 000
Mortaring of wall heating (the difference in the price of 3 € / m^2 to1900 m^2 of wall panels)	6 000
Additional protection of the building (4cm extra insulation –surface façades 1500 m^2)	6 000
Triple glasses (the difference in cost compared to the double glass20 € / m^2 to 1200 m^2	24 000
Total	**363 000**

Table 3. *Investment cost alternative with split systems and district heating system*

Variants of the Belgrade district heating + air conditioners	**Price €**
Shore Belgrade power plants (3800 m^2 of space at a cost 32 € / m^2)	121 600
Connection to the main heating pipelines (price depends on the length of the route)	10 000
Distribution substation with radiator heating (material + run approx. 30 € / m^2)	114 000
Installation of air conditioners (60 pcs. at a cost of 400 € / piece)	24 000
Total	**269 600**

The difference in cost between investment these variants is 93 400 €, and € 24.58 in the investment per m^2 usable area of the building. Analysis of maintenance costs of both options are presented in Tables 4 and 5

Table 4. *The exploitation cost variations with geothermal heat pump*

Variants of geothermal energy						
Electricity consumption	**Unit measure** $[kW]$	**Hours of work** $[h]$	**Unit Price (€/kW)**	**Price (€/year)**	**Depreciation life (year)**	**Price €**
Heat pumps - Summer (passive and active cooling)	16	800	0,05	640	20	12800
Heat pumps Winter (max. utilization rate of 0.6)	35	2000	0,05	3500	20	70000
Split System	45	80	0,05	180	20	3600
Circulator pump	8	3600	0,05	1440	20	28800
System maintenance insurance included				4000	20	80000
Total €						**195200**

Table 5. *Exploitation cost alternative with split systems and district heating system*

Variants of the Belgrade district heating + split system						
Electricity Consumption	Unit measure $[kW]$	Hours of work $[h]$	Unit price (€/kW)	Price (€/year)	Depreciation (year)	**Price (€)**
split syistem	90	400	0,05	1800	20	36 000
	measures flat (per m^2)	Hours of work $[h]$	Unit price (€/m^2)	Months of the year	Depreciation life (year)	**Price (€)**
Belgrade district heating	3800		0,65	12	20	**592800**
Total €						**628 800**

Operation costs over the depreciable life of 20 years are higher in district heating systems through district heating for 433,600 € or 21,680 €/ year. Annual cost of heating and cooling in the variant with geothermal heat pumps and air conditioners amounted 2.57 €/ m^2 or 8.27 €/ m^2 in the variant with district heating systems and air conditioners. Investment in specified system with geothermal pumps would be worthwhile for a period of 4.3 years.

CONCLUSION

Geothermal energy is available at any time, regardless of current weather conditions, which is not case with, for example, wind or sun. The availability of geothermal plants is greater than 0.95, as far as the availability of coal-fired power plants or nuclear fuels. The limitation, however, the fact that it can be used only on geothermal site, resulting in the current level of geothermal energy in the world is low compared to the existing potential. The reason for this due to the fact that the capital investment large and that only investment in research geothermal source is over 20% of the total investment. Intensive use of geothermal energy is closely linked to the economic factor and use of this type of renewable energy in different sectors of life.

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UDK:622.79:330.1:546.56/.59(045)=861

Mile Bugarin, Gordana Slavković*, Vladan Marinković**

CENOVNI ASPEKT KORISNIH METALA IZ JALOVIŠTA**

Izvod

Zadnju deceniju karekteriše trend porasta cena obojenih i plemenitih metala na Svetskom tržištu. Bakar, zlato i srebro su vekovno poznatni proizvodi Borskog rudarenja. Eksploatcijom i prerdom bakra stvorena su i flotacijska jalovišta sa ispitanim sadžajima korisnih metala. Jalovišta koja sadrže značajno učešće metala mogu se eksploatisati i vrednovati korisne sirovine. Ovoj rad razmatra cenovni aspekt ovih sirovina odnosno kretanje cena bakra, zlata i srebra, kao i prognozu cena ovih metala. Cenovni aspekt metala je bitan deo vrednovanja proizodnje metala iz flotacijskih jalovišta.
***Ključne reči**: cena, bakar, zlato, jalovište, prognoza*

1. UVOD

Stručne analize pokazuju i da je eksploatacija flotacijskih jalovišta, odnosno dobijanje metala iz ove otpadne sirovine, daleko jeftinije od standardnog postupka rudarenja koje podrazumeva miniranje, otkopavanje, bušenje kojih u ovom slučaju nema. Ukoliko flotacijska jalovišta ne sadrži značajno učešće bakra i plemenitih metala sprovodi se postupak rekultivacije. Trend rasta cena metala intenzivirao je dalja geološka istraživanja kao i stručno-naučna usvršavanja tehnologija proizvodnje i prerade bakra kojim bi se obezbedila budućnost rudarenja i kvalitetnog života na ovim prostorima u skladu sa evropskim zakonodvstvom u pogedu zaštite životne sredine. Prilikom vrednovanja za potrebe studija i projekata, opredeljenje za cene bakra, zlata i srebra neće biti na osnovu trenutnih tržišnih cena već na osnovu prosečnih kretanja ovih cena u prošlosti i njihovih prognoza koje se baziraju na očekivanim ponudama i tražnjama za metalima, tj. zavise od privredne i političke situacije u svetu. Analiza cena metala obuhvata kako nihovo sagledavanje u prošlosti (po mogućnosti za što duži vremenski period), tako i razmatranje prognoza renomiranih institucija.

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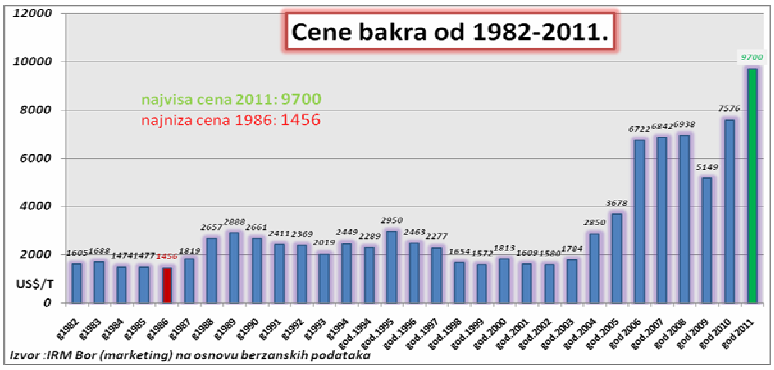
* *Institut za rudarsvo i metalurgiju Bor*
** *Rad je proizašao iz projekta broj 37001 "Uticaj rudarskog otpada iz RTB-a Bor na zagađenje vodotokova sa predlogom mera i postupaka za smanjenje štetnog dejistva na životnu sredinu", koji je finansiran sredstvima Ministarstva za prosvetu i nauku Republike Srbije*

2. KRETANJE CENE BAKRA

Cene metala nezavisno od proizvodne cene odredjuju pre svega cene na svetskim berzama (LME, NYMEX, COMEX, SHME...). Na sledećem grafiku prikazano je kretanje cena bakra od 1982-2011. godine.

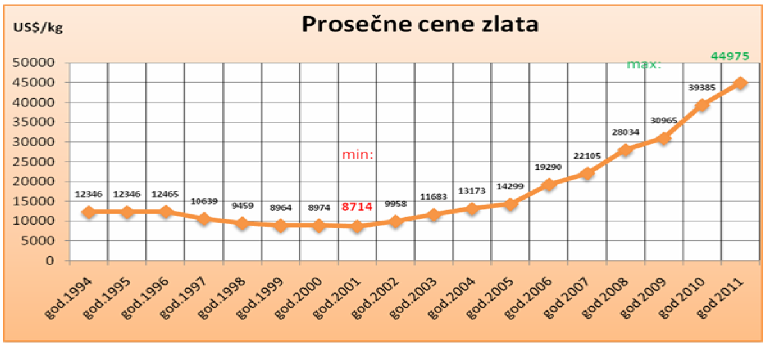


Grafik 1. *Kretanje cene bakra*

Grafik 1. pokazuje da je u posmatranom periodu najniža cena bakra bila 1986 godine 1456 US$/toni. Od 2004 godine postoji trend rasta cene bakra sa privremenim padom cene u 2009. godini. U 2011 godini cena bakra dostigla je istorijski maksimum: prosek 9700 US$/toni.

3. KRETANJE CENE ZLATA

Na sledećem grafiku prikazano je kretanje cena zlata od 1994-2011. godine.

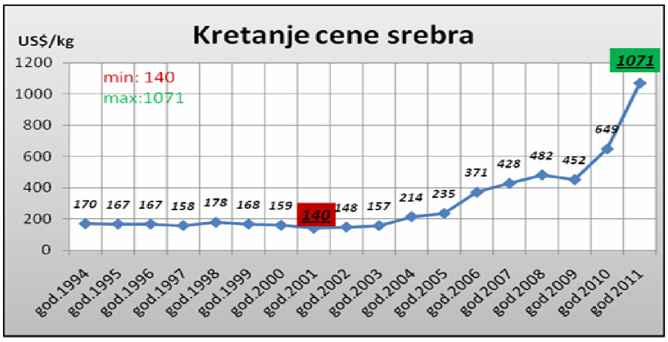


Grafik 2. *Kretanje cene zlata*

Grafik 2. pokazuje da je u posmatranom periodu najniža cena zlata bila 2001. godine 8714 US$/kg. Od 2001. godine postoji trend rasta cene zlata. U 2011. godini cena zlata dostigla je istorijski maksimum: prosek 44975 US$/kg.

4. KRETANJE CENE SREBRA

Na sledećem grafiku prikazano je kretanje cena srebra od 1994-2011. godine.



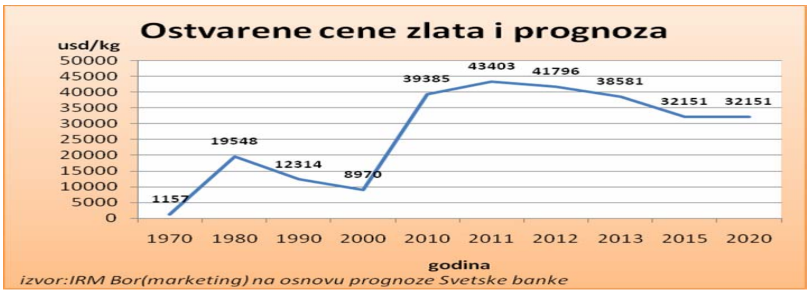
Grafik 3. *Kretanje cene srebra*

Grafik 3. pokazuje da je u posmatranom periodu najniža cena srebra bila 2001. godine 140 US$/kg. Od 2001. godine postoji trend rasta cene srebrani. U 2011. godini cena srebra dostigla je istorijski maksimum: prosek 1071 US$/kg.

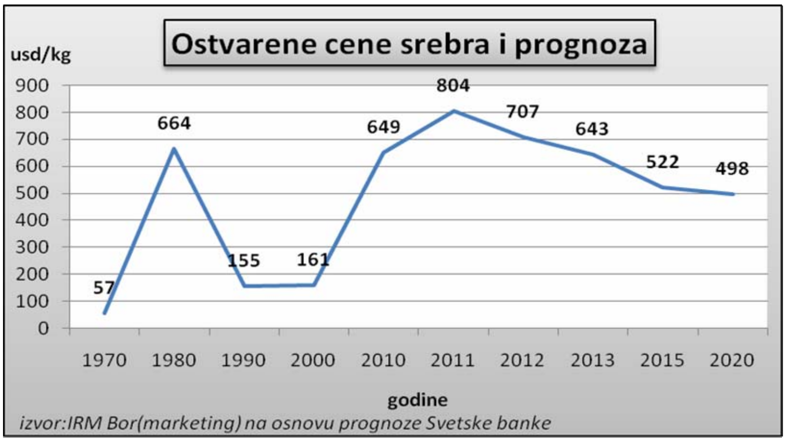
5. PROGNOZE CENA

Veoma je kompleksno i neizvesno predvideti buduće cene, ali se mogu uzeti trendovi sa berze metala kao i analize specijalizovanih eksperata. Kompleksnost se ogleda u preplitanju privredno-ekonomskih, političkih i geopolitičkih prilika u svetu. Najnovije prognoze Svetske banke (World bank prospect from january 2011.) predstavljene su na sledećim graficima za bakar, zlato i srebro kao i ostvarene cene ovih metala u predhodnim godinama.

Grafik 4. *Ostvarene cene bakra i prognoza*



Grafik 5. *Ostvarene cene zlata i prognoza*



Grafik 6. *Ostvarene cene srebra i prognoza*

Grafici 4, 5 i 6 pokazuju prosečne ostvarene cene metala u predhodnim godinama kao prognoze cena metala (Svetska banka) do 2020. godine.

6. ZAKLJUČAK

Analiza cena metala bakra, zlata i srebra obuhvata kako nihovo sagledavanje u prošlosti (po mogućnosti za što duži vremenski period) tako i razmatranje prognoza renomiranih institucija kao što je Svetska banka.

Najniža cena bakra u posmatranom periodu bila je 1986. godine 1456 US$/toni. Od 2004 godine postoji trend rasta cene bakra sa privremenim padom cene u 2009. godini. U 2011. godini cena bakra dostigla je istorijski maksimum: prosek 9700 US$/toni. Dugoročna prognoza cene za ovaj metal je oko 6000 US$/toni.

Najniža cena zlata bila 2001. godine 8714 US$/kg. Od 2001 godine postoji trend rasta cene zlata. U 2011 godini cena zlata dostigla je istorijski maksimum: prosek 44975 US$/kg. Dugoročna prognoza cene za ovaj metal je od 32000-40000 US$/kg.

Najniža cena srebra bila 2001 godine 140 US$/kg.Od 2001 godine postoji trend rasta cene srebra. U 2011 godini cena srebra dostigla je istorijski maksimum: prosek 1071 US$/kg. Dugoročna prognoza cene za ovaj metal je oko 500 US$/kg.

Prilikom vrednovanja metala za potrebe studija i projekata ,opredeljenje za cene ovih metala neće biti na osnovu trenutnih tržišnih cena već na osnovu prosečnih kretanja ovih cena u prošlosti i njihovih prognoza koje se baziraju na očekivanim ponudama i tražnja-ma za metalima,tj. zavise od privredne i političke situacije u svetu.

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Mile Bugarin, Gordana Slavković*, Vladan Marinković**

PRICING ASPECT OF VALUABLE METALS FROM TAILING DUMPS**

Abstract

The last decade is characterized with the trend of higher prices of ferrous and precious metals on the world markets. Copper, gold and silver have been produced for centuries in the Bor mining complex. Flotation tailing dumps were created by copper mining and processing with tested contents of valuable metals. Tailing dumps with significant content of metals can be exploitated and useful raw materials can be evaluated.

This paper discusses the pricing aspect of these raw materials or price movements of copper, gold and silver as well as the forecast of prices of these metals. Pricing aspect of metals is an important part of evaluation the production of metals from the flotation tailing dumps.

Key words: *price, copper, gold, tailing dump, forecast*

1. INTRODUCTION

Expert analyses show that the exploitation of flotation tailing dumps, or obtaining metals from these waste materials is far cheaper than the standard mining method including blasting, excavation and drilling which there are not present in this case. If the flotation tailing dumps do not contain a significant share of copper and precious metals, the reclamation process is carried out. The upward trend in metal prices has intensified further geological explorations as well as the expert-scientific improvements of copper production and processing technology in order to ensure the future of mining and quality of life in this region in accordance with the European Legislation in terms of environmental protection.

In evaluation of metals for the needs of studies and projects, the commitment to the prices of copper, gold and silver will not be based on the current market prices but based on average movements of these prices in the past and their forecasts based on the expected supplies and demands for metals, i.e. they depend on the economic and political situation in the world. Analysis of metal prices includes both their perception in the past (preferably for longer

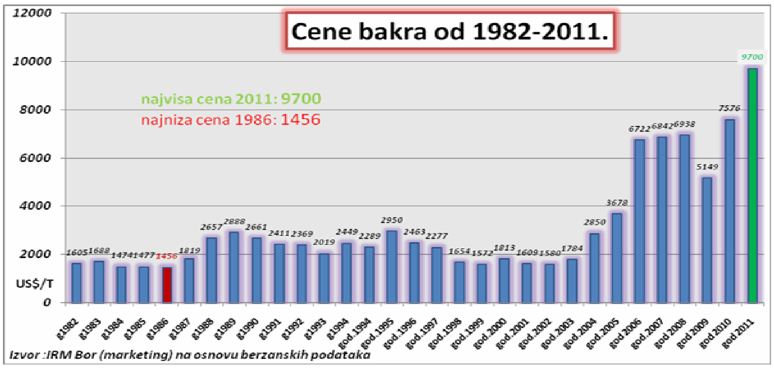
* *Mining and Metallurgy Institute Bor*

** *This paper is the result of Project No. 37001, "The impact of mining waste from RTB Bor on the pollution of surrounding water systems with the proposal of measures and procedures for reduction the harmful effects on the environment", funded by the Ministry of Education and Science of the Republic of Serbia.*

period of time p) and considering the forecasts of reputable institutions.

2. COPPER PRICE MOVEMENT

Metal prices independent of production price determine above everything else the prices on the world markets (LME, NYMEX, COMEX, SHME...). The following graph shows the movement of copper price from 1982-2011.

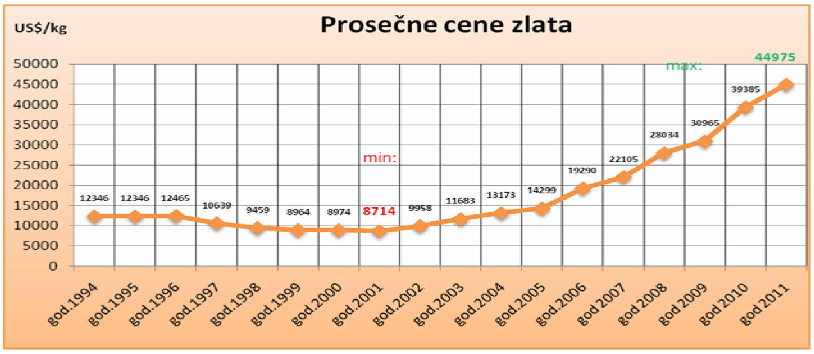


Graph 1. *Copper price movement*

Graph 1 shows that in this period the lowest price of copper in 1986 was 1456 US$ /t. Since 2004 there is the trend of increase the copper price with a temporary drop in copper prices in 2009. In 2011 the copper price reached a historical maximum: the average 9700 US$ /t.

3. GOLD PRICE MOVEMENT

The following graph shows the price movement of gold from 1994-2011.

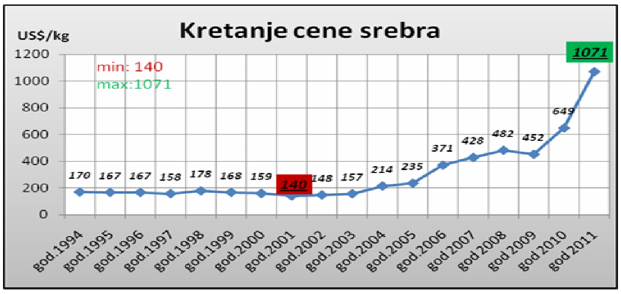


Graph 2. *Gold price movement*

Since 2001 there is the trend of increase the gold price. In 2011 the gold price reached a historical maximum: the average 44975 US$/kg.

4. SILVER PRICE MOVEMENT

The following graph shows the price movement of silver from 1994-2011.

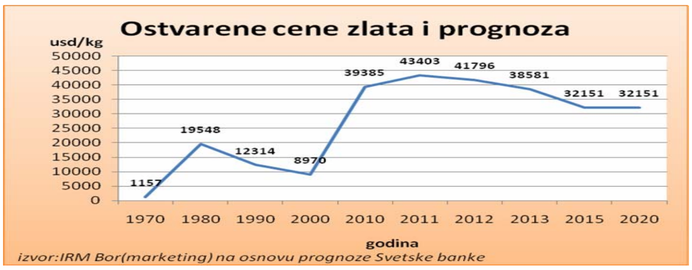


Graph 3. *Silver price movement*

Graph 3 shows that in this period the lowest price of silver in 2001 was 140 US$ /kg. Since 2001 there is the trend of increase the silver price. In 2011 the silver price reached a historical maximum: the average 1071 US$/kg.

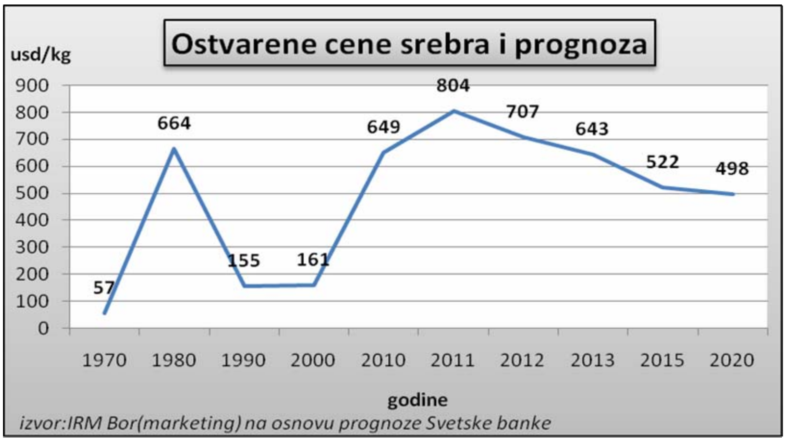
5. PRICE FORECASTS

It is very complex and uncertain to predict future prices, but the trends can betaken trends from exchange of metals as well as the analyses of specialized experts. The complexity is reflected in the intertwining of economic-commercial, political and geopolitical conditions in the world. The newesr forecasts of the World Bank (World Bank prospect from January 2011) are present in the following graphs for coper, gold and silver as well as the realized prices of these metals in the previons years.



Grafik 4. *Ostvarene cene bakra i prognoza*

Graph 5. *The realized gold prices and forecast*



Graph 6. *The realized silver prices and forecast*

Graphs 4, 5 and 6 show the average realized metal pricess in the previous years as well as the metal price forecasts (World Bank) until 2020.

6. CONCLUSION

Analysis of metal prices of copper, gold and silver covers both their their perception in the past (preferably for as long as possible period of time) and considering the forecasts of renowned institutions such as the World Bank.

The lowest price of copper in 1986 was 1456 US$ /t. Since 2004 there is the trend of increase the copper price with a temporary drop in copper prices in 2009. In 2011 the copper price reached a historical maximum: the average 9700 US$ /t. Long term forecast of price for this metal is about 6000 US$ /t.

The lowest price of gold in 2001 was 8714 US$ /kg. Since 2001 there is the trend of increase the gold price. In 2011 the gold price reached a historical maximum: the average 44975 US$/kg. Long term forecast of price for this metal is from 32000-40000 US$/kg.

The lowest price of silver in 2001 was 140 US$ /kg. Since 2001 there is the trend of increase the silver price. In 2011 the silver price reached a historical maximum: the average 1071 US$/kg. Long term forecast of price for this metal is about 500 US$/kg.

In evaluation of metals for the needs of studies and projects, the commitment to the prices of these metals will not be based on the current market prices but based on average movements of these prices in the past and their forecasts based on the expected supplies and demands for metals, i.e. they depend on the economic and political situation in the world.

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[2] Metal Bulletin Research London, GFMS, JM

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Gordana Slavković, Biserka Trumić*, Draško Stanković**

PROGNOZE CENA METALA PLATINSKE GRUPE U PROZVODNJI KATALIZATORSKIH MREŽA I HVATAČA**

Izvod

U radu se predstavljaju cenovni aspekti platinske grupe metala u proizvodnji katalizatorskih mreza i hvatača. Za ocenu ekonomske isplativosti bilo koje proizvodnje neophodno je razmotriti i ove tržišne aspekte proizvoda. Analizira se period kretanja cena platinske grupe metala: platine, paladijuma i rodijuma od trideset i više godina i prezentiraju se prognoze cena ovih metala poznatih banaka.

Ključne reči: *platinski metali, katalizatorske mreže, katalizatorski hvatači, prognoza cena*

1. UVOD

Platina i platinski metali su elementi osme grupe Periodnog sistema elemenata i pripadaju tzv. prelaznim metalima sa delimično popunjenom 4-d (rutenijum, rodijum, paladijum) i 5-d (osmijum, iridijum, platina) elektronskom orbitalom.

Posledice takve elektronske konfiguracije je niz karakterističnih osobina: povišena koroziona postojanost, vremenska otpornost na visokim temperaturama, niz povoljnih fizičko-mehaničkih svojstava metala platinske grupe.

U svetu potražnja za plemenitim metalima je u stalnom porastu kako u hemijskoj i industriji nakita, tako i auto industriji. Visoka cena platine i platinskih metala u svetu čini istraživanja veoma skupim, pa iz tog razloga planiranje eksperimenata i te kako velikog značaja ima.

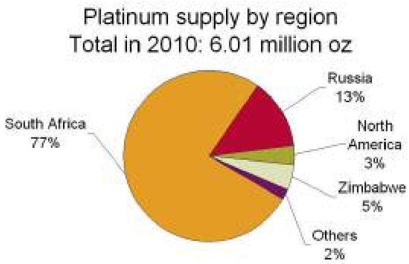
2. PLATINA

Najveći proizvođač platine je Južna Afrika, a najveća potražnja za njom je iz Kine. Platina se najviše koristi za izradu autokatalizatora i nakita. Takođe ima primenu u industriji hemije, nafte i elektrike. Za nakit se najviše koristi u Kini, a za autokatalizatore u Evropi. Od 2005. godine platina počinje da se koristi u medicini za lečenje raka i to najviše u zemljama Evrope i

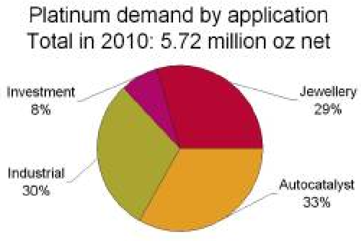
* *Institut za rudarstvo i metalurgiju Bor*

** *Rad je proizašao iz projekta broj 34029 "Razvoj tehnologije proizvodnje Pd katalizatora-hvatača za smanjenje gubitka platine u visoko temperaturnim procesima katalize", koji je finansiran sredstvima Ministarstva za prosvetu i nauku Republike Srbije*

Severne Amerike. Ponuda platine po regionima planete, kao i potražnja platine po oblastima primene prikakazani su na graficima 1 i 2.



Grafik 1. *Ponuda platine po regioni*



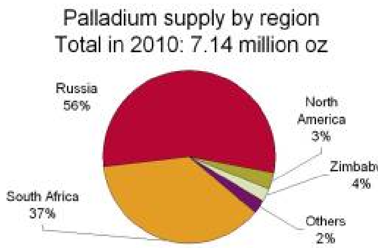
Grafik 2. *Potražnja za platinom po oblastima primene*

***(Izvor:** platinum.matthey.com)*

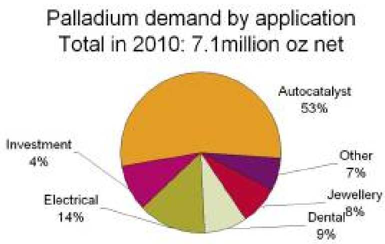
Prema graficima 1 i 2 može se primetiti da je najveći svetski proizvodjač platine Južna Afrika (77%) a da se ista najviše koristi za izradu autokatalizatora (33%) i u hemijskoj industriji (30%).

3. PALADIJUM

Najveći proizvodjač paladijuma je Rusija koja proizvodi 56% ukupne svetske proizvodnje paladijuma. Sledi Južna Afrika sa učešćem od 37 % u svetskoj proizvodnji. Ovaj plemeniti metal se najviše koristi za izradu autokatalizatora (53%), u industriji elektronike (14%), hemijskoj industriji i u zubarstvu. Ponuda paladijuma po regionima planete, kao i potražnja istog po oblastima primene prikakazani su na graficima 3 i 4.



Grafik 3. *Ponuda paladijuma po regionim*

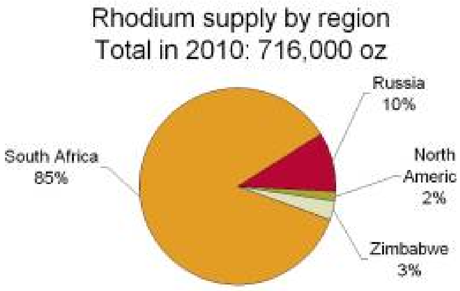


Grafik 4. *Potražnja za paladijumom po oblastrima primene*

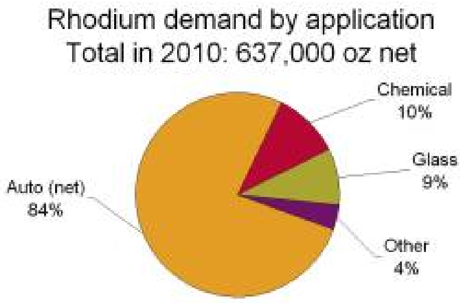
***(Izvor:** platinum.matthey.com)*

4. RODIJUM

Najvažniji resursi ovog elementa se nalaze u J.Africi (85%), u pesku reka Uralskih planina (10%), u Severnoj Americi. Najvažniji izvoznik rodijuma je J.Afrika (više od 80%), sledi Rusija. Rodijum se najviše koristi u autoindustriji za izradu autokatalizatora - 84%.



Grafik 5. *Ponuda rodijuma po regionima*

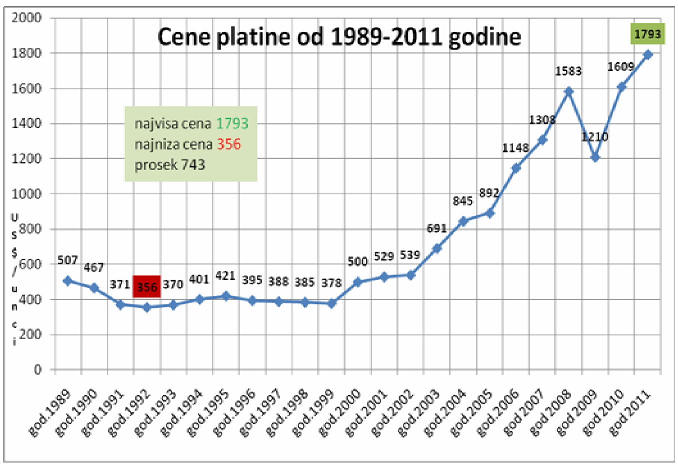


Grafik 6. *Tražnja za rodijumom po oblastrima primene*

(Izvor: *platinum.matthey.com***)**

5. KRETANJE CENE PLATINE

Analizairani period kretanja cene platine je od 1989-2011 godine. Kretanje cene platine prikazan je na grafiku 7.

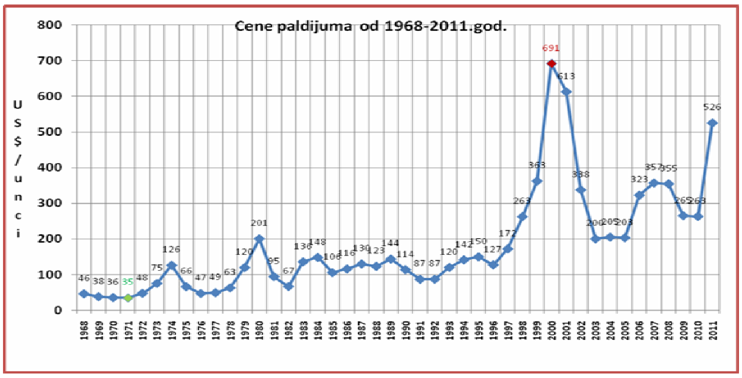


Grafik 7. *Kretanje cene platine*

Na osnovu grafika 7. može se zaključiti da je platina za posmatrani period, godine 1992., imala najnižu cenu od 356 $/unci tj. 11446 $/kg. Najvišu prosečnu cenu platina je dostigla 2011. god. – 1793 $/unci tj. 57646 $/kg.

6. KRETANJE CENE PALADIJUMA

Analizairani period kretanja cene paladijuma je od 1968-2011 godine. Kretanje cene paladijuma prikazan je na grafiku 8.

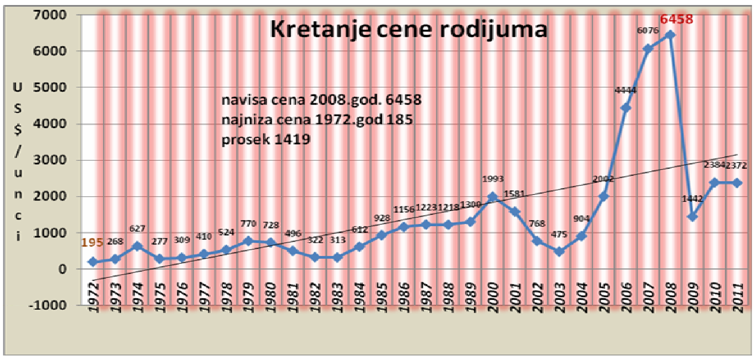


Grafik 8. *Kretanje cene paladijuma*

Na osnovu dobijenog grafika može se zaključiti da je paladijum u posmatranom period, godine 1971.imao najnižu cenu od 35 $/unci tj. 1125 $/kg. Najvišu prosečnu cenu paladijum je dostigao 2000 god. – 691 $/unci tj. 22216 $/kg.

7. KRETANJE CENE RODIJUMA

Analizairani period kretanja cene rodijuma je od 1972-2011 godine. Kretanje cene rodijuma prikazan je na grafiku 9.



Grafik 9. *Kretanje cene rodijuma*

Na osnovu predstavljenog grafika može se zaključiti da je rodijum u posmatranom period, godine 1972. najnižu cenu od 195 $/unci tj. 6269 $/kg. Najvišu prosečnu cenu rodijum je dostigao 2008 god. 6458 $/unci tj. 207.629 $/kg. Najviša prosečna cena rodijuma je u 2008., i to u junu maksimum od 9776 usd$/oz - što je ujedno i istorijski maksimum. Rast cene usledio je zbog povećane tražnje rodijuma pre svega iz Azije tj. Kine. U decembru 2008. cena rodijuma drastično je pala na 1214 $/oz.

8. PROGNOZE CENA METALA PLATINSKE GRUPE za 2011. i 2012.godini

Prognoze kretanja cena metala platinske grupe od strane nekih svetskih banaka su :

8.1. HSBC –HONKONG AND SHANGHAI BANKONG CORPORATION

Ova banka prognozira cenu platine od 1.750 $/unci i paladijuma od 675 $/unci u 2011. godini.

8.2. UBS –UNION BANK OF SWICHERLAND

Švajcarska banka za 2011. godinu prognozira za platinu cenu do 1.800 dolara za uncu, a za 2012. cena je u rasponu od 1.845 -1.833 $/unci. Ova banka prognozira cenu paladijuma na 625 dolara / unci u 2011. i 700 dolara / unci u 2012.

Za rodijum cena je u proseku od 2.925 dolara / unci u 2011. i 4.250 dolara za uncu u 2012. godini.

8.3. BOA MERRILL LYNCH-BANK OF AMERICA

Ova banka prognozira cenu platine u 2011.godini od 1.750 $/unci a u 2012.godini 1.650 $/unci. Očekuju da cena Pd bude do 650 $/unci 2011. I 550 $/unci 2012.

8.4. SCOTIAMOCATTA-BANK FROM CANADA

Ova banka planira da se paladijumom trguje u rasponu od 450-700 $/unci, a za cenu rodijuma predviđa raspon od 2.000-4.000 $/unci.

8.5. JPMORGAN-NEW YORK BANK

Ova banka očekuje prosečnu cenu platine u 2011.godini od 1.788 $/unci a za paladijum 675 $/unci.

8.6. BARCLAYS CAPITAL RITISH INVESTMENT BANK

U 2011. godini ova banka očekuje za platinu cenu od 1.690 $/unci a za paladijum 554 $/unci. Za 2012 god. prognoza cene platine je 1.590 $/unci a za paldijum prognoza je od 600 $/unci.

8.7. U 2011. godini analitičari **FORECAST SUMMARY-London Buliton Market Analylst/LBM** predviđaju kretanje cene platine do 1.813 US$/unci, a za paladijum do 814,65 $/unci.

9. ZAKLJUČAK

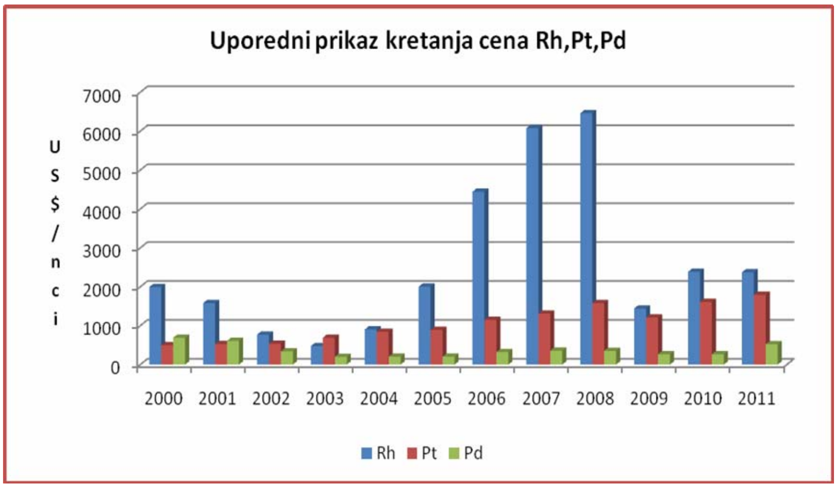
Analiza cena platinske grupe metala obuhvata kako nihovo sagledavanje u prošlosti (po mogućnosti za što duži vremenski period) tako i razmatranje prognoza renomiranih institucija. Najviša cena platine bila je 2060 USD/oz maja 2008. Prosečna cena platine za period januar-mart 2011. iznosi 1793 US$/oz – prosečni godišnji istorijski maksimim.

Najviša cena paldijuma bila je 2000. godine – 691 $/oz što je i najviša cena u proteklih četrdeset godina. Najniža cena Pd bila je 1971. god. i iznosila je 35 $ /oz.

Rodijum je 1972. godine imao najnižu cenu od 195 $/unci tj. 6269 $/kg. Najvišu prosečnu cenu rodijum je dostigao 2008 god. i ona je iznosila 6.458 $/unci tj. 207.629 $/kg.

Prilikom vrednovanja proizvoda od platinskih metala, za potrebe studija i projekata, opredeljenje za cene ovih metala neće biti na osnovu trenutnih tržišnih cena već na osnovu prosečnih kretanja ovih cena u prošlosti kao i njihovih prognoza koje se baziraju na očekivanim ponudama i tražnjama za metalima, tj. zavise od privredne i političke situacije u svetu.

Uporedni prikaz kretanja cena platine, rodijuma i paladijuma prikazan je na grafiku 11.



Grafik 11. *Uporedni prikaz kretanja cena metala*

Sa grafika 11. može se zaključiti da je najveće oscilacije cena imao rodijum tj. najskuplji metal iz grupe platinskih metala, pa su zbog toga i prognoze cena za ovaj metal retke i u rasponima.

LITERATURA

[1] Berzanski podaci (LME, NYMEX, COMEX)

[2] Metal Bulletin Research London, GFMS, JM

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Gordana Slavković, Biserka Trumić*, Draško Stanković**

PRICE FORECAST FOR PLATINUM GROUP METALS IN THE PRODUCTION OF CATALYST NETS AND CATCHERS**

Abstract

This paper presents the aspects of price for platinum group metals in the production of catalyst nets and catchers. To evaluate the economic viability of any production, it is necessary to consider these market aspects of product. The period of price variations of platinum group metals is analyzed: platinum, palladium and rhodium for more than thirty years and presented the price forecasts of those metals by known banks are present.

Key words: *platinum group metals, catalyst net, catalyst catchers, price forecast*

1. INTRODUCTION

Platinum and platinum group metals are elements of the eighth group of the periodic table of elements and belong to so-called transition metals with partially completed 4-d (ruthenium, rhodium, palladium) and 5-d (osmium, iridium, platinum) electronic orbital.

The consequences of such electronic configuration have a range of typical characteristics: the increased corrosion resistance, weather resistance to high temperatures, a series of favorable physical-mechanical properties of platinum group metals.

The world demand for precious metals is in permanent increase both in chemical and jewelry industry, and the auto industry.

High price of platinum and platinum group metals in the world makes the researches very expensive, and therefore the planning of experiments is very important.

2. PLATINUM

The largest producer of platinum is South Africa, and the greatest demand for it is from China. Platinum is mostly used for making automotive catalysts and jewelry. It also has usage in the chemical, oil and electricity industry. It is mainly used for jewelry in China, and for automotive catalysts in Europe. Since 2005, platinum began to be used in medicine to treat cancer, mostly in Europe and North America. The supplies of platinum by the planet regions as well as platinum demand by the fields of usage are shown in graphs 1 and 2.

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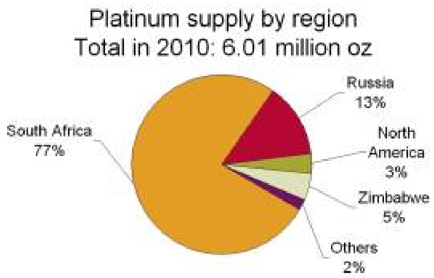
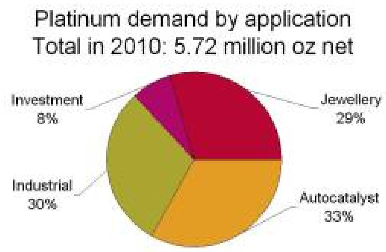
** *This paper is the result of Project No. 34029, “Development of technology of Pd catalyst-traps for reducing platinium loss in catalysis processes at high-temperature”, funded by the Ministry of Education and Science of the Republic of Serbia.*





Graph 1. *Platinum supply by regions* **Graph 2.** *Platinum demand per areas of usage*

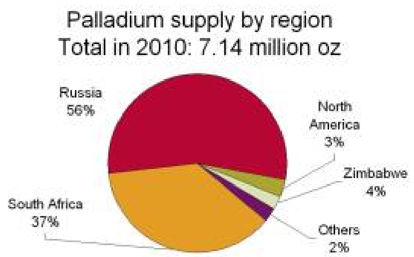
(Source: platinum.matthey.com)

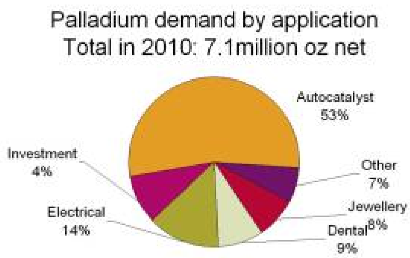
According to the graphs 1 and 2, it can be observed that the largest platinum producer in the world is South Africa (77%) and that the same is mostly used for making the automotive catalysts (33%) and in the chemical industry (30%).

3. PALLADIUM

The largest producer of palladium is Russia that produces 56% of total world production of palladium.

South Africa follows with a share of 37% in the world production. This precious metal is mostly used for making the automotive catalysts (53%), in the electronics industry (14%), chemical industry and dentistry. The supplies of palladium by the planet regions as well as palladium demand by the fields of usage are shown in graphs 3 and 4.



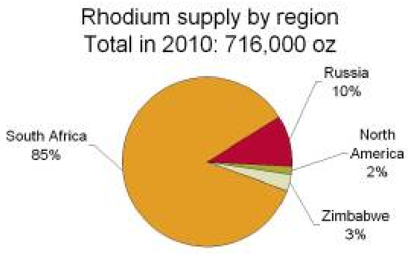


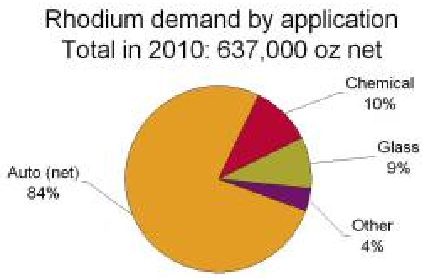
Graph 3. *Palladium supply by regions* **Graph 4.** *Palladium demand per areas of usage*

(Source: platinum.matthey.com)

4. RHODIUM

The most important resources of this element are in South Africa (85%), in the river sand of the Uralic Mountains (10%), in North America. The most important exporter of rhodium is South Africa (more than 80%), followed by Russia. Rhodium is mostly used in the automotive industry for making automotive catalysts - 84%.



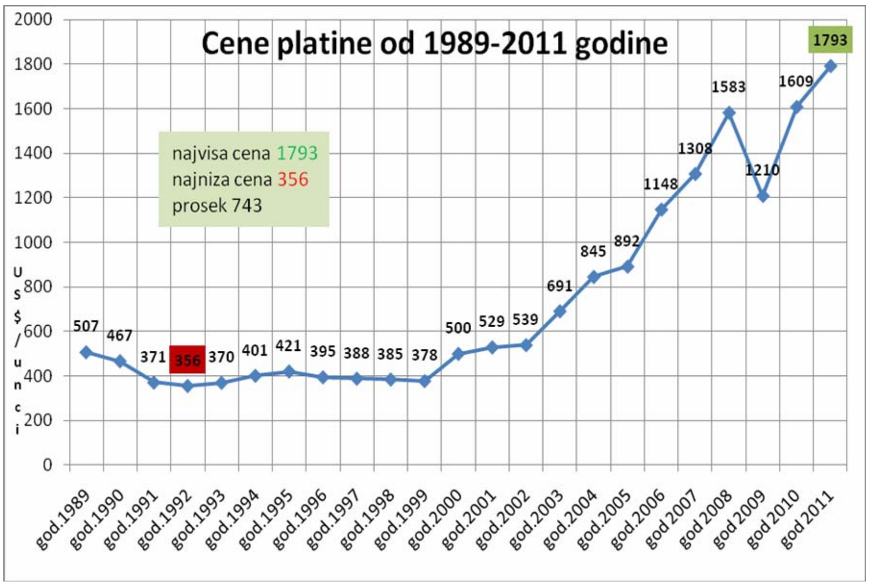


Graph 5. *Rhodium supply by regions* **Graph 6.** *Rhodium demand per areas of usage*

(Source: platinum.matthey.com)

5. PLATINUM PRICE MOVEMENTS

Analyzed period of platinum price movement is 1989-2011. Platinum price movement is shown in Graph 7.

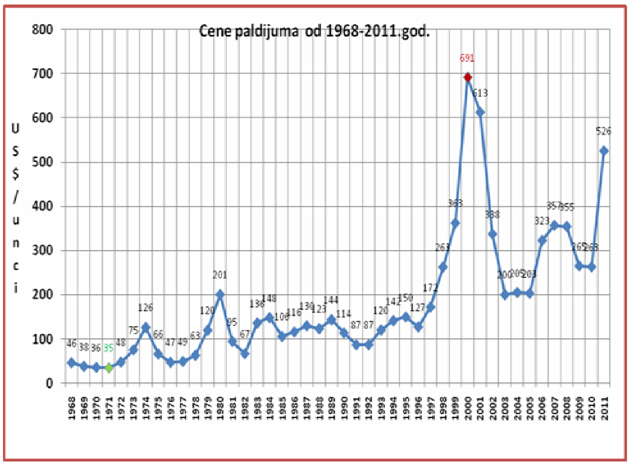


Graph 7. *Platinum price movements*

Based on Graph 7, it can be concluded that platinum for the reference period, in 1992, had the lowest price of 356 $/oz, i.e. 11,446 $/kg. Platinum reached the highest average price in 2011 - 1793 $/oz, i.e. 57,646 $/kg.

6. PALLADIUM PRICE MOVEMENTS

Analyzed period of palladium price movement is 1968-2011. Palladium price movement is shown in Graph 8.

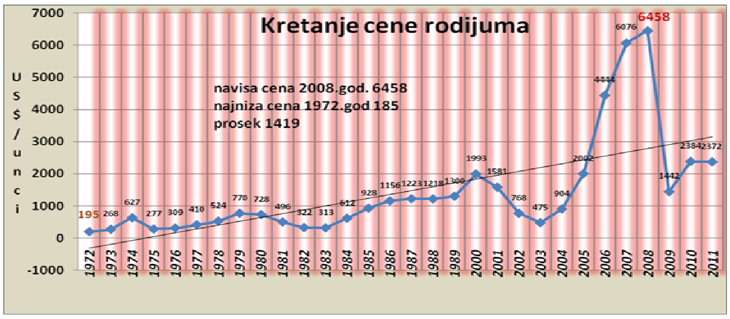


Graph 8. *Palladium price movements*

Based on Graph 8, it can be concluded that palladium for the reference period, in 1971, had the lowest price of 35 $/oz, i.e. 1,124 $/kg. Palladium reached the highest average price in 2000 - 691 $/oz, i.e. 22,216 $/kg.

7. RHODIUM PRICE MOVEMENTS

Analyzed period of rhodium price movement is 1972-2011. Rhodium price movement is shown in Graph 9.



Graph 9. *Rhodium price movements*

Based on Graph 9, it can be concluded that rhodium for the reference period, in 1972, had the lowest price of 195 $/oz, i.e. 6.269 $/kg. Rhodium reached the highest average price in 2008 – 6,458 $/oz, i.e. 207,629 $/kg. The highest average price of rhodium is in 2008 and, in June, to maximum of 9,776 $/oz, which is the historical maximum. The price increase was followed by the increased demand for rhodium, primarily from Asia, i.e. China. In December 2008, the price of rhodium drastically dropped to 1,214 $/oz.

8. PRICE FORECAST FOR PLATINUM GROUP METALS FOR 2011 AND 2012

Forecasts of price movements the platinum group of metals by some world banks are:

8.1. HSBC –HONKONG AND SHANGHAI BANKONG CORPORATION

This bank forecasts the price of platinum to 1,750 $/oz and palladium to 675 $/oz in 2011.

8.2. UBS –UNION BANK OF SWITZERLAND

Swiss Bank for 2011, forecasts the price of platinum price to 1,800 $/oz and for 2012, the price in the range of 1,845 - 1,833 $/oz. This bank forecasts the price of palladium to 625 $/oz in 2011, and 700 $/oz in 2012.

For rhodium, the price is an average of 2,925 $/oz in 2011 and 4,250 $/oz in 2012.

8.3. BOA MERRILL LYNCH-BANK OF AMERICA

This bank forecasts the price of platinum in 2011 to 1,750 $/oz, and in 2012 to 1,650 $/oz. It is expected that the price of Pd will be 650 $/oz in 2011 and 550 $/oz in 2012.

8.4. SCOTIAMOCATTA-BANK FROM CANADA

This bank plans to trade with palladium in the range of 450-700 $/oz, and forecasts the price of rhodium in the range of 2,000 – 4,000 $/oz.

8.5. PMORGAN-NEW YORK BANK

This bank expects the average price of platinum in 2011 to 1,788 $/oz, and for palladium to 675 $/oz.

8.6. BARCLAYS CAPITAL –BRITISH INVESTMENT BANK

In 2011, this Bank expects the price for platinum to 1,690 $/oz, and for palladium to 554 $/oz. For 2012, the forecast for platinum price is 1,590 $/oz, and for palladium to 600 $/oz.

8.7. In 2011, the analysts of the **FORECAST SUMMARY-London Billiton Market Analyst/LBM** forecast the movements of platinum price to 1,813 US$/oz, and for palladium to 814.65 $/oz.

9. CONCLUSION

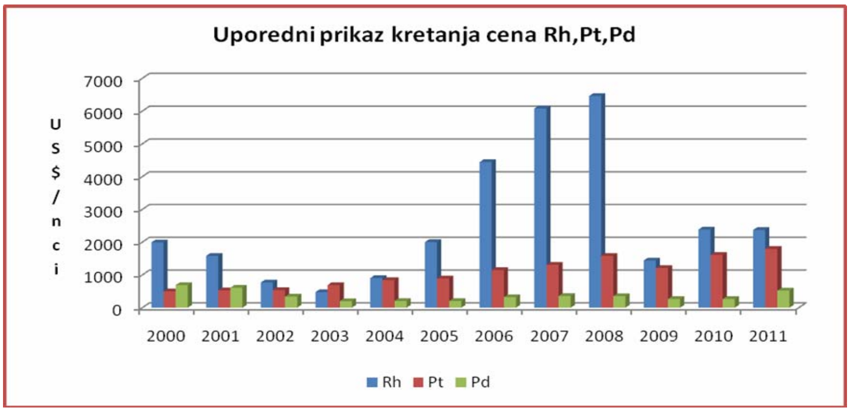
Analysis the price of platinum group metals includes both their shares in the past the past (preferably for a longer period of time) and considering the forecasts of reputable institutions. The highest price of platinum was 2,060 US$/oz in May 2008. Average platinum price for the period January-March 2011 is 1,793 US$/oz - average annual historical maximum.

The highest palladium price was in 2000 - 691 $/oz, which is the highest price over the past forty years. The lowest palladium price was in 1971, and it was 35 $/oz.

The lowest rhodium price was in 1972 to 195 $/oz, i.e. 6,269 $/kg. The highest average price of rhodium was in 2008 to 6,458 $/oz, i.e. 207,629 $/kg.

In evaluating the products of the platinum group metals, for studies and projects, the commitment to the prices of these metals will not be based on the current market prices, but based on the average movements of prices in the past and their forecasts, which are based on the expected supplies and demands for metals, i.e. they depend on the economic and political situation in the world.

Comparative review of the prices of platinum, rhodium and palladium is shown in Graph 11.



Graph 11. *Comparative review of movements the metal prices*

It can be concluded from Graph 11 that rhodium had the highest price oscillations, i.e. the most expensive metal in the platinum group metals, and therefore the price forecasts for this metal are rare and in the ranges.

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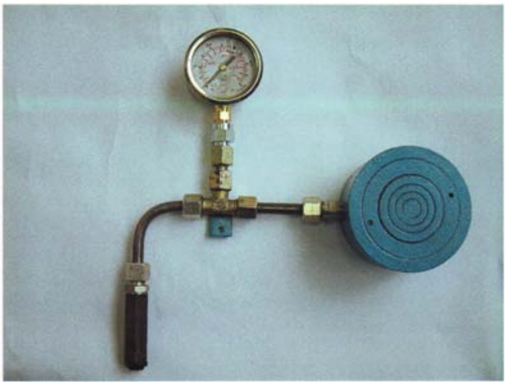

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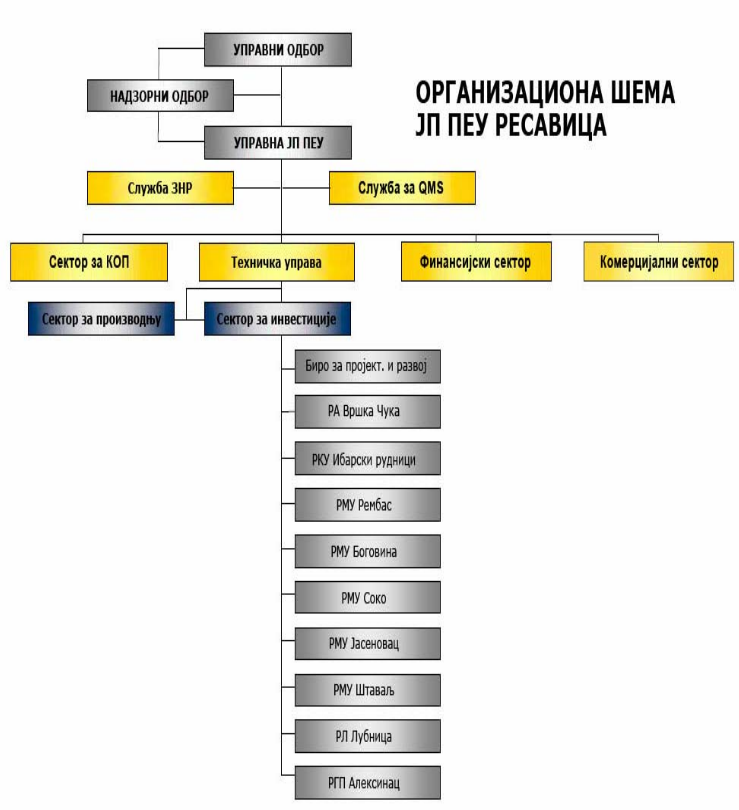

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