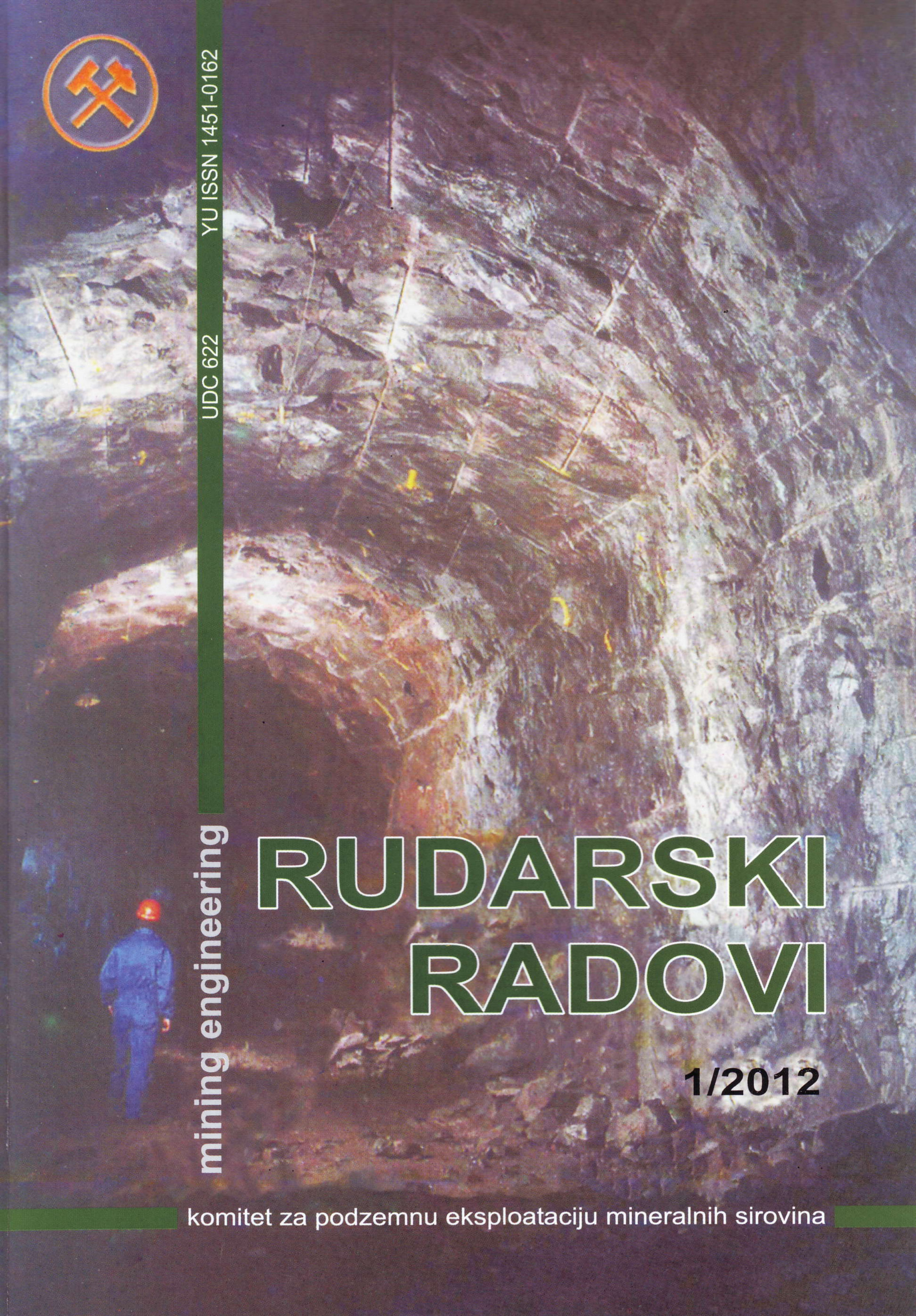

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ANALIZA SLEGANJA TEMELJA USLED PROGRESIVNOG PROVLAŽAVANJA PRAŠINASTIH GLINA[****]

Izvod

Nisu retki primeri vidljivih oštećenja (pukotine i prsline po fasadnim zidovima) na starijim građevinama Beograda koje su fundirane plitko. Najčešće je to posledica neravnomernih sleganja koja često nastaju usled nepredviđenog provlažavanja tla ispod temelja. Ovo provlažavanje je uglavnom lokalnog karaktera i u takvim uslovima je neminovna pojava diferencijalnih sleganja. U ovom radu su izloženi rezultati istraživanja koji ukazuju da pored veličine zone uticaja promene vlažnosti u tlu u horizontalnom pravcu i proračunatih vrednosti sleganja, treba anlizirati i ukupnu promenu zapremine tla ispod temelja.

Ključne reči: *provlažavanje tla, sleganje, statička penetracija, promena zapremine.*

1. UVOD

Voda predstavlja vitalnu i najaktivniju komponentu tla jer je stalno u pokretu. Njeno prisustvo u tlu zavisi od brojnih faktora, a pre svega od raspoložive količine (padavine, kvašenja, curenja drenažnih sistema, i sl.), ali i od brzine kojom voda osvaja tlo (razvijen proces evapo-transpiracije, prisustvo vegetacije, postojanja prirodnih i veštačkih drenažnih sistema i sl.). Manje promene vlažnosti događaju se tokom godine kao rezultat sezonskih promena usled jakih kiša, česte promene temperature i sl. Međutim, veći uticaj na tlo one imaju za vreme dugotrajnih padavina odnosno dugačkih sušnih perioda. Ovaj efekat klimatskih promena postaje važniji ukoliko je prisutna i vegetacija (npr. neke posebne vrste drveća dnevno mogu da iscrpe preko stotine litara vode u vrelim danima) izazivajući u određenoj meri

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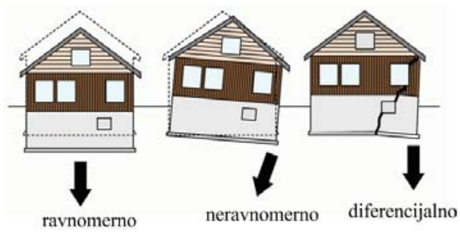
[****] *Ovaj rad je proizašao iz projekata TR 36014 "Geotehnički aspekti istraživanja i razvoja savremenih tehnologija građenja i sanacija deponija komunalnog otpada" i TR 36009 koje finansira Ministarstvo za prosvetu i nauku Republike Srbije.*

skupljanje tla, što opet može da izazove vidljive deformacije na objektu. Suprotno tome, odsustvo vegetacije vodi ka povećanju vlažnosti, ne retko i bubrenju glinenih tla, koje ponekad može izazvati izdizanje objekta. Zbog toga se na objektima mogu javiti različiti vidovi deformacija [1] (slika 1). Napominjemo da za vreme dužih sušnih perioda i korenje drveća može prouzrokovati uglavnom mehaničke štete na podzemnim delovima objekta usled njihovog rasta u dublje i vlažnije delove terena.



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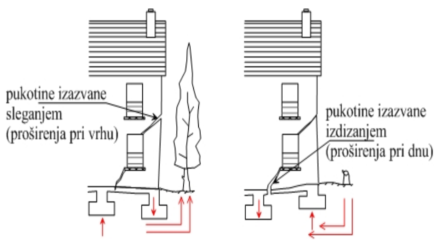


b)

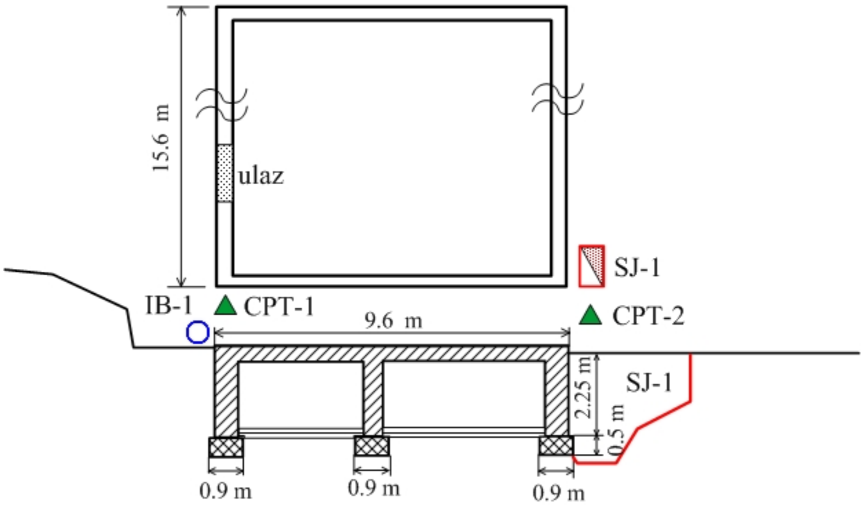
Sl. 1. a) *Opšti tipovi sleganja temelja, b) Pojava dijagonalnih pukotina u zidovima zbog diferencijalnih sleganja*

Samo kretanje vode, bilo da se obavlja pod uticajem gravitacionih odnosno negravitacionih sila (prirodnog ili antropogenog porekla), u najtešnjoj je interakciji s čvrstom komponentom, menjajući često mehanička svojstva a samim tim i fizičko stanje, pa i ponašanje. U početku su promene lagane, gotovo neprimetne, ali vremenom mogu biti i nepredvidljivih razmera dovodeći često objekat u stanje koje zahteva hitne intervencije. Zato, ako se ispolje ove promene (npr. ispod temelja objekata) stvaraju se i uslovi za pojavu diferencijalnog sleganja u temeljnoj konstrukciji, što dalje može da dovede do velikih oštećenja objekta [2], [3]. Međutim, nepoznavanje konkretnih uslova koji postoje u tlu posle provlažavanja, navodi na projektovanje i izvođenje tehnički i ekonomski neadekvatnih sanacionih mera. U tom slučaju od značaja je poznavati veličinu zone uticaja promene vlažnosti, odnosno veličinu zone u kojoj dolazi do promene fizičko-mehaničkih svojstava tla. Veličina ove zone je različita za različite vrste tla. Zato će se u ovom radu dati neki rezultati istraživanja u određivanju veličine zone uticaja promene vlažnosti u prašinastim glinama.

2. IZVEDENA ISTRAŽIVANJA

Predmet ovog rada je prizemni objeka u okolini Surčina, koji je u konstruktivnom smislu izgrađen od masivnih zidova. Osnova objekta je pravougaona, dimenzija 15,6 x 9,6 m. Zidovi su od opeke u krečnom malteru debljine d = 0,55 m. Krovna konstrukcija je drvena na dve vode sa krovnim pokrivačem od crepa. Objekat ima podrum koji je ukopan u zemlju 2,1 m, u odnosu na nultu kotu terena. Fundiran je na temeljnim trakama širine B = 0,9 m na koti fundiranja od 2,75 m. Kontaktno naprezanje ispod temeljnih traka je reda veličine Δq = 130 kN/m^2. Odvodnjavanje vode sa krova vrši se preko olučnih vertikala koje prikupljenu vodu izvode na površinu terena tako da dalji odvod vode nije regulisan. Posle dužeg korišćenja objekta (više desetina godina), došlo je do ozbiljnih oštećenja na mestima olučnih vertikala u vidu vrlo progresivnih i razvijenih pukotina koje ugrožavaju njegovu dalju eksploataciju [4].

U cilju odredjivanja fizičko-mehaničkih karakteristika kao i veličine zone uticaja promene vlažnosti u tlu, izvršena su određena geomehanička istraživanja na mestima olučnih vertikala gde su oštećenja i najveća tj. na mestima gde je najverovatnije došlo do provlažavanja tla. Sprovedena geomehanička istraživanja su obuhvatala iskop jedne istražne jame (SJ-1) u zoni najvećih oštećenja tj. pretpostavljenoj zoni provlažavanja, izvođenja jedne istražne bušotine (IB-1) van zone provlažavanja, dve statičke penetracije (CPT-1 i CPT-2) i geomehanička laboratorijska ispitivanja. Situacija objekta sa položajem istražnih radova prikazana je na slici 2.

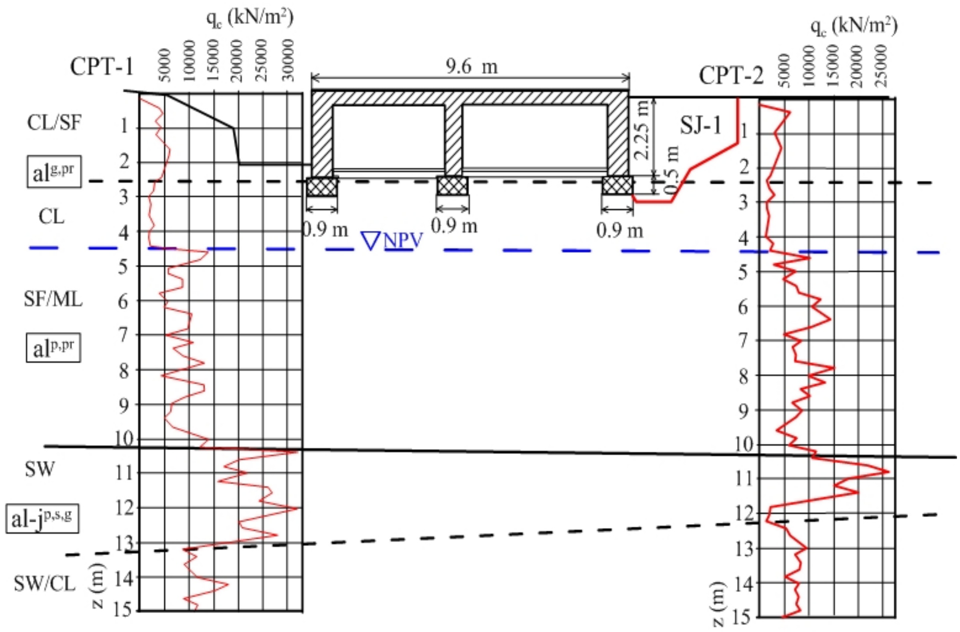


Sl. 2. *Prikaz načina fundiranja objekta sa položajem istražnih radova*

Istražna jama (SJ-1) je ručno kopana do dubine od 3,5 m tj. oko 0,7 m ispod kote fundiranja obuhvatajući tako i zonu provlažavanja. Nakon iskopa istražne jame izvršena je provera dimenzija temelja, inženjerskogeološko kartiranje tla i uzimanje 5 neporemećenih uzoraka za geomehanička laboratorijska ispitivanja. Pored istražne jame, urađena je i jedna istražna bušotina (IB-1) i dva opita statičke penetracije (CPT-1 i CPT-2). Dubina sondiranja terena je bila 15 m. Prva statička penetracija (CPT-1) je izvedena pored istražne bušotine IB-1 u zoni koja nije bila zahvaćena provlažavanjem. Druga statička penetracija (CPT-2) iz tehničkih razloga nije izvedena u samoj zoni provlažavanja, tj. na mestu istražne jame (SJ-1), već na oko 2.0 m od objekta. U cilju utvrđivanja fizičkomehaničkih svojstava tla izvršeni su identifikaciono-klasifikacioni opiti, opiti direk-tnog smicanja i edometarski opiti stišljivosti.

3. REZULTATI ISTRAŽIVANJA

Rezultati statičko penetracionog sondiranja, inženjerskogeološko kartiranje istražne bušotine i istražne jame, pokazali su da je teren izgrađen od sledećih litoloških članova (slika 3):



$al^{g,pr}$		$al^{p,pr}$	$al\text{-}j^{p,s,g}$	
CL/SF	CL	SF/ML	SW	SW/CL
γ = 19 kN/m³	γ = 19 kN/m³	γ' = 11 kN/m³	γ' = 11 kN/m³	γ' = 11 kN/m³
q_c = 3000 - 4000 kN/m²	q_c = 1500 - 2000 kN/m²	q_c = 6000 - 7000 kN/m²	q_c = 15000 - 22000 kN/m²	q_c = 6000 - 10000 kN/m²
M_v = 11000 kN/m²	M_v = 4450 kN/m²	M_v = 15500 kN/m²	M_v = 50000 kN/m²	M_v = 27000 kN/m²

Sl. 3. *Geotehnički presek terena*

- Prašinaste gline, prašine i prašinast pesak ($al^{g,pr}$), smeđe i sive boje, sa značajnim udelom organskih primesa i pojavom tankih proslojaka peska; srednje stišljive; srednje i slabije vodopropusne. U istražnoj jami (SJ-1) od dubine 2,0 m pa nadalje u zoni fundiranja, utvrđena je zona provlažavanja sa izrazito vlažnim, mekanim, i jače stišljivim prašinastim glinama.
- Pesak prašinast ($al^{p,pr}$), finozrn do krupnozrn sive i smeđe boje sa neujednačenim udelom sitnozrne frakcije i karakterističnom pojavom proslojaka mulja sa dosta organskog detritusa. Postoji nanos sa izraženom finom stratifikacijom materijala, srednjeg stepena zbijenosti sa karakterističnim i učestalim gradacionim prelazima ka šljunku.
- Srednjezrn i krupnozrn pesak, šljunak i prašinasta glina ($al\text{-}j^{p,š,g}$) – rečno jezerski sedimenti (sa Corbicula flumi-nalis) međusobno izpreplitani slabo sortirani sa čestim lateralnim gradacionim prelazima, a izrazito heterogeni po parametru otporno-deformabilnih i filtracionih karakteristika.

Kako je terenskim istraživanjima pouzdano utvrđeno da je oštećenje objekta nastalo usled provlažavanja sloja prašinastih glina i prašina, to se u tabeli 1 daju zbirni rezultati laboratorijskih ispitivanja samo za ovu sredinu [5].

Tabela 1. *Rezultati identifikaciono-klasifikacionih i deformabilno-otpornih karakteristika prašinastih glina*

Istražni rad	**Prirodna vlažnost**	**Plastičnost i konzistencija**				**(USCS)**
	w **(%)**	w_L **(%)**	w_P **(%)**	I_p	I_c	
SJ-1	27.4 – 34.6	34.8-40.5	21.8-22.7	13.0-17.8	0.12-0.49	CL, CL/SF
IB-1	22.1 – 26.8	30.0-38.0	20.0-22.0	13.0-17.8	0.58-0.59	
Istražni rad	**Zapr. težina**	**Modul stišljivosti Mv (kN/m²)**			**Čvrstoća smicanja**	
	γ_d **(kN/m³)**	**50-100**	**100-200**	**200-400**	$\varphi'(^0)$	**c' (kPa)**
SJ-1	13.5-14.6	1850-2120	2810-3110	4150-6020	28	5
IB-1	15.1-16.0	3450-4450	4500-6250	5500-11000	19-22	15-40

Upoređujući rezultate fizičko-mehaničkih svojstava tla pre i posle provlažavanja (SJ-1 sa IB-1), vidi se da su vrednosti svih fizičko-mehaničkih svojstava tla znatno smanjene. Ovo je potvrđeno i na osnovu rezultata kartiranja istražne jame (SJ-1) jer je pouzdano utvrđeno da je temeljno podtlo provlaženo. Naime, prirodna vlažnost (w) i indeks konsistencije (I_c), pokazuju da se sloj provlaženih prašinastih glina ($al^{g,pr}$) nalazi u vrlo mekom konzistentnom stanju:

$$w_p\,(21.8 - 22.7\ \%) < w\,(27.4 - 34.6\ \%) < w_l\,(34.8 - 40.5\ \%)$$

$$0.12 < Ic < 0.49$$

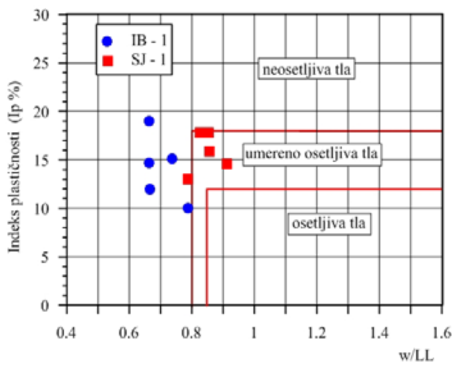
Saglasno tome i vrednosti zapreminske težine u suvom stanju (γ_d), efektivne kohezije (c') i modula stišljivosti (Mv), su izrazito niske:

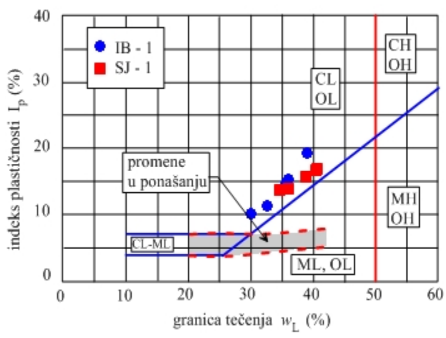
$$\gamma_d = 13.50\ kN/m^3$$

$$c' = 5.0\ kN/m^2$$

$$Mv = 1850\ kN/m^2$$

Pored toga treba napomenuti da dobijene vrednosti granica konzistencije i odgovarajućih indeksnih pokazatelja, ukazuju da se prašinaste gline u provlaženoj zoni, mogu prema kriterijumu Braya i Sancia [6], [7], svrstati umereno osetljiva tla u pogledu opasnosti od pojave likvefakcije koji se odnosi na sitnozrne sredine.





Sl. 4. *Analiza procene opasnosti od pojave likvefakcije prema kriterijumima a) Braya i Sancia (2006) i b) Boulangera i Idrissa (2006)*

Dobijeni rezultati poslužili su za formiranje geotehničkog modela terena (GMT-1), koji reprezentuje temeljno podtlo dela objekta na kome je došlo do oštećenja. I po rezultatima opita statičke penetracije (CPT-1 i CPT-2), može se videti da postoje izvesna odstupanja u pogledu vrednosti q_c, a s obzirom da se ova istraživanja izvode direktno na terenu, može se reći da ona reprezentuju prirodne uslove koji vladaju u podtemeljnom tlu do dubine od 15 m. Rezultati dobijeni iz CPT-1 opita, iskorišćeni su za formiranje geotehničkog modela terena (GMT-2) koji reprezentuje zonu objekta u kojoj nisu vidljiva oštećenja. Merodavne vrednosti fizičko-mehaničkih parametara tla za analizirane geotehničke modele terena, prikazane su u tabeli 2.

Za ovako usvojene geotehničke modele terena izvršen je proračun dozvoljene nosivosti tla i sleganja temelja. Proračun dozvoljene nosivosti tla je izveden na osnovu "Pravilnika o tehničkim normativima za temeljnje građevinskih objekata", a prema modelu GMT-1 koji karakteriše provlaženu zonu tla. Rezultat proračuna je pokazao da je dozvoljena nosivost tla veća od stvarnog opterećenja jer iznosi:

$$Q_a = 164 \text{ kN/m}^2 > \Delta q = 130 \text{ kN/m}^2$$

Tabela 2. *Usvojeni geotehnički modeli terena GMT-1 i GTM-2*

GTM-1					
Litološki član	h (m)	φ' (^{0})	c' (kN/m^2)	Mv (kN/m^2)	γ (kN/m^3)
$al^{g,pr}$	2.75	27	5	2000	19
GTM-2					
Litološki član	**h (m)**	**d (m)**	**q_c (kN/m^2)**	**Mv (kN/m^2)**	**γ, γ' (kN/m^3)**
$al^{g,pr}$	2,5	2,5	3000-4000	11000	19
	4,5	2,0	1500-2000	4450	19
$al^{p,pr}$	10,0	5,5	6000-7000	15500	11
$al\text{-}j^{p,š,g}$	12,5	2,5	15000-22000	50000	11
	/	/	6000-10000	27000	11

Pošto ovaj rezultat nije pokazao da je došlo do sloma tla, izvršen je i proračun sleganja. Proračun sleganja je sproveden za oba geotehnička modela tla uključujući tako i zonu u kojoj je došlo do provlažavanja. Analiza sleganja je izvršena za karakterističnu tačku temelja, a za geotehnički model terena GMT-1 tj. za provlaženu zonu i oštećeni deo objekta, dobijene su računske vrednosti sleganja od:

$$\text{(GTM-1)}\ \rho_1 = 3{,}8 \text{ cm}$$

Za geotehnički model terena (GMT-2), tj. za uslove koji vladaju u delu objekta gde nije došlo do oštećenja, ili bolje reći za uslove koji su vladali u temeljnom tlu nakon izgradnje objekta, računske vrednosti sleganja su reda veličine:

$$\text{(GTM-2)}\ \rho_2 = 1{,}8 \text{ cm}$$

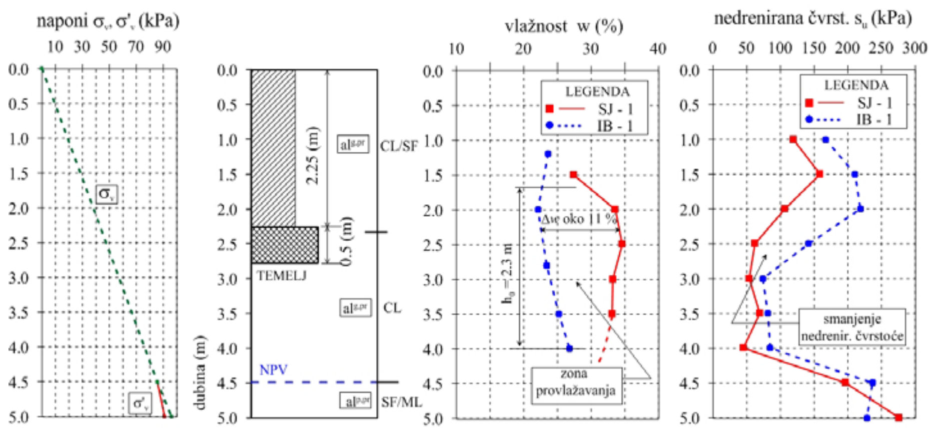
Ovi rezultati pokazuju da su sračunate vrednosti sleganja u dozvoljenim granicama ali da je naknadno sleganje usled provlažavanja izazvalo i pojavu diferencijalnog sleganja.

$$\Delta\rho = \rho_1 - \rho_2 = 2{,}0 \text{ cm}$$

4. DISKUSIJA

Rezultati veličina otpora konusa statičkih penetracija CPT-1 i CPT-2, ne pokazuju bitno odstupanje, što navodi na zaključak da je tlo u zoni CPT-2 u sličnom stanju kao tlo u zoni IB-1, tj. da u zoni CPT-2 nije došlo do provlažavanja tla. Na osnovu ovoga može se zaključiti da veličina zone uticaja promene vlažnosti u tlu u horizontalnom pravcu nije velika i da je ona za prašinaste gline ($al^{g,pr}$) manja od 2,0 m. Takođe, i razlika u računskim sleganjima temelja se javila na relativno malom rastojanju do 2,0 m što može da bude jedan od bitnih uzroka pojave diferencijalnog sleganja.

Međutim, pretpostavka je da razlika u sleganju od 2 cm, nije preterano velika da bi se sa sigurnošću moglo zaključiti da je to i jedini uzrok nastanka pukotina i oštećenja na objektu usled lokalnog provlažavanja tla. Zato se mora potražiti još neki od eventualnih uzroka oštećenja. Ukoliko se analiziraju rezultati promene vlažnosti tla sa dubinom, videlo bi se da je u zoni provlažavanja, vlažnost povećana u proseku za oko w = 11 %. Ovi rezultati promene vlažnosti tla sa dubinom, za oba geotehnička modela, prikazani su na slici 5 u vidu dijagrama. Iako je istražna jama (SJ-1) izvedena do dubine od 3,5 m, može se sa dijagrama zapaziti da se povećanje vlažnosti javlja i u vertikalnom pravcu za neku određenu dubinu h_0. U konkretnom slučaju prosečna vertikalna dubina promene vlažnosti iznosi oko h_0 = 2,3 m (od recimo 1,7 m do 4,0 m). Takođe se može uzeti u obzir i činjenica da je u zoni provlažavanja tla stepen zasićenja S_r = 100 %. Nedre-nirana čvrstoća smicanja određena je na osnovu rezultata opita statičke penetracije, korišćenjem teorijske zavisnosti u obliku $q_c = N_k\, s_u + \sigma_v$ (za N_k usvojena maksimalna vrednost od 25).



Sl. 5. *Primena fizičko-mehaničkih karakteristika u funkciji dubini*

Pošto ukupna zapremina tla (V) zavisi i od zapremine čvrstih čestica (V_s), možemo da definišemo specifičnu zapreminu v kao odnos V/V_s, odnosno

$$V = v \cdot V_S \quad \text{.......................................(1)}$$

Ovo se može napisati i kao:

$$v = \frac{V}{V_S} = \frac{V_S + V_p}{V_S} = 1 + e \quad \text{...........(2)}$$

Pošto se zapremina čvrstih čestica ne menja, to promena ukupne zapremine (ΔV) zavisi od promene specifične zapremine (Δ*v*), pa se može napisati:

$$\Delta V = \Delta v \cdot V_S \quad \Rightarrow \quad \Delta v = \frac{\Delta V}{V_S} \text{ ....(3)}$$

Kako se radi o zasićenom tlu sa stepenom zasićenja $S_r = 1$, onda je na osnovu (2):

$$S_r = \frac{w \cdot G_s}{e} = \frac{w \cdot G_s}{v - 1} \text{ ....(4)}$$

odnosno za $S_r = 1$

$$v = 1 + w \cdot G_s \text{ ....(5)}$$

Iz ovog sledi da promena specifične zapremine zavisi od promene vlažnosti tj.:

$$\Delta v = \Delta w \cdot G_s \text{ ....(6)}$$

Zamenom ove vrednosti u jednačini (3) dobija se promena ukupne zapremine:

$$\Delta V = V_s \cdot \Delta w \cdot G_s \text{ ....(7)}$$

Veza jednačina (1) i (5) definiše ukupnu zapreminu tj.:

$$V = (1 + w \cdot G_s) \cdot V_s \text{ ....(8)}$$

Ukoliko sada posmatramo tzv. blok tla jedinične površine ali do dubine h_0, ukupna zapremina je:

$$V = 1 \cdot h_0 = (1 + w \cdot G_s) \cdot V_s \text{ ....(9)}$$

ili

$$V_s = \frac{h_0}{(1 + w \cdot G_s)} \text{ ....(10)}$$

Zamenom ove vrednosti u (7) dobija se promena zapremine jediničnog bloka tla o dubine u kojoj je izražena promena vlažnosti:

$$\Delta V = \frac{h_0 \cdot \Delta w \cdot G_s}{1 + w \cdot G_s} \text{ ....(11)}$$

Ovako izračunata promena zapremine u vertikalnom pravcu u neku ruku predstavlja vertikalno izdizanje ρ (za slučaj smanjenja vlažnosti javilo bi se dopunsko sleganje). Međutim, kada se radi o glinenom tlu, poznato je da ono usled povećanja vlažnosti povećava i zapreminu (bubri). Zbog toga na ovako izračunatu prosečnu promenu zapremine u vertikalnom pravcu treba uzeti u obzir i bočno širenjene (za slučaj gubljenja vode, bočno skupljanje) pa je:

ΔV = ρ + *bočno širenje...............(12)*

što navodi na zaključak da je:

$$\rho < \Delta V$$

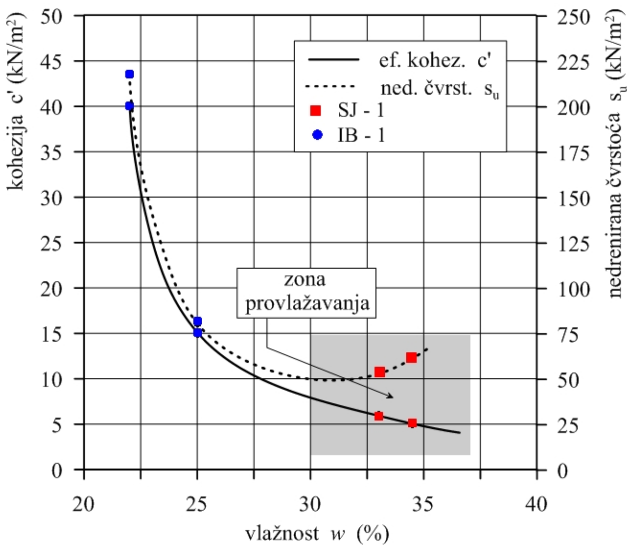
Na osnovu empirijskih rezultata, Driscoll [8] je predložio da se za bubrenje/sleganje usvoji redukovana vrednost promene zapremine od:

$$\rho \leq \frac{\Delta V}{3} \quad do \quad \frac{\Delta V}{4}$$

Primenjujući ovo rešenje, dobijena je promena jedinične zapremine od ΔV = 35.7 cm, odnosno:

$\rho = 8.9 – 11.9\ cm$

Kako je zidana konstrukcija od opeke u krečnom malteru i stara više desetina godina, može se zaključiti da se konsolidaciono sleganje završilo, pa ovako dobijene vrednosti, predstavljaju dopunsko izdizanje usled intenzivnog provlažavanja. Ono se desilo na kratkom rastojanju, pa objekat verovatno nije mogao da prihvati ovolike deformacije. Na to ukazuju i pukotine čija je širina u prizemnom delu veća. U svakom slučaju, glavni uzrok oštećenja objekta je svakako provlažavanje temeljnog tla, i to najvećim delom usled oštećenja olučnih sistema. Međutim, kada se radi o prašinastim materijalima sa provla-žavanjem je vrlo moguće i ispiranje finih čestica, što takodje dovodi do promene zapremine [9]. Ovaj proces promene fizičko-mehaničkih karakteristika tla usled provlažavanja, šematski je ilustrovan na slici 6.



Sl. 6. *Smanjenje čvrstoće smicanja tla ispod objekta usled provlažavanja*

5. ZAKLJUČAK

Na osnovu sprovedenih istraživanja i analiza može se zaključiti da se neplanirano provlažavanje tla dešava u određenoj zoni uticaja promene vlažnosti koja je recimo u horizontalnom pravcu za slučaj ispitivanih prašinastih glina ($al^{g,pr}$) relativno mala, L < 2.0 m. Kako je provlažavanje tla uglavnom lokalnog karaktera, to je i zona naknadnog sleganja lokalnog nivoa. Posledica ovoga je pojava diferencijalnih sleganja na vrlo kratkom rastojanju, što može da izazove manja ili veća oštećenja na objektu. Da bi se objekat vratio u eksploataciono stanje potrebno je izvršiti njegovu sanaciju koja mora obuhvatiti sanaciju temelja, sanaciju sistema za kontrolisano odvođenje površinskih i atmosferskih voda i sanaciju same konstrukcije. Sve ove mere sanacije zahtevaju znatno veća finansijska sredstva od onih koja su bila potrebna za izgradnju održavanje sistema koji treba da spreči provlažavanje temeljnog tla.

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ANALYSIS OF FOUNDATION SETTLEMENT FROM PROGRESSIVE MOISTENING OF SILTY CLAY****

Abstract

Examples of visible damage (cracks on the facade walls) on the shallow founded older buildings in Belgrade are not rare. Usually, this is a consequence of unequal settlement resulting from the unexpected moistening of soil under foundation. Moistening usually occurs locally and differential settlement is inevitable in such conditions. This paper presents the research results, which indicate that besides the size of impact zone of moisture change in the soil in horizontal direction and calculated settlement values, total change of soil volume under foundations should be analyzed.

Key words: *soil moistening, settlement, static penetration, volume change*

1. INTRODUCTION

Water is vital and the most active component of the soil because it is constantly in movement. Its presence in the soil depends on many factors, primarily on the available quantity (rainfall, wetting, leaking of drainage systems, etc.) and the rate of water penetration into ground (developed process of evapotranspiration, the presence of vegetation, the existence of natural and artificial drainage systems etc.). Less moisture changes occur during the year as the result of seasonal changes due to heavy rains, frequent temperature changes and similar. However, it has more influence on the ground during prolonged rainfall and long dry periods. The effect of climate changes becomes more important if the vegetation is present (eg. a particular tree species can exhaust per day over hundreds of liters of water on hot days), causing in a certain

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degree the soil shrinkage, which in turn can cause visible deformation on a facility. Contrary to this, the absence of vegetation leads to the increased humidity, often swelling of clay soils, which can sometimes cause elevation of the building. Therefore, different types of deformations may appear on building [1] (Figure 1). Please note that during prolonged dry periods, the tree roots can mainly cause mechanical damages of the underground parts of a facility due to their growth in deeper and wetter parts of terrain.

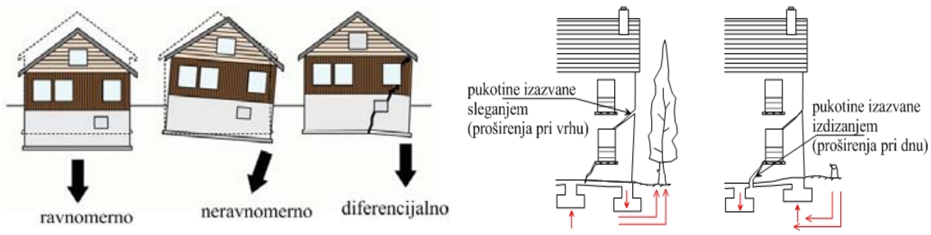

Fig. 1. *a) The general types of soil settlement, b) appearance of diagonal cracks in the walls due to differential settling*

The water movement, whether it is done under the influence of gravitational forces or non-gravitational forces (natural or anthropogenic), is in the closest interaction with the solid component, often changing the mechanical properties and therefore the physical condition, and behavior. At the beginning, the changes are light, almost imperceptible, but over time they can be with unpredictable scales leading often the facility in a condition that requires emergency intervention. Therefore, if these changes are expressed (e.g. under the foundations of buildings), the conditions are created for the occurrence of differential settlement in the basic structure, which can still cause great damages to the facility [2], [3]. However, the lack of specific conditions that exist in the soil after moistening leads to a design and implementation of technically and economically inadequate rehabilitation measures. In this case, it is important to know the size of influence zone of moisture changes, or the zone size in which there is a change of physical-mechanical properties of soil. The size of this zone is different for different types of soil. Therefore, this paper will present some research in determining the size of influence zone of moisture changes in silty clay.

2. REALIZED INVESTIGATIONS

The subject of this work is a ground-floor building near Surčin, which was built of massive walls in the constructive sense. The base of building is rectangular, size 15.6 x 9.6 m. The walls are of brick in a lime mortar, thickness d = 0.55 m. The roof structure is timber on two water with tile roof. The building has a basement that was dug into the ground 2.1 m, compared to zero above ground level. It was founded on the fundamental bands, width B = 0.9 m at funding height of 2.75 m. Contact stresses below the fundamental bands is $\Delta q = 130\ kN/m^2$. Water drainage from the roof is done through gutters which run out the collected water on the surface so that further water drainage is not regulated. After long use of building (several decades), there were serious defects in places of vertical gutters in the form of progressive and developed cracks that threat its future exploitation [4].

In order to determine the physical and mechanical characteristics as well as the size of influence zone of moisture changes in the soil, the certain geomechanical testing was carried out in the places of gutter verticals where the damages are the greatest, i.e. the places with possible moistening of soil.

Conducted geotechnical investigations included the excavation of an exploration pit (NS-1) in the zone of greatest damage, i.e. assumed moistening zone, carrying out one exploration hole (IB-1) outside the zone of moistening, two static penetrations (CPT-1 and CPT-2) and geomechanical laboratory testing. The situation facilities with the exploration works is shown in Figure 2

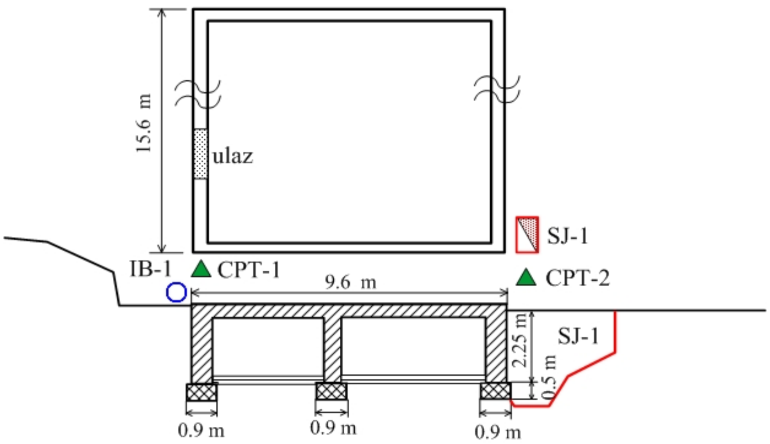

Fig. 2. *Review of facility founding with a position of exploration works*

Exploration (NS-1) was manually excavated to a depth of 3.5 m, i.e. about 0.7 m below the level of funding including the zone moistening After excavation the exploration pit, a check of foundation dimensions was carried out, then engineering mapping of soil and taking 5 undisturbed samples for geomechanical laboratory testing. In addition to the exploration pit, one exploration drill hole (IB-1) was made and two static penetration tests (CPT-1 and CPT-2).

Sounding depth of a field was 15 m.

The first static penetration (CPT-1) is derived next to the exploration drill hole in the IB-1 zone, which was not affected by moistening. The second static penetration (CPT-2), for technical reasons, was not performed inside the moistening zone, i.e. the place of exploration pit (NS-1), but at about 2.0 m from the facility. In order to determine the physic-mechanical properties of soil properties, the identification-classification experiments were carried out, the experiments with direct shear and oedometric compressibility experiments.

3. INVESTIGATION RESULTS

The results of static penetration sounding, engineering-geological mapping of exploration drill hole and exploration pit have shown that the ground is made of the following lithological members (Figure 3):

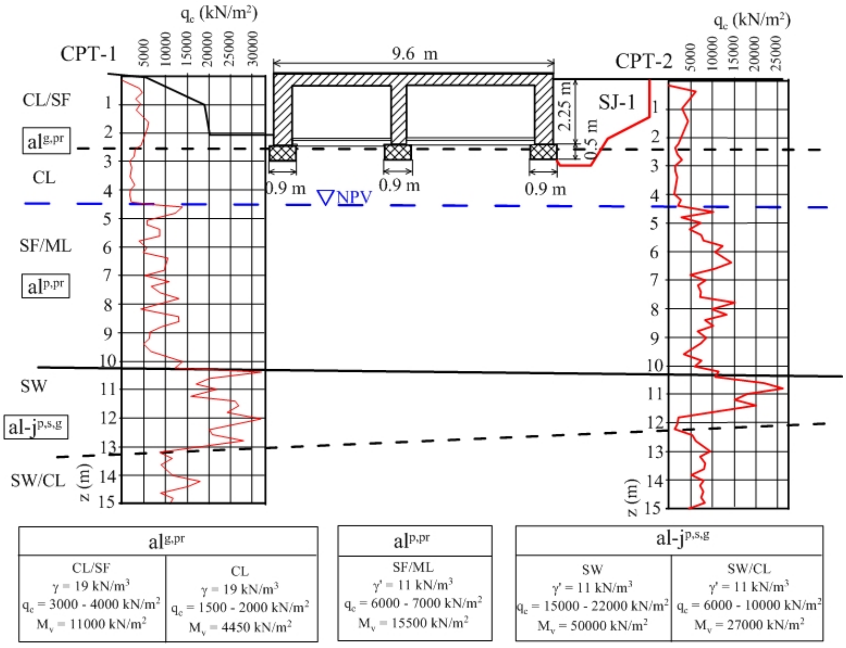

Fig. 3. *Geotechnical section of the ground*

- Silty clay, dust and dusty sand ($al^{g,pr}$), brown and gray in color, with significant organic impurities and occurrence of thin interlayers of sand; medium compressible; medium and poor water permeable. In the exploration pit (NS-1) from 2.0 m depth and further in the funding zone, the zone of moistening was found with distinctly very wet, soft, and more compressible silty clay.- Dusty sand ($al^{p,pr}$), fine grain to coarse grain and brown in color with an uneven share of fine-grained fraction and characteristic appearance of interlayers of mud with lot of organic detritus. There is a deposit with the expressed fine material stratification, medium degree of compaction with characteristic and frequent gradual transitions to gravel.
- Medium grain and coarse grain sand, gravel and silty clay ($al\text{-}j^{p,\check{s},g}$) – river-lake sediments (with Corbicula fluminalis) interlaced, poorly sorted with often gradual lateral transitions and expressly heterogeneous per parameter of resistant-deformable and filtration properties.

As the field explorations have reliably established that the damage of facilities was caused by moistening the layer of silty clay and dust, Table 1 gives the summary results of laboratory tests only for this environment [5].

Table1. *Results of identification-classificationand deformation-resistant properties of silty clay*

Exploration work	Natural moisture	Plasticity and consistency				(USCS)
	w (%)	w_L (%)	w_P (%)	I_p	I_c	
NS-1	27.4 – 34.6	34.8-40.5	21.8-22.7	13.0-17.8	0.12-0.49	CL, CL/SF
IB-1	22.1 – 26.8	30.0-38.0	20.0-22.0	13.0-17.8	0.58-0.59	
Exploration work	Volumetric weight	Compressibility module Mv (kN/m^2)			Shear strength	
	γ_d (kN/m^3)	50-100	100-200	200-400	w_P'(^{0})	c' (kPa)
NS-1	13.5-14.6	1850-2120	2810-3110	4150-6020	28	5
IB-1	15.1-16.0	3450-4450	4500-6250	5500-11000	19-22	15-40

Comparing the results of physico-mechanical properties of the soil before and after moistening (NS-1 to IB-1), it is seen that the values of all physico-mechanical properties of the soil are significantly reduced. This was also confirmed on the basis of results of mapping the exploration pit (NS-1) because it is reliably established that the basic subsoil is moistened. Namely, the natural moisture (w) and consistency index (I_c) show that the layer of moistened silty clay ($al^{g,pr}$) is in a very soft consistent state:

$$w_p \text{ (21.8 - 22.7 \%)} < w \text{ (27.4 - 34.6 \%)} < w_l \text{ (34.8 - 40.5 \%)}$$

$$0.12 < Ic < 0.49$$

According to this, the value of gravity in a dry state (γ_d), effective cohesion (c') and modulus of compressibility (Mv), are extremely low:

$$\gamma_d = 13.50 \text{ kN/m}^3$$
$$c' = 5.0 \text{ kN/m}^2$$
$$M_v = 1850 \text{ kN/m}^2$$

In addition, it should be noted that the obtained values of consistency limits and corresponding index indicators, suggest that silty clay in the zone, according to the criteria of Bray and Sancio[6], [7], can be classified into moderately sensitive soils, in terms of danger of liquefaction occurrence, related the fine-grained environments.

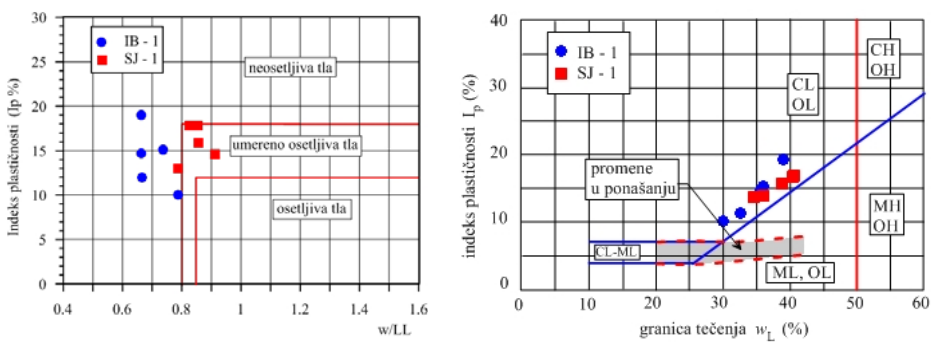

Fig. 4. *Analysis of risk assesment from liquefactionoaccurence according to the criteria a) Bray and Sancio (2006) and b) Boulanger and Idriss (2006)*

The obtained results were used to form a geotechnical model of field (GMT-1), which represents a fundamental subsoil of a facility part which was damaged. And the results of static penetration tests (CPT-1 and CPT-2) can show that there are some discrepancies regarding the q_c value of qc, and considering that these explorations are carried out directly in the field, it could be said that they represent the natural conditions prevailing in the sub-base soil to a depth of 15 m. The obtained results from CPT-1 experiment were used for formation the geotechnical field model (GMT-2) which represents the zone of facility in which the damages are not visible. Relevant values of physic-mechanical soil parameters for analyzed geotechnical field models are shown in Table 2.

For such adopted geotechnical field models, the calculation of allowable bearing capacity of soil and foundation settlement was done. Calculation of allowable bearing capacity is derived based on the "Rules of technical standards for construction facilities with foundation," and according to the model GMT-1, which characterizes the moistening zone of soil. The result of calculation showed that permitted bearing capacity of soil is greater than the actual load as follows:

$$Q_a = 164 \text{ kN/m}^2 > \Delta q = 130 \text{ kN/m}^2$$

Table 2. *The adopted geotechnical field models GMT-1 and GTM-2*

GTM-1					
Lithological member	**h (m)**	**φ' (^{0})**	**c' (kN/m^2)**	**Mv (kN/m^2)**	**(kN/m^3)**
$al^{g,pr}$	2.75	27	5	2000	19
GTM-2					
Lithological member	**h (m)**	**d (m)**	**q_c (kN/m^2)**	**Mv (kN/m^2)**	**γ, γ' (kN/m^3)**
$al^{g,pr}$	2.5	2.5	3000-4000	11000	19
	4.5	2.0	1500-2000	4450	19
$al^{p,pr}$	10.0	5.5	6000-7000	15500	11
$al\text{-}j^{p,š,g}$	12.5	2.5	15000-22000	50000	11
	/	/	6000-10000	27000	11

Since this result has not shown that there was a breakdown of the soil, a calculation of settlement was made. Calculation of settlement was carried out for both geotechnical models of soil including the zone of moistening. Analysis of settlement was carried out for the specific point of foundation, and the calculating value of settlement were obtained for a geotechnical field model GMT-1, i.e. for moistening zone and damaged part of the facility, as follows:

$$(\text{GTM-1})\ \rho_1 = 3{,}8 \text{ cm}$$

For geotechnical field model (GMT-2), i.e. prevailing conditions in a part of the building where no damage has occurred, or rather the conditions that prevailed in the underlying soil after construction of the facility, the calculating values of settlement are as follows:

$$(\text{GTM-2})\ \rho_2 = 1{,}8 \text{ cm}$$

These results show that the calculated values of settlement are within allowable limits, but that subsequent settlement, due to the moistening, has caused the occurrence of differential settlement.

$$\Delta\rho = \rho_1 - \rho_2 = 2{,}0 \text{ cm}$$

4. DISCUSSION

The results of cone resistance values of static penetrations CPT-1 and CPT-2, show no significant deviation, which suggests that the soil in zone of CPT-2 is in a similar state as the soil in zone IB-1, i.e. there was no moistening of soil in the CPT-2 zone. Based on this, it can be concluded that the size of the zone of influence of soil moisture changes in the horizontal direction is not great and that the influence zone size moisture changes in a horizontal direction is not large and it is less than 2.0 m for silty clay ($al^{g,pr}$). Also, the difference in computational soil settlements has occurred on a relatively small distance of 2.0 m, what can be one of important causes for differential settlement occurrence.

However, it is assumed that the difference in settlement of 2 cm is not too big to be concluded with certainty that this is the only cause of cracks and damages on a facility due to local soil moistening.

Therefore, some other possible causes of damage have to be found out. If the results of soil moisture changes with depth are analyzed, it will be seen that, in the moistening zone, the moisture is increased of about w = 11%. These results of soil moisture changes with depth, for both geotechnical models, are presented in Figure 5 in the form of diagrams. Although the exploration pit (NS-1) was made to a depth of 3.5 m, it can be observed from a diagram that increased soil moisture also appears in the vertical direction to the certain depth h_0. In this case, the average vertical depth of moisture changes is about h_0 = 2.3 m (from e.g. 1.7 m to 4.0 m). It can also be taken into account the fact that the zone of soil moistening the degree of saturation is S_r = 100%. The nondrained shear strength was determined on the basis of the results of static penetration tests, using the theoretical dependence in the form $q_c = N_k\, s_u + \sigma_v$ (for N_k the adopted maximum value of 25).

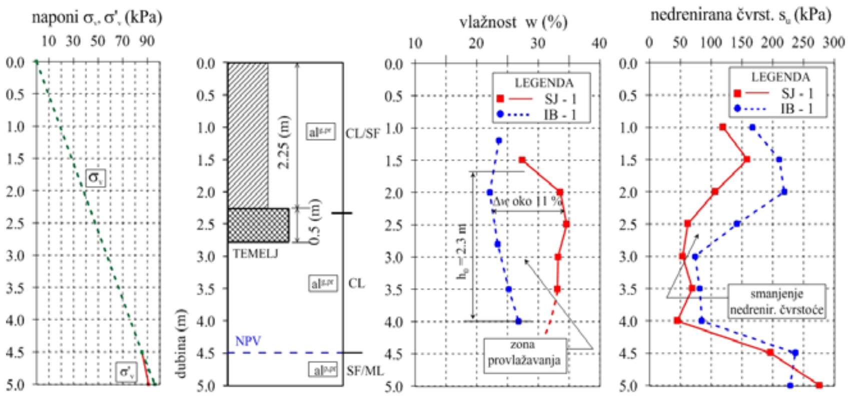

Fig.5. *The use of physico-mechanical properties in the function of depth*

Since total volume of soil (V) depends on the volume of solids (V_s), the specific volume v can be defined as the ratio V/V_s, that is:

$$V = v \cdot V_s \quad \text{....................................(1)}$$

This can be written as

$$v = \frac{V}{V_s} = \frac{V_S + V_P}{V_S} = 1 + e \text{ ....(2)}$$

Since the volume of solid particles does not change, so the change of total volume (ΔV) depends on the specific volume change (Δ*v*), and it can be written

$$\Delta V = \Delta v \cdot V_S \quad \Rightarrow \quad \Delta v = \frac{\Delta V}{V_S} \text{ ......(3)}$$

Since this is a saturated soil with saturation degree $S_r = 1$, then based on (2)

$$S_r = \frac{w \cdot G_s}{e} = \frac{w \cdot G_s}{v - 1} \text{ ....(4)}$$

that is for $S_r = 1$

$$v = 1 + w \cdot G_s \text{ ....(5)}$$

From this it follows that the specific volume change depends on humidity change, i.e.

$$\Delta v = \Delta w \cdot G_s \text{ ....(6)}$$

Substitution of this value in equation (3) gives total volume change

$$\Delta V = V_s \cdot \Delta w \cdot G_s \text{ ....(7)}$$

The connection of equations (1) and (5) defines total volume, i.e.

$$V = (1 + w \cdot G_s) \cdot V_s \text{ ....(8)}$$

If so called block of the unit surface is observed, but to a depth h_0, total volume is:

$$V = 1 \cdot h_0 = (1 + w \cdot G_s) \cdot V_s \text{ ....(9)}$$

or

$$V_s = \frac{h_0}{(1 + w \cdot G_s)} \text{ ....(10)}$$

Substitution of this value in equation (7) gives the volume change of unit soil block to a depth where the change of humidity is expressed:

$$\Delta V = \frac{h_0 \cdot \Delta w \cdot G_s}{1 + w \cdot G_s} \text{ ....(11)}$$

Thus the calculated volume changes in the vertical direction in a way represents a vertical rise ρ (in the case of moisture reduction of moisture, the additional settlement will occure). However, when it is a clay soil, it is known that, due to the increased moisture, it also increases the volume (swelling). Because of this, the lateral spreading (in the sace of water loss, lateral shrinkage) should be taken into account on such calculated average volume change in the vertical direction, so

$$\Delta V = \rho + \textit{lateral spreading} \text{.........(12)}$$

Suggesting the conclusion that

$$\rho < \Delta V$$

Based on empirical results, Driscoll [8] suggested that the reduced value of volume change has to be adopted for swelling/settlement, as follows

$$\rho \leq \frac{\Delta V}{3} \quad do \quad \frac{\Delta V}{4}$$

Applying this solution, the ochange of unit volune was obtained of $\Delta V = 35.7$ cm, that is

$\rho = 8.9 - 11.9\ cm$

As the construction was built of a brick in lime mortar, and it is several decades old, it can be concluded that the consolidated settlement ended, and thus obtained values represent additional uplift due to intense moistening. It happened in a short distance, so the facility could not possibly accept such deformations. The cracks point out to it with a width in the ground floor is higher.In any case, the main cause of damage of the facility is certainly moistening the foundation soil, and mostly due to a damage of gutter systems. However, when it is a fact with dusty materials with moistening, the washing of very fine particles is possible, which also leads to the volume change [9]. This process of changes the physico-mechanical characteristics of the soil due to moistening, is schematically illustrated in Figure 6.

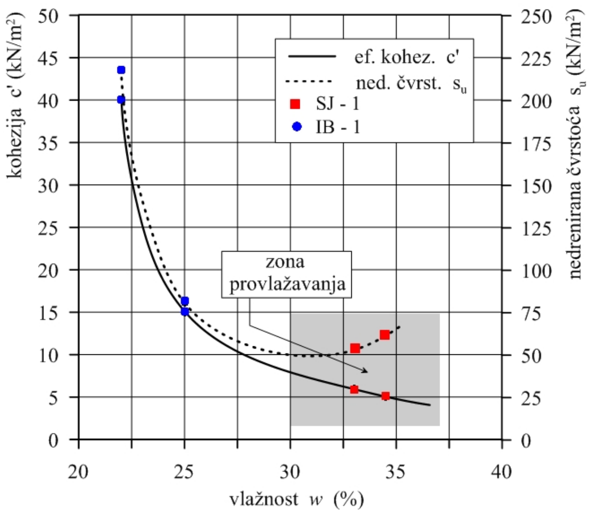

Fig. 6. *Reduction of shear strength of soil under facility due to moistening*

5. CONCLUSION

Based on the conducted investigations and analyses, it can be concluded that the unintended soil moistening occurs in a particular affected zone by changes in moisture, which is, for example in the horizontal direction, in the case of the tested silky clay ($al^{g,pr}$) relatively small, L < 2.0 m. As the soil moistening has mainly the local character, so the zone of subsequent settlement has the local level. The consequence of this is the appearance of differential settlement in a very short distance, which can cause minor or major damage on the facility. To return the facility in exploitation condition, it is necessary to make its rehabilitation, which must include the foundation repair, rehabilitation of system for controlled discharge of surface and storm water and rehabilitation of the structure. All these measures of rehabilitation require much higher financial resources than those needed to build and maintain a system that should prevent the moistening of foundation soil.

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BITNO POBOLJŠANJE ISKORIŠĆENJA RASPOLOŽIVIH GEOLOŠKIH REZERVI LEŽIŠTA JUŽNI REVIR MAJDANPEK U NOVO DEFINISANOJ OPTIMALNOJ KONTURI KOPA PRIMENOM SOFTVERSKIH PAKETA WHITTLE I GEMCOM[***]

Izvod

Značajan porast cene bakra na svetskoj berzi metala, čija donja granica dugoročno gledano neće biti ispod 6.000 $ po toni katodnog bakra, kao i smanjenje troškova eksploatacije rude uvođenjem visokokapacitativne opreme u proces proizvodnje, zahteva novo sagledavanje razvoja površinskog kopa Južni revir Majdanpek i definisanje nove konačne (optimalne) granice otkopavanja za date tehno ekonomske parametre.

Primenom savremenih softverskog alata za strateško planiranje i optimizaciju ležišta Whittle i softvera za konstrukciju površinskih kopova Gemcom definisana je optimalna kontura kopa po principu ostvarenja maksimalnog profita. U novodefinisanoj konturi analiziranog kopa zahvaćene su značajno veće geološke rezerve ležišta u odnosu na dosadašnja sagledavanja i one iznose 172.388.652 t rude sa srednjim sadržajem bakra u rudi 0,383 %.

Ključne reči: *Whittle, Gemcom, geološke rezerve, optimizacija, Južni revir Majdanpek*

UVOD

Rudnik bakra Majdanpek u proizvodnom, tehnièkom i tehnološkom smislu predstavlja kompleksan rudarski sistem koji ima aktivnosti od geoloških istraživanja mineralnih resursa, eksploatacije i pripreme rude do niza prateæih aktivnosti kao neophodne podrške osnovnim delatnostima. Proizvodnja i prerada rude u RBM, koja se na dva površinska kopa Severni revir i Južni revir odvija sa promenljivim kapacitetom neprekidno veæ više od 50 godina, od izuzetnog je znaèaja za proizvodnju bakra u sistemu RTB-a.

Ležište bakra Južni revir Majdanpek, nalazi se južno od grada Majdanpek, u njegovoj neposrednoj blizini.

Prema Planu razvoja proizvodnje rude bakra u RTB-u, koji je usvojen od strane

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menadžmenta kompanije, površinski kop Južni revir označen je kao nosioc proizvodnje rude bakra u RBM, sa godišnjim kapacitetom na otkopavanju i preradi od 8,5 miliona tona.

Značajan porast cene bakra na svetskoj berzi metala, čija donja granica dugoročno gledano neće biti ispod 6.000 $ po toni katodnog bakra uticala je na povećanje geoloških rezervi rude usled sniženja graničnog sadržaja bakra sa 0,2% na 0,15%. Sva dosadašnja pozitivna dešavanja u proizvodnji rude bakra u svetu ukazuju na potrebu sagledavanja novog koncepta razvoja površinskog kopa Južni revir i definisanje nove konačne (optimalne) granice otkopavanja za date tehno - ekonomske parametre.

Na osnovu postojećih uslova eksploatacije definisan je dalji razvoj površinskog kopa, uz ostvarivanje maksimalne dobiti (profita) pri eksploataciji i maksimalno iskorišćenje geoloških rezervi.

GEOLOŠKE REZERVE U LEŽIŠTU

Geološke rezerve ležište bakra Južni revir Majdanpek sračunate su metodom mini blokova, pri čemu su dimenzije blokova 15×15×15 m. Osnova za za proračun geoloških rezervi jeste digitalni blokmodel ležišta koji je formiran u softveru Gemcom.

U zavisnosti od promene graničnog sadržaja bakra u rudi, menjaju se i količine geoloških rezervi u ležištu. Granični sadržaj predstavlja promenjiv ekonomski pokazatelj, na koga utiču brojni činioci i kao takav podložan je varijacijama i u pogledu vremena i u pojedinim delovima rudnog tela. On pre svega zavisi od cene bakra na svetskom tržištu, troškova proizvodnje, iskorišćenja i razblaženja pri otkopavanju, flotacijskom i metalurškom tretmanu rude, investicija i dr.

Za potrebe izrade *Feasibility study površinskog kopa Južni revir rudnika bakra Majdanpek* 2001. godine proračunate su geološke rezerve u konturi graničnog sadržaja bakra 0,2% koji je u datim uslovima ocenjen kao realan i koji je obezbeđivao pokrivanje troškova dobijanja bakra iz rude. U tabeli 1 date su količine proračunatih geoloških rezervi.

Tabela 1. *Geološke rezerve u ležištu bakra Južni revir Majdanpek, u konturi graničnog sadržaja 0,20% Cu*

Količina rude (t)	Srednji sadržaj Cu (%)	Količina Cu (t)	Srednji sadržaj Au (g/t)	Količina Au (kg)	Srednji sadržaj Ag (g/t)	Količina Ag (kg)
420030400	0,335	1356670	0,188	76255	1,260	510883

Promena navedenih faktora koji utiču na vrednost graničnog sadržaja bakra u rudi, a naročito značajan porast cene bakra na svetskoj berzi metala, pri čemu se procenjuje da će dugoročno biti oko 6.000 $ po toni katodnog bakra, uzrokovala je potrebu da se ponovo odredi vrednost graničnog sadržaja. Granični sadržaj je određen tako što je izvršena optimizacija više varijanti kopova u konturi graničnog sadržaja bakra 0,15% i 0,20%. Na osnovu analize rezultata urađenih varijanti usvojena je vrednost graničnog sadržaja bakra 0,15%.

U tabeli 2 dat je pregled geoloških rezervi ležišta bakra Južni revir Majdanpek sa srednjim sadržajem i količinama korisnih komponenti u konturi graničnog sadržaja 0,15% Cu.

Tabela 2. *Geološke rezerve u ležištu bakra Južni revir Majdanpek, u konturi graničnog sadržaja 0,15% Cu*

Količina rude (t)	**Srednji sadržaj Cu (%)**	**Količina Cu (t)**	**Srednji sadržaj Au (g/t)**	**Količina Au (kg)**	**Srednji sadržaj Ag (g/t)**	**Količina Ag (kg)**
463.127.844	0,316	1.465.556	0,178	82.156	1,365	632.274

ISKORIŠĆENJE GEOLOŠKIH REZERVI LEŽIŠTA U KONTURI KOPA DEFINISANOJ U FEASIBILITY STUDY IZ 2001. GOD.

Optimalna kontura površinskog kopa u *Feasibility study* definisana je korišćenjem softvera za optimizaciju *Whittle 4D* i softvera za projektovanje *Gemcom*.

Ukupne zahvaćene količine rude i jalovine u optimalnoj konturi površinskog kopa Južni revir Majdanpek, definisanoj u *Feasibility study* iznose:

- ukupna količina iskopina, t 338.298.983
- količina jalovine, t 233.891.693
- količina rude, t 106.407.291
- granični sadržaj bakra u rudi, % Cu 0,20

Na osnovu količina geoloških rezervi rude i količina rude u optimalnoj konturi kopa definisanoj u *Feasibility study* (eksploatacione količine rude) dobija se stepen iskorišćenja geoloških rezervi ležišta Južni revir Majdanpek:

$$l = R_e / R_g \cdot 100, \% \qquad (1)$$

gde su:

R_e – eksploatacione rezerve, (t)

R_g – geološke rezerve, (t).

$$l = 22{,}98\%$$

ISKORIŠĆENJE GEOLOŠKIH REZERVI LEŽIŠTA U NOVO DEFINISANOJ OPTIMALNOJ KONTURI KOPA

Nova optimalna kontura kopa dobijena je optimizacijom ležišta primenom softvera *Whittle 4.1.3*. Optimizacija površinskog kopa izvršena je na osnovu blok modela ležišta i novo definisanih ulaznih tehno-ekonomskih parametara, koji obuhvataju cenu proizvodnje (u eksploataciji, flotaciji i metalurškoj preradi), gubitke, razblaženja i cene proizvoda i investicije. Ekonomska vrednost ležišta određena je na osnovu vrednosti metala bakra, zlata i srebra. Ekonomski efekti eksploatacije rude bakra obračunati su za granični sadržaj metala bakra u rudi od 0,15 %. Blokovi čiji je sadržaj bakra ispod graničnog sadržaja tretiraju se kao jalovina.

Izbor završne – optimalne konture kopa izvršen je za baznu cenu bakra od 6.000 $. U slučaju poremećaja na tržištu metala, eksploatacija rude u projektovanoj konturi kopa biće rentabilna do granične cene bakra od 4.560 $. Veća cena od planirane znači da će i ekonomski efekti poslovanja biti bolji od planiranih, tj. ostvariće se veći profit.

Na osnovu graničnog sadržaja metala bakra u rudi izvršen je obračun osnovnih pokazatelja optimizacije u dobijenim granicama kopova, prikazani u tabeli 3. U ovoj tabeli, pored fizičkih pokazatelja, odnosno ukupnih količina iskopina (rude i jalovine), za pojedine granice kopa prikazan je obračunati profit kao sadašnja vrednosti PV (*Present value*) za tri varijante analize – „*best case*“, „*worst case*“ i „*specified case*“ koje definišu način prostornog razvoja kopa.

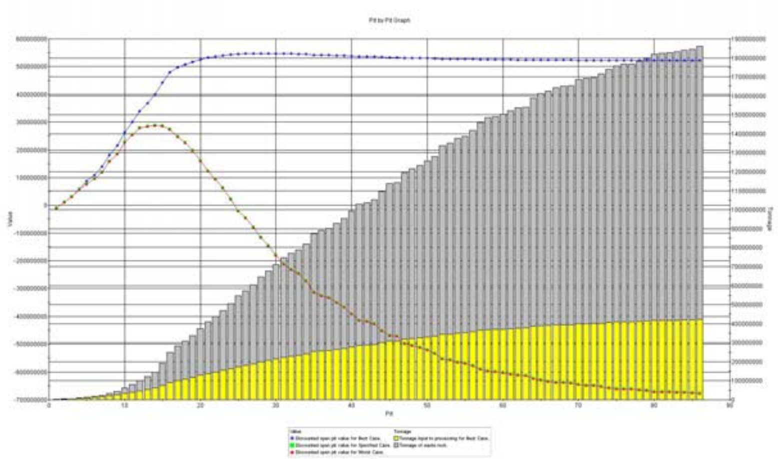
Određeni karakteristični pokazatelji iz tabele 3 prikazani su i grafički na slici 1. Iz ove tabele i sa grafika može se videti da je, za usvojene tehnoekonomske parametre kao optimalan izabran kop broj 24.

Tabela 3. *Osnovni pokazatelji u granicama kopova za GS=0,15 % Cu*

Kop	Iskopine t	Ruda t	Otkrivka t	Novčani tok (cashflow best) $ disc	Novčani tok (cashflow specified) $ disc	Novčani tok (cashflow worst) $ disc	Vek kopa (best) god
1	2 302 906	1 912 371	390 535	- 9 344 431	- 9 344 431	- 9 344 431	0.4
2	4 039 078	3 226 408	812 670	11 439 520	11 439 520	11 439 520	0.7
3	5 986 761	4 585 159	1 401 602	30 082 524	30 082 524	30 082 524	1.0
4	9 362 613	7 038 325	2 324 288	59 367 300	56 263 673	56 263 673	2.2
5	13 636 219	9 602 186	4 034 033	86 626 318	77 093 089	77 093 089	2.7
6	17 373 844	12 051 904	5 321 940	108 028 257	96 437 521	96 437 521	3.1
7	23 302 946	15 563 235	7 739 711	138 231 813	119 923 506	119 923 506	3.5
8	34 318 646	20 942 520	13 376 126	180 890 243	157 900 716	157 900 716	4.1
9	44 337 317	25 772 607	18 564 710	214 648 680	185 869 581	185 869 581	4.7
10	62 111 273	33 614 979	28 496 294	263 115 511	227 134 543	227 134 543	5.6
11	79 657 237	39 686 567	39 970 670	300 499 064	253 281 635	253 281 635	6.3
12	100 797 909	47 809 283	52 988 626	339 195 291	278 518 314	278 518 314	7.3
13	120 833 343	54 219 026	66 614 317	368 525 680	284 102 927	284 102 927	8.0
14	146 138 243	62 738 800	83 399 443	400 350 576	287 949 676	287 949 676	9.0
15	191 543 021	76 376 331	115 166 690	441 733 965	285 322 652	285 322 652	11.0
16	248 279 535	91 006 391	157 273 144	479 333 416	274 294 504	274 294 504	13.4
17	281 550 596	100 902 532	180 648 064	496 489 003	247 567 966	247 567 966	14.5
18	308 679 865	109 059 200	199 620 665	507 683 126	225 932 314	225 932 314	15.5
19	335 841 804	117 416 168	218 425 636	516 493 002	196 741 221	196 741 221	16.5
20	374 134 429	128 922 471	245 211 958	525 769 911	160 689 276	160 689 276	17.8
21	408 955 442	138 223 785	270 731 657	532 427 495	122 650 208	122 650 208	18.9
22	436 769 379	145 755 948	291 013 431	536 724 940	94 002 630	94 002 630	19.8
23	468 847 771	155 288 077	313 559 694	539 873 486	61 886 870	61 886 870	20.9
24	**506 593 841**	**164 394 316**	**342 199 525**	**543 261 888**	**22 190 377**	**22 190 377**	**22.0**
25	548 155 155	174 956 777	373 198 378	545 515 648	-21 290 269	- 21 290 269	23.2
26	572 318 338	181 433 655	390 884 683	546 314 134	-45 100 916	- 45 100 916	24.0
27	604 290 073	188 919 620	415 370 453	547 254 409	-80 059 049	- 80 059 049	24.9
28	646 217 017	200 173 014	446 044 003	547 130 124	-116 323 284	-116 323 284	26.3
29	676 711 254	207 429 174	469 282 080	547 232 012	-147 503 717	-147 503 718	27.2
30	709 826 636	214 697 978	495 128 658	547 238 885	-179 875 881	-179 875 882	28.0
31	747 603 904	223 691 253	523 912 651	546 449 491	-212 517 671	-212 517 671	29.3
32	769 660 899	228 645 478	541 015 421	545 930 783	-231 827 198	-231 827 198	29.9
33	787 220 506	233 158 133	554 062 373	545 355 197	-245 728 464	-245 728 465	30.5
34	818 922 693	240 047 751	578 874 942	544 433 888	-272 955 116	-272 955 115	31.5
35	872 999 324	252 772 544	620 226 780	542 225 129	-312 820 138	-312 820 139	33.5
36	890 704 707	256 599 153	634 105 554	541 745 151	-324 910 533	-324 910 532	33.9
37	902 191 262	259 288 874	642 902 388	541 365 536	-332 842 423	-332 842 422	34.2
38	927 354 940	264 570 021	662 784 919	540 569 702	-350 967 913	-350 967 914	34.9
39	953 246 108	270 351 581	682 894 527	539 640 540	-367 342 943	-367 342 944	35.7

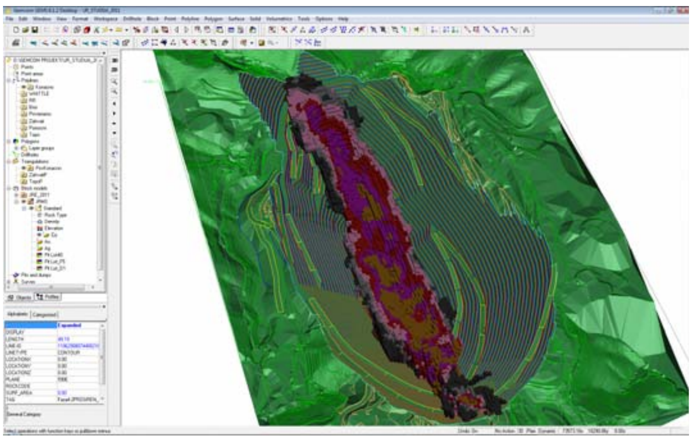
40	993 480 437	278 626 271	714 854 166	538 132 118	-392 125 203	-392 125 203	37.0
41	1 030 096 885	286 143 260	743 953 625	536 884 341	-413 906 011	-413 906 012	37.9
42	1 036 767 072	287 555 140	749 211 932	536 660 886	-417 705 266	-417 705 267	38.0
43	1 051 716 364	290 620 171	761 096 193	536 147 146	-426 208 901	-426 208 902	38.4
44	1 097 439 686	299 538 249	797 901 437	534 599 398	-450 851 840	-450 851 840	40.0
45	1 137 955 282	307 839 747	830 115 535	533 219 761	-469 599 691	-469 599 691	41.7
46	1 143 317 571	309 010 357	834 307 214	533 058 950	-472 060 337	-472 060 337	41.8
47	1 195 003 408	317 865 040	877 138 368	531 700 333	-497 948 139	-497 948 139	43.0
48	1 214 482 312	321 334 887	893 147 425	531 169 422	-505 589 534	-505 589 534	43.6
49	1 234 708 370	325 347 128	909 361 242	530 556 565	-512 854 265	-512 854 265	44.4
50	1 256 707 928	329 292 434	927 415 494	529 992 878	-521 562 220	-521 562 221	44.9
51	1 281 534 685	333 367 228	948 167 457	529 456 436	-532 520 826	-532 520 826	45.5
52	1 337 761 743	343 849 098	993 912 645	528 139 566	-552 478 337	-552 478 337	47.4
53	1 349 231 937	345 913 302	1 003 318 635	527 866 633	-555 884 858	-555 884 859	47.6
54	1 376 148 705	350 336 598	1 025 812 107	527 349 464	-565 410 919	-565 410 919	48.5
55	1 386 791 835	352 114 852	1 034 676 983	527 129 333	-568 818 625	-568 818 626	48.7
56	1 417 666 497	357 351 319	1 060 315 178	526 479 663	-577 925 967	-577 925 966	49.6
57	1 460 618 004	364 698 524	1 095 919 480	525 687 921	-590 433 709	-590 433 709	50.9
58	1 484 027 784	368 710 766	1 115 317 018	525 308 549	-597 436 576	-597 436 576	51.5
59	1 489 690 950	369 836 696	1 119 854 254	525 180 827	-598 378 490	-598 378 490	51.6
60	1 502 205 349	371 614 950	1 130 590 399	525 015 241	-602 516 770	-602 516 770	51.8
61	1 520 729 420	374 242 119	1 146 487 301	524 740 547	-607 800 911	-607 800 911	52.3
62	1 536 638 785	376 779 929	1 159 858 856	524 496 434	-611 829 200	-611 829 200	52.8
63	1 540 638 467	377 432 254	1 163 206 213	524 445 224	-612 805 381	-612 805 381	52.9
64	1 588 866 712	385 287 797	1 203 578 915	523 705 632	-624 139 963	-624 139 963	54.8
65	1 611 985 356	388 915 793	1 223 069 563	523 401 577	-629 409 487	-629 409 487	55.2
66	1 626 074 679	390 721 698	1 235 352 981	523 264 344	-633 579 892	-633 579 892	55.5
67	1 644 571 347	393 474 814	1 251 096 533	523 066 621	-637 371 403	-637 371 403	56.0
68	1 651 716 793	394 406 008	1 257 310 785	522 991 613	-638 612 404	-638 612 404	56.2
69	1 654 246 479	394 799 189	1 259 447 290	522 959 453	-639 111 752	-639 111 752	56.3
70	1 683 686 422	398 856 111	1 284 830 311	522 657 952	-645 682 469	-645 682 469	57.4
71	1 693 697 901	400 339 479	1 293 358 422	522 549 351	-648 031 278	-648 031 278	57.5
72	1 695 189 782	400 527 134	1 294 662 648	522 534 103	-648 396 315	-648 396 315	57.6
73	1 715 567 408	403 056 008	1 312 511 400	522 327 284	-652 628 866	-652 628 866	58.1
74	1 739 609 022	406 060 173	1 333 548 849	522 132 350	-657 525 893	-657 525 893	58.7
75	1 758 604 491	408 689 197	1 349 915 294	521 944 218	-660 429 434	-660 429 434	59.3
76	1 767 532 841	409 895 550	1 357 637 291	521 853 380	-661 829 170	-661 829 170	59.5
77	1 767 579 585	409 904 486	1 357 675 099	521 852 913	-661 836 216	-661 836 216	59.5
78	1 782 780 046	412 049 114	1 370 730 932	521 727 975	-664 379 228	-664 379 228	59.9
79	1 796 656 020	413 666 521	1 382 989 499	521 612 846	-666 964 109	-666 964 109	60.3
80	1 820 171 556	416 883 464	1 403 288 092	521 416 097	-670 793 096	-670 793 096	61.0
81	1 824 844 259	417 419 621	1 407 424 638	521 377 012	-671 563 421	-671 563 421	61.1

82	1 826 296 604	417 544 724	1 408 751 880	521 370 895	-671 745 589	-671 745 589	61.2
83	1 833 020 385	418 411 511	1 414 608 874	521 312 195	-672 764 711	-672 764 711	61.4
84	1 840 792 371	419 206 811	1 421 585 560	521 252 197	-674 167 324	-674 167 324	61.6
85	1 845 380 057	419 751 904	1 425 628 153	521 216 946	-674 851 913	-674 851 913	61.7
86	1 862 011 201	421 808 016	1 440 203 185	521 088 137	-677 639 340	-677 639 340	62.2



Sl. 1. *Grafički prikaz podataka za kopove*

Nakon detaljne grafičke obrade kopa broj 24 u softveru *Gemcom 6.2* dobijena je konačna kontura kopa Južni revir Majdanpek. Ova granica prikazana je na slici 2 u 3D formatu.



Sl. 2. *Izgled završne konture površinskog kopa sa prikazom blok modela ležišta (kop broj 24 detaljno grafički obrađen u softveru Gemcom 6.2)*

Ukupne zahvaćene količine rude i jalovine u novo definisanojoptimalnoj (konačnoj) konturi površinskog kopa Južni revirMajdanpek (eksploatacione rezerve) iznose:

- ukupna količina iskopina, t 549.874.663
- količina jalovine, t 377.858.498
- količina rude, t 172.016.165
- granični sadržaj bakra u rudi, % Cu 0,150

Na osnovu količina geoloških rezervi rude i rude u novo definisanoj optimalnoj konturi kopa dobija se stepen iskorišćenja geoloških rezervi ležišta Južni revir Majdanpek prema izrazu (1):

$$l = 37{,}14\,\%$$

ZAKLJUČAK

Proizvodnja i prerada rude bakra u RBM odvija se dva površinska kopa Severni revir i Južni revir i od izuzetnog je značaja za ukupnu proizvodnju u sistemu RTB-a.

Značajan porast cene bakra na svetskoj berzi metala, čija donja granica dugoročno gledano neće biti ispod 6.000 $ po toni katodnog bakra uticala je na povećanje geoloških rezervi rude bakra u ležištu Južni revir Majdanpek usled sniženja graničnog sadržaja bakra sa 0,2% na 0,15%. Usled toga nastala je potreba da se izvrši novo sagledavanje razvoja površinskog kopa Južni revir i definiše nova konačna (optimalna) granice otkopavanja za nove tehno ekonomske parametre.

Za potrebe izrade *Feasibility study površinskog kopa Južni revir rudnika bakra Majdanpek* 2001. godine proračunate su geološke rezerve u konturi graničnog sadržaja bakra 0,2% koji je u datim uslovima ocenjen kao realan i koji je obezbeđivao pokrivanje troškova dobijanja bakra iz rude. Proračunate količine geoloških rezervi iznosile su 420.030.400 t.

Promena faktora koji utiču na vrednost graničnog sadržaja bakra u rudi, a naročito porast cene bakra na svetskoj berzi metala, pri čemu se procenjuje da će dugoročno biti iznad 6.000 $ po toni katodnog bakra, zatim sniženje troškova proizvodnje, povećanje iskorišćenja i u flotacijskoj i metalurškojpreradi rude i dr. uzrokovala je potrebu da se ponovo odredi vrednost graničnog sadržaja. Granični sadržaj je određen tako što je izvršena optimizacija više varijanti kopova u konturi graničnog sadržaja bakra 0,15% i 0,20%. Na osnovu analize rezultata urađenih varijanti odlučeno je da se odabere vrednost graničnog sadržaja bakra 0,15%. Proračunate količine geoloških rezervi iznose 172.016.165 t.

Na osnovu proračunatih količina geoloških rezervi rude i količina rude u optimalnim kontura kopa (eksploatacione količine rude) dobija se stepen iskorišćenja geoloških rezervi ležišta Južni revir Majdanpek koji iznosi:

- u optimalnoj konturi kopa definisanoj u *Feasibility study* I = 22,98 %
- u novo definisanoj optimalnoj konturi kopa I = 37,14 %.

Na osnovu sprovede analize može se zaključiti da je iskorišćenje geoloških rezervi ležišta Južni revir Majdanpek bitno poboljšano u novo definisanoj optimalnoj konturi kopa, u odnosu na iskorišćenje u konturi kopa koja je definisana u *Feasibility study* iz 2001. godine.

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IMPORTANT IMPROVEMENT OF UTILIZATION THE AVAILABLE GEOLOGICAL RESERVES OF THE SOUTH MINING DISTRICT DEPOSIT IN MAJDANPEK IN THE NEW DEFINED OPTIMUM CONTOUR OF THE OPEN PIT USING THE WHITTLE AND GEMCOM SOFTWARES[***]

Abstract

A significant increase in copper prices on the World Metal Exchange, whose lower limit in the long term will not be under $ 6,000 per ton of cathode copper, and reduction the operating costs of ore mining by introduction the high capacity mining equipment in the production process, requires a new consideration of the open pit South Mining District Majdanpek and defining the new final (optimum) limit of mining for the given techno economic parameters.

Using the modern software tools for strategic planning and optimization of deposit Whittle and software for design the open pit Gemcom, the optimum contour of open pit was designed on the principle of maximum profit realization. In the newly-defined contour of analyzed open pit, significantly higher geological reserves of deposit are affected in relation to the previous considerations of those amounts to the amount of 172,388,652 t of ore with average copper content in the ore of 0.383%.

Key words: *Whittle, Gemcom, geological reserves, optimization, South Mining District Majdanpek*

INTRODUCTION

The Copper Mine Majdanpek in the production, technical and technological terms is a complex mining system that has the activities from geological explorations of mineral resources, mining and mineral processing to a series of related activities as the necessary support of core activities. The ore production and processing process in RBM, at two open pits - the North and South Mining District, with variable

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capacity continuously for more than 50 years, is of a great importance for copper production in the system of RTB.

The South Mining District copper deposit in Majdanpek is located in the south of the town Majdanpek in the immediate vicinity.

According to the Plan of development the copper production in RTB, adopted by the management of company, the open pit South Mining District is designated as the holder of copper production in RBM, with annual capacity of mining and processing of 8.5 million tons.

A significant increase in copper prices on the World Metal Exchange, whose lower limit in the long term will not be under $ 6,000 per ton of cathode copper resulted in increase of geological reserves due to decrease of copper content from 0.2% to 0.15%. All previous positive developments in the production of copper ore in the world implies to a need for consideration a new concept of development the open pit South Mining District and defining a new final (optimum) limit of mining for the given techno economic parameters.

Based on the existing mining conditions, further development of the open pit was defined, with realization of maximum profit in mining and maximum utilization of geological reserves.

GEOLOGICAL RESERVES OF DEPOSIT

Geological reserves of the copper deposit Southern Mining District Majdanpek were calculated by the method of mini-blocks, with block sizes 15 × 15 × 15 m. Base for calculation the geological reserves is a digital block model of deposit, formed in the software Gemcom.

Depending on a change of copper limit content in the ore, the amounts of geological reserves are changed in the deposit. Limit content is a variable economic indicator, which is affected by many factors and as such is subject to variations in terms of time and in some parts of the ore body. It mainly depends on the copper price on the world market, production costs, efficiency and dilution in mining, flotation and metallurgical treatment of ore, investments, etc.

For the needs of development the *Feasibility Study of the Open Pit South Mining District Majdanpek,* the geological reserves were calculated in 2001 in a contour of the copper limit content of 0.2%, which is, in the given conditions, rated as real and provided covering the costs of copper production from the ore. Table 1 gives the calculated amounts of geological reserves.

Table 1 *Geological reserves in the copper deposit South Mining District Majdanpek, in a contour of limit Cu content of 0.20% Cu*

Ore amount (t)	Average Cu content (%)	Cu amoun t(t)	Average Au content (g/t)	Au amount (kg)	Average Ag content (g/t)	Ag amount (kg)
420030400	0,335	135667 0	0,188	76255	1,260	510883

Changing the factors that affect the value of limit copper content in the ore, and in particular a significant increase in copper prices on the World Metal Exchange, where it is estimated to be a long term about 6,000 $ per ton of cathode copper, resulted in a need to re-determine the value of limit content. Limit content is determined by the optimization of many variants of the open pits in a contour of limit copper content of 0.15% and 0.20%. Based on the result analysis of performed variants, the adopted value of limit copper content is 0.15%.

Table 2 gives a review of geological reserves of the copper deposit South Mining District Majdanpek with average content and amounts of useful components in a contour of limit Cu content of 0.15%.

Table 2. *Geological reserves in the copper deposit South Mining District Majdanpek, in a contour of limit Cu content of 0.15% Cu*

Ore amount (t)	Average Cu content (%)	Cu amount(t)	Average Au content (g/t)	Au amount (kg)	Average Ag content (g/t)	Ag amount (kg)
463127844	0,316	1465556	0,178	82156	1,365	632274

UTILIZATION OF GEOLOGICAL RESERVES OF DEPOSIT IN A CONTOUR OF THE OPEN PIT, DEFINED IN THE FEASIBILITY STUDY FROM 2001

Optimum contour of the open pit in the *Feasibility Study* is defined using the software for optimization *Whittle 4D* and software for design *Gemcom*.

Total volume of ore and waste rock in an optimum contour of the open pit South Mining District Majdanpek, defined in the *Feasibility Study* are:

total quantity of excavations, t 338.298.983
quantity of waste, t 233.891.693
quantity of ore, t 106.407.291
limit copper content in the ore, % Cu 0.20

Based on the amounts of geological reserves of ore and the amount of ore in the optimum pit contour, defined in the *Feasibility Study* (mining quantities of ore), a degree of utilization the geological reserves is obntained in the South Mining District Majdanpek:

$$l = R_e / R_g \cdot 100, \% \qquad (1)$$

where:

R_e – mining reserves, (t)
R_g – geological reserves, (t).

$$l = 22{,}98\%$$

UTILIZATION THE GEOLOGICAL RESERVES OF DEPOSIT IN A NEWLY-DEFINED OPTIMUM CONTOUR OF THE OPEN PIT

The new optimum open pit contour is obtained by optimization of deposit using the software *Whittle 4.1.3.* Open pit optimization was performed on the basis of a deposit block model and newly-defined techno-economic parameters, which include the production costs (in mining, flotation and metallurgical treatment), losses, dilution, and product prices and investments. The economic value of deposit was determined on the basis of value of copper, gold and silver metals. The economic effects of copper ore mining were calculated for the limit of copper metal content in the ore of 0.15%. Blocks with copper content below the limit content are treated as waste rock.

Selection of final - optimum pit contour was made for the base copper price of 6,000 $. In the case of disturbances on the metal market, the ore mining in the projected contour will be profitable to the limit copper price of $ 4,560. Higher copper price than the planned one means that the economic effects of business will be better than expected, i.e. a higher profit will be achieved.

Based on the limit content of copper metal in the ore, the calculation of basic optimization indicators was carried out for the obtained open pit limits, shown in Table 3. In this table, in addition to the physical parameters, i.e. total amount of excavations (ore and waste rock), the calculated profit is presented for individual limits of open pits as the present value PV (*Present value*) for three variants of analyses – *the* "*best case*", "*worst case*" and "*specified case*" that define the way of spatial mine development.

Some typical indicators are shown in Table 3 graphically in Figure 1.

This table and graphics present the open pit no.24 as optimum selected for adopted techno-economic parameters.

Table 3. *Basic indicators within the open pit limits for GS=0.15 % Cu*

ž	Excavations t	Ore t	Overburden t	Cash flow best $ disc	Cash flow specified $ disc	Cash flow worst $ disc	Life time of open pit (best) year
1	2 302 906	1 912 371	390 535	- 9 344 431	- 9 344 431	- 9 344 431	0.4
2	4 039 078	3 226 408	812 670	11 439 520	11 439 520	11 439 520	0.7
3	5 986 761	4 585 159	1 401 602	30 082 524	30 082 524	30 082 524	1.0
4	9 362 613	7 038 325	2 324 288	59 367 300	56 263 673	56 263 673	2.2
5	13 636 219	9 602 186	4 034 033	86 626 318	77 093 089	77 093 089	2.7
6	17 373 844	12 051 904	5 321 940	108 028 257	96 437 521	96 437 521	3.1
7	23 302 946	15 563 235	7 739 711	138 231 813	119 923 506	119 923 506	3.5
8	34 318 646	20 942 520	13 376 126	180 890 243	157 900 716	157 900 716	4.1
9	44 337 317	25 772 607	18 564 710	214 648 680	185 869 581	185 869 581	4.7
10	62 111 273	33 614 979	28 496 294	263 115 511	227 134 543	227 134 543	5.6
11	79 657 237	39 686 567	39 970 670	300 499 064	253 281 635	253 281 635	6.3
12	100 797 909	47 809 283	52 988 626	339 195 291	278 518 314	278 518 314	7.3
13	120 833 343	54 219 026	66 614 317	368 525 680	284 102 927	284 102 927	8.0
14	146 138 243	62 738 800	83 399 443	400 350 576	287 949 676	287 949 676	9.0
15	191 543 021	76 376 331	115 166 690	441 733 965	285 322 652	285 322 652	11.0
16	248 279 535	91 006 391	157 273 144	479 333 416	274 294 504	274 294 504	13.4
17	281 550 596	100 902 532	180 648 064	496 489 003	247 567 966	247 567 966	14.5
18	308 679 865	109 059 200	199 620 665	507 683 126	225 932 314	225 932 314	15.5
19	335 841 804	117 416 168	218 425 636	516 493 002	196 741 221	196 741 221	16.5
20	374 134 429	128 922 471	245 211 958	525 769 911	160 689 276	160 689 276	17.8
21	408 955 442	138 223 785	270 731 657	532 427 495	122 650 208	122 650 208	18.9
22	436 769 379	145 755 948	291 013 431	536 724 940	94 002 630	94 002 630	19.8
23	468 847 771	155 288 077	313 559 694	539 873 486	61 886 870	61 886 870	20.9
24	**506 593 841**	**164 394 316**	**342 199 525**	**543 261 888**	**22 190 377**	**22 190 377**	**22.0**
25	548 155 155	174 956 777	373 198 378	545 515 648	-21 290 269	- 21 290 269	23.2
26	572 318 338	181 433 655	390 884 683	546 314 134	-45 100 916	- 45 100 916	24.0
27	604 290 073	188 919 620	415 370 453	547 254 409	-80 059 049	- 80 059 049	24.9
28	646 217 017	200 173 014	446 044 003	547 130 124	-116 323 284	-116 323 284	26.3
29	676 711 254	207 429 174	469 282 080	547 232 012	-147 503 717	-147 503 718	27.2
30	709 826 636	214 697 978	495 128 658	547 238 885	-179 875 881	-179 875 882	28.0
31	747 603 904	223 691 253	523 912 651	546 449 491	-212 517 671	-212 517 671	29.3
32	769 660 899	228 645 478	541 015 421	545 930 783	-231 827 198	-231 827 198	29.9
33	787 220 506	233 158 133	554 062 373	545 355 197	-245 728 464	-245 728 465	30.5
34	818 922 693	240 047 751	578 874 942	544 433 888	-272 955 116	-272 955 115	31.5
35	872 999 324	252 772 544	620 226 780	542 225 129	-312 820 138	-312 820 139	33.5
36	890 704 707	256 599 153	634 105 554	541 745 151	-324 910 533	-324 910 532	33.9

37	902 191 262	259 288 874	642 902 388	541 365 536	-332 842 423	-332 842 422	34.2
38	927 354 940	264 570 021	662 784 919	540 569 702	-350 967 913	-350 967 914	34.9
39	953 246 108	270 351 581	682 894 527	539 640 540	-367 342 943	-367 342 944	35.7
40	993 480 437	278 626 271	714 854 166	538 132 118	-392 125 203	-392 125 203	37.0
41	1 030 096 885	286 143 260	743 953 625	536 884 341	-413 906 011	-413 906 012	37.9
42	1 036 767 072	287 555 140	749 211 932	536 660 886	-417 705 266	-417 705 267	38.0
43	1 051 716 364	290 620 171	761 096 193	536 147 146	-426 208 901	-426 208 902	38.4
44	1 097 439 686	299 538 249	797 901 437	534 599 398	-450 851 840	-450 851 840	40.0
45	1 137 955 282	307 839 747	830 115 535	533 219 761	-469 599 691	-469 599 691	41.7
46	1 143 317 571	309 010 357	834 307 214	533 058 950	-472 060 337	-472 060 337	41.8
47	1 195 003 408	317 865 040	877 138 368	531 700 333	-497 948 139	-497 948 139	43.0
48	1 214 482 312	321 334 887	893 147 425	531 169 422	-505 589 534	-505 589 534	43.6
49	1 234 708 370	325 347 128	909 361 242	530 556 565	-512 854 265	-512 854 265	44.4
50	1 256 707 928	329 292 434	927 415 494	529 992 878	-521 562 220	-521 562 221	44.9
51	1 281 534 685	333 367 228	948 167 457	529 456 436	-532 520 826	-532 520 826	45.5
52	1 337 761 743	343 849 098	993 912 645	528 139 566	-552 478 337	-552 478 337	47.4
53	1 349 231 937	345 913 302	1 003 318 635	527 866 633	-555 884 858	-555 884 859	47.6
54	1 376 148 705	350 336 598	1 025 812 107	527 349 464	-565 410 919	-565 410 919	48.5
55	1 386 791 835	352 114 852	1 034 676 983	527 129 333	-568 818 625	-568 818 626	48.7
56	1 417 666 497	357 351 319	1 060 315 178	526 479 663	-577 925 967	-577 925 966	49.6
57	1 460 618 004	364 698 524	1 095 919 480	525 687 921	-590 433 709	-590 433 709	50.9
58	1 484 027 784	368 710 766	1 115 317 018	525 308 549	-597 436 576	-597 436 576	51.5
59	1 489 690 950	369 836 696	1 119 854 254	525 180 827	-598 378 490	-598 378 490	51.6
60	1 502 205 349	371 614 950	1 130 590 399	525 015 241	-602 516 770	-602 516 770	51.8
61	1 520 729 420	374 242 119	1 146 487 301	524 740 547	-607 800 911	-607 800 911	52.3
62	1 536 638 785	376 779 929	1 159 858 856	524 496 434	-611 829 200	-611 829 200	52.8
63	1 540 638 467	377 432 254	1 163 206 213	524 445 224	-612 805 381	-612 805 381	52.9
64	1 588 866 712	385 287 797	1 203 578 915	523 705 632	-624 139 963	-624 139 963	54.8
65	1 611 985 356	388 915 793	1 223 069 563	523 401 577	-629 409 487	-629 409 487	55.2
66	1 626 074 679	390 721 698	1 235 352 981	523 264 344	-633 579 892	-633 579 892	55.5
67	1 644 571 347	393 474 814	1 251 096 533	523 066 621	-637 371 403	-637 371 403	56.0
68	1 651 716 793	394 406 008	1 257 310 785	522 991 613	-638 612 404	-638 612 404	56.2
69	1 654 246 479	394 799 189	1 259 447 290	522 959 453	-639 111 752	-639 111 752	56.3
70	1 683 686 422	398 856 111	1 284 830 311	522 657 952	-645 682 469	-645 682 469	57.4
71	1 693 697 901	400 339 479	1 293 358 422	522 549 351	-648 031 278	-648 031 278	57.5
72	1 695 189 782	400 527 134	1 294 662 648	522 534 103	-648 396 315	-648 396 315	57.6
73	1 715 567 408	403 056 008	1 312 511 400	522 327 284	-652 628 866	-652 628 866	58.1
74	1 739 609 022	406 060 173	1 333 548 849	522 132 350	-657 525 893	-657 525 893	58.7
75	1 758 604 491	408 689 197	1 349 915 294	521 944 218	-660 429 434	-660 429 434	59.3
76	1 767 532 841	409 895 550	1 357 637 291	521 853 380	-661 829 170	-661 829 170	59.5
77	1 767 579 585	409 904 486	1 357 675 099	521 852 913	-661 836 216	-661 836 216	59.5
78	1 782 780 046	412 049 114	1 370 730 932	521 727 975	-664 379 228	-664 379 228	59.9
79	1 796 656 020	413 666 521	1 382 989 499	521 612 846	-666 964 109	-666 964 109	60.3
80	1 820 171 556	416 883 464	1 403 288 092	521 416 097	-670 793 096	-670 793 096	61.0
81	1 824 844 259	417 419 621	1 407 424 638	521 377 012	-671 563 421	-671 563 421	61.1

82	1 826 296 604	417 544 724	1 408 751 880	521 370 895	-671 745 589	-671 745 589	61.2
83	1 833 020 385	418 411 511	1 414 608 874	521 312 195	-672 764 711	-672 764 711	61.4
84	1 840 792 371	419 206 811	1 421 585 560	521 252 197	-674 167 324	-674 167 324	61.6
85	1 845 380 057	419 751 904	1 425 628 153	521 216 946	-674 851 913	-674 851 913	61.7
86	1 862 011 201	421 808 016	1 440 203 185	521 088 137	-677 639 340	-677 639 340	62.2

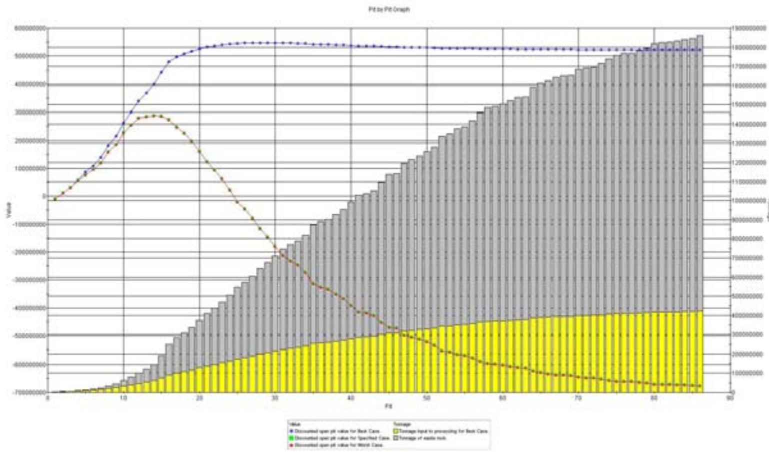

Fig. 1. *Graphical presentation of data for open pits*

After detailed graphical processing the open pit no.24 in the software *Gemcom6.2,* the final contour of the open pit South Mining District Majdanpek was obtained. This limit is shown in Figure 2 in 3D.

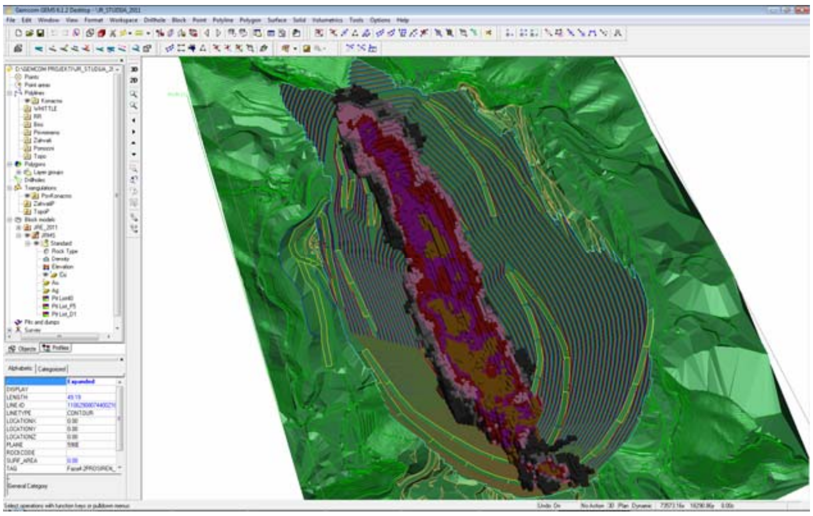

Fig. 2. *View of the final contours of the open pit showing the deposit block model (open pit no. 24 with in detail graphics, processed in the software Gemcom 6.2)*

Total included quantities of ore and waste rock in the newly-defined optimum (final) contour of the open pit South Mining District Majdanpek (mining reserves) are:

total quantity of excavations, t	549.874.663
quantity of waste, t	377.858.498
quantity of ore, t	172.016.165
limit copper content in the ore, % Cu	0.150

Based on the amount of geological reserves of ore and ore in the newly-defined optimum contour of the open pit, the efficiency degree of geological reserves in the South Mining District Majdanpek is obtained according to the expression (1):

$$l = 37{,}14\,\%$$

CONCLUSION

The ore production and processing in RBM have been done at two open pits - the North and South Mining District and have great importance for copper production in the system of RTB.

A significant increase in copper prices on the World Metal Exchange, whose lower limit in the long term will not be under $ 6,000 per ton of cathode copper resulted in increase of geological reserves of copper ore in the South Mining District Majdanpek due to decrease of limit copper content from 0.2% to 0.15%. All this implies a need for consideration a new concept of development the open pit South Mining District and defining a new final (optimum) limit of mining for the mew techno economic parameters.

For the needs of development the *Feasibility Study of the Open Pit South Mining District Majdanpek,* the geological reserves were calculated in 2001 in a contour of the copper limit content of 0.2%, which is, in the given conditions, rated as real and provided covering the costs of copper production from the ore. Calculated amounts of geological reserves were 420,030,400 t.

Changing the factors that affect the value of limit copper content in the ore, and in particular a significant increase in copper prices on the World Metal Exchange, where it is estimated to be a long term over 6,000 $ per ton of cathode copper, then decrease of production costs, increase of utilization both in the flotation and metallurgical treatment, etc., resulted in a need to re-determine the value of limit content. Limit content is determined by the optimization of many variants of the open pits in a contour of limit copper content of 0.15% and 0.20%. Based on the result analysis of performed variants, the adopted value of limit copper content is 0.15%. Calculated amounts of geological reserves are 172,016,165 t.

Based on the calculated amount of geological ore reserves and amounts of ore in the optimum open pit contours (mining ore quantities)in optimal amounts of contour mining (exploitation quantities of ore), the efficiency degree of geological reserves in the South Mining District Majdanpek is obtained, which is:

- in the optimum open pit contour, defined in the *Feasibility Study* I = 22.98%
- in the newly-defined optimum open pit contour I = 37.14%.

It could be concluded, based on carried out analysis that the utilization of the ore reserves in the deposit of the South Mining District Majdanpek was significantly improved in the newly-defined open pit contour regarding to the utilization in the open pit contour, defined in the *Feasibility study* of 2001.

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*Lidija Đurđevac Ignjatović**

ODREĐIVANJE PARAMETARA SMICANJA STENA PO POVLAŠĆENIM RAVNIMA**

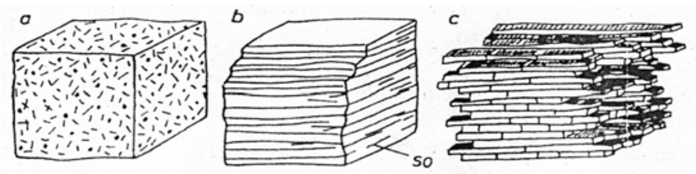
Izvod

U ovom radu su prikazani rezultati ispitivanja otpornosti na smicanje u elastičnoj, anizotropnoj sredini u tri uzajamno normalna pravca. Istražno područje je transverzalno izotropan model, koga karakterišu konstantna fizičko-mehanička svojstva u različitim pravcima u ravni izotropije i različita svojstva u pravcima normalnim na ravan izotropije.
Ključne reči: *smicanje, anizotropna sredina, fizičko-mehanička svojstva, ravan izotropije.*

UVOD

S obzirom na raznolikost građe dela stenske mase i njenih svojstava na kojoj se izvode određeni građevinski objekti, teško je razraditi model koji opisuje svu složenost građe i naponsko-deformacijsko stanje. Takav model mora biti jednostavan, a dobijeni rezultati prihvatljivi.

Prvi korak istraživanja u području mehanike stena je razrada strukturnog modela koji odražava sve osobenosti građe dela stenske mase i njihova fizičko-mehanička i deformaciona svojstva, sl. 1.



Sl. 1. *Strukturni model*

a – neorijentisana izotropna zrnasta struktura
b – slojevita anizotropna struktura
c – škriljava anizotropna struktura

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** *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*

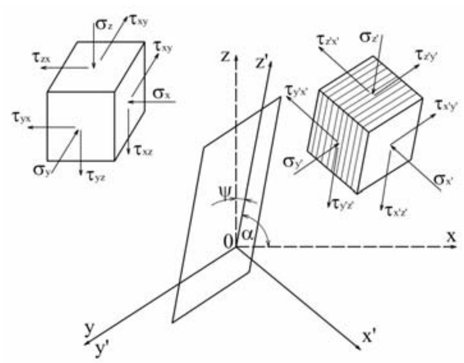
Na osnovu strukturnog stenskog modela moguće je razraditi takav geomehanički model stenske mase na koji se mogu primeniti numeričke analize metode konačnih elemenata pri rešavanju praktičnih zadataka pri temeljenju određenih građevinskih objekata.

Anizotropni elastični model stenske mase razlikuje se od izotropnog elastičnog modela što uzima u obzir izmenu svojstava masiva u različitim pravcima. U opštem slučaju [1] anizotropnu sredinu karakteriše 18 nezavisnih konstanti dok, poređenja radi, izotropnu sredinu karakterišu samo dve konstante (modul elastičnosti i koeficijent Poisson-a).

Najjednostavniji anizotropni model stenske mase je transverzalno-izotropni [3] (u prirodi su joj najbliži tankoslojni škriljci).

1. ŠKRILJAVA ANIZOTROPNA STRUKTURA PREGRADNOG MESTA H.E. „BRODAREVO 2“

Škriljci lokaliteta pregradnog mesta H.E. „Brodarevo 2“ su subvertikalne tankoslojne uslojenosti, koja se vrlo brzo na vazduhu razdvaja i prelazi u trošnu sredinu. Prema strukturnom modelu je škriljava anizotropna sredina. Ta sredina se karakteriše konstantnim fizičko-mehaničkim svojstvima u različitim pravcima u ravni izotropije poklapajući se sa ravni x’y’z’, sl. 2. i različitim svojstvima u pravcu ose x’, koja predstavlja osu simetrije elastične sredine.



Sl. 2. *Šema napona u transverzalno-izotropnoj sredini*

1.1. Priprema uzoraka škriljca za određivanje parametara otpora smicanju

Za definisanje parametara otpora smicanju škriljca (transverzalno izotropnog modela) u tri uzajamno okomita pravca neophodno je pripremiti 9 (devet) uzoraka dimenzija (8 x 8 x 8) [cm], koristeći Mor-Kulon-ov zakon, da rušenje stene nastupa po ravni najmanjeg otpora smicanju. Definisanje tog zakona je moguće sa minimum tri tačke:

$$\tau = c + \sigma_n \, tg\varphi \qquad (1)$$

τ - otpor smicanju po unapred definisanoj ravni

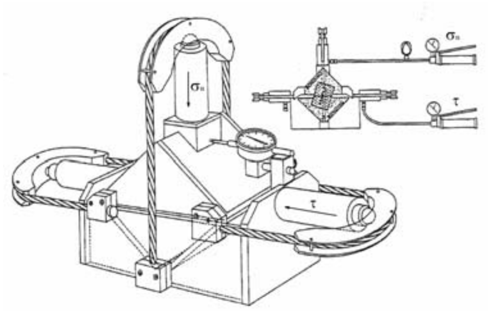
c – kohezija

σ_n - normalni napon

φ - ugao unutrašnjeg trenja

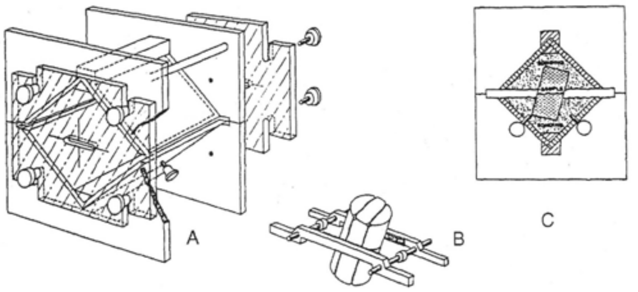
Pripremljeni uzorci škriljca se ubetoniraju u kalupima, sl. 5, i smeste u komoru za smicanje, sl. 3. i sl. 4. Betonirani uzorak škriljca smešten u komoru za smicanje pod pritiskom normalnog opterećenja σn se nalazi u troosnom stanju napona. Ravan po kojoj se izvodi smicanje opterećena je samo smičućim naponom jer je beton po toj ravni prekinut. Pomeranje pri smicanju se prati komparaterom tačnosti 10-2 [mm], a normalno opterećenje se drži konstantnim.

Takođe, postoje predlozi za izmenu pripreme, kao i metode za otkopavanje preostalih rezervi, imajući u vidu specifičnosti zaleganja rudnog tela Tilva Roš u nižim delovima [4], [5].



Sl. 3. *Izgled aparature za smicanje duž željene ravni*

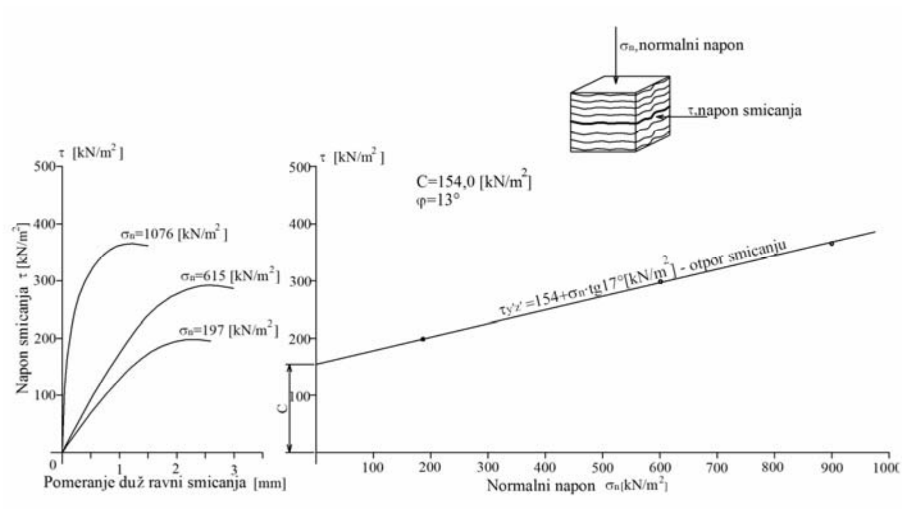
Sl. 4. *Aparatura za smicanje*



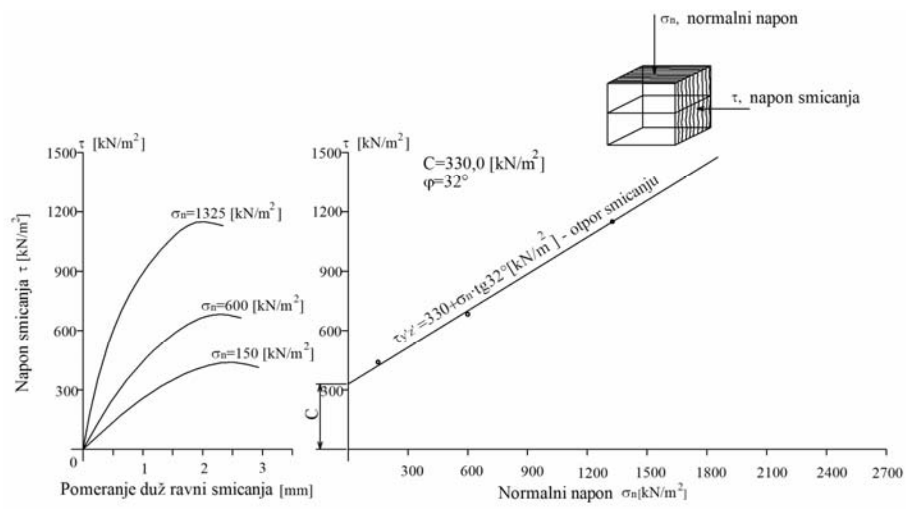
Sl. 5. *Pribor za orijentaciju uzorka i betoniranje*

1.2. Rezultati opita otpora smicanju u transverzalno izotropnoj sredini

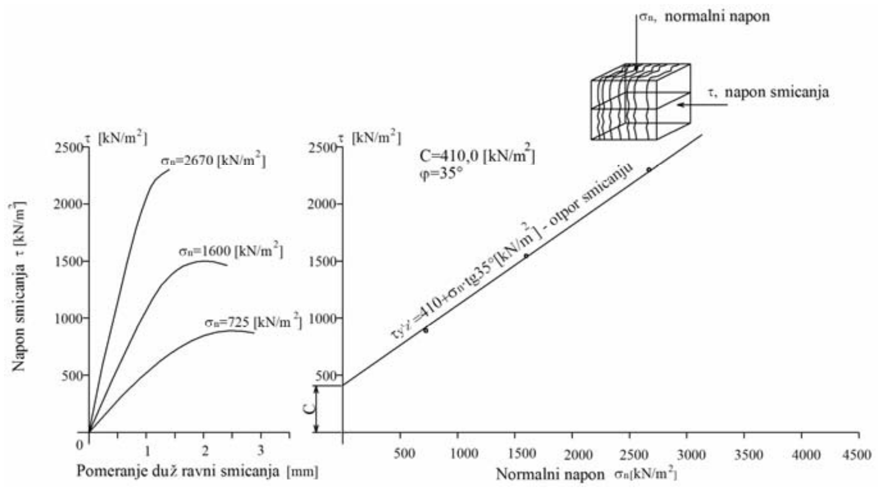
Na po tri uzorka škriljca definisani su parametri otpora smicanju duž tri međusobno normalna pravca. Dobijeni rezultati su prikazani dijagramski na sl. 6., 7. i 8.



Sl. 6. *Smicanje duž ravni škriljavosti* $\tau_{y'z'}$

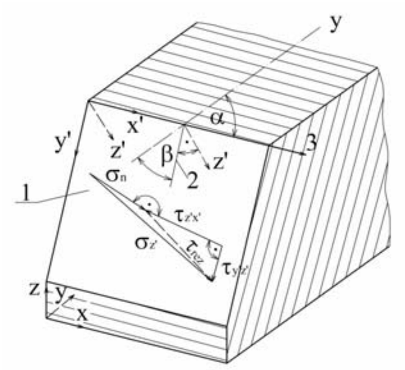


Sl. 7. *Smicanje u ravni* $\tau_{z'x'}$



Sl. 8. *Smicanje u ravni* $\tau_{y'x'}$

Nakon ove nesreće, eksploatacija rude bakra u borskoj Jami ograničena je na rudna tela Brezanik, sa proizvodnjom od oko 15.000 t mesečno i T, sa projektovanom proizvodnjom od 10.000 t rude mesečno. Rudno telo Borska reka još uvek je u fazi otvaranja.



Sl. 9. *Određivanje normalnog* σ_n *i smičućeg* τ*rez. napona*

1 - ravan slojevitosti

2 - pravac pada

3 - pravac pružanja

U slučaju kao što je prikazano na sl. 9, osa z' je normalna na ravan slojevitosti. Veza sa nepokretnim koordinatnim sistemom u prostoru, ugao pružanja je α i pada β, sl. 9.

$l_1 = \sin\alpha$; $m_1 = \cos\alpha$

$l_2 = \cos\beta\cos\alpha$; $m_2 = -\cos\beta\sin\alpha$; $n_2 = -\sin\beta$

$l_3 = -\sin\beta\cos\alpha$; $m_3 = \sin\beta\sin\alpha$; $n_3 = -\cos\beta$

U koordinatnom sistemu x', y' i z', $\tau_{rez.}$ i σ_n se predstavljaju sledećim izrazima:

$$\tau_{rez.} = \sqrt{\tau_{y'z'}^2 + \tau_{z'x'}^2} \quad (2)$$

$$\sigma_n = \sigma_z' \quad (3)$$

Smičući i normalni naponi se dobiju:

$$\begin{aligned}\tau_{y'z'} &= \sigma_x \cdot l_2 \cdot l_3 + \sigma_y \cdot m_2 m_3 + \\ &+ \sigma_z \cdot n_2 \cdot n_3 + \tau_{xy}(l_2 \cdot m_3 + l_3 \cdot m_2) + \\ &+ \tau_{yz}(m_2 \cdot n_3 + m_3 \cdot n_2) + \\ &+ \tau_{zx}(n_2 \cdot l_3 + n_3 \cdot l_2)\end{aligned} \quad (4)$$

$$\begin{aligned}\tau_{z'x'} &= \sigma_x \cdot l_3 \cdot l_1 + \sigma_y \cdot m_3 \cdot m_1 + \\ &+ \tau_{xy}(l_3 \cdot m_3 + l_3 \cdot m_1) + \tau_{yz} \cdot m_1 \cdot l_3 + \\ &+ \tau_{zx} \cdot n_3 \cdot l_1\end{aligned} \quad (5)$$

$$\begin{aligned}\sigma_z' &= \sigma_x \cdot l_3^2 + \sigma_y \cdot m_3^2 + \sigma_z \cdot n_3^2 + \\ &+ 2\tau_{xy} \cdot l_3 m_3 + 2\tau_{yz} \cdot m_3 n_3 + \\ &+ 2\tau_{yx} \cdot n_3 l_3\end{aligned} \quad (6)$$

Pri razmatranju rušenja izazvanog naponom σts tj. naponom normalnim na ravan slojevitosti, a saglasno izrazu 3, primenjuje se sledeći kriterijum rušenja:

$$\sigma_n = -\sigma_{ts} \quad (7)$$

Ovde se mora naglasiti da za primenu numeričkih metoda naponsko-deformacijske analize za odabrani strukturni model (transverzalno izotropni), neophodna su još dodatna ispitivanja, koja bi se obavila na uzorcima reprezentativnog škriljca, a to su definisanje pomeranja u tri uzajamno okomita pravca pri jednoosnom opterećenju i definisanje parametara deformabilnosti u tri uzajamno normalne ravni.

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DEFINING THE SHEAR STRENGTH PARAMETERS OF ROCKS ON PREFERENTIAL PLANES**

Abstract

The shear test results are presented in this work in the elastic, anisotropy environment in three mutually perpendicular directions. The research area is a transversely isotropic model, which is characterized with constant physical-mechanical properties in different directions in a isotropy plane and different properties in the directions perpendicular to the isotropy plane.
Key words: *shear test, anisotropy environment, physical-mechanical properties, plane of isotropy*

1. INTRODUCTION

Considering the diversity of rock mass structure and its properties on which the certain buildings are constructed, it is difficult to develop a model that describes a complexity of material structure and stress-strain state. Such model has to be simple, and the obtained results acceptable.

The first step of investigation in the field of rock mechanics is to develop a structural model that reflects all characteristics of the rock mass structure and their physical-mechanical properties and deformation, Figure 1.

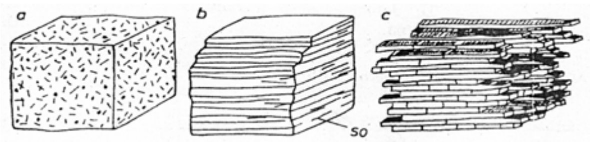

Figure 1. *Structural model*

a – non-oriented isotropic grain structure
b - layered anisotropic structures
c - schistose anisotropic structure

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** *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*

Based on the structural rock model, it is possible to develop such geomechanical model of the rock mass for applying the numerical analysis of finite element method in solving the practical problems in foundation of certain buildings.

Anisotropic elastic model of the rock mass is different from the isotropic elastic model which takes into account the properties of massive change in different directions. In general, [1] anisotropic environment is characterized by 18 independent constants, while, by comparison, an isotropic environment is characterized by only two constants (elastic modulus and the Poisson ratio).

The simplest model of an anisotropic rock mass is a transversely-isotropic [3] (in nature, the closest sample to this model is thin-layered shale).

1. THE SCHISTOSE ANISOTROPIC STRUCTURE FROM A PARTITION PLACE OF THE POWER PLANT "BRODAREVO 2"

Shales from a partition place of the Power Plant "Brodarevo 2" are thin-layered sub-vertical stratifications that are quickly separated in the air and transforms into a dilapidated environment. According to the structural model, it is a schistose anisotropic environment. This environment is characterized by constant physical and mechanical properties in different directions in the plane of isotropy coinciding with the plane x'y'z', Figure 2, and with various properties in the direction of x' axis, which is the axis of symmetry of the elastic environment.

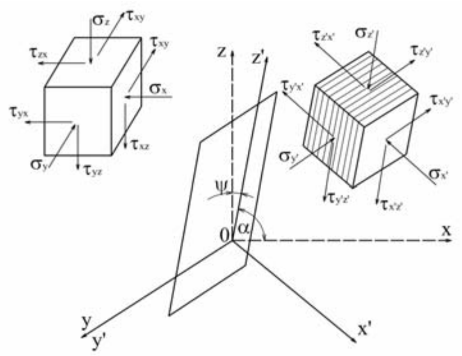

Fig. 2. *Stresses scheme in a transversely-isotropic environment*

1.1. Preparation of Shale Samples for Determination the Shear Resistance Parameters

For defining the parameters of shear resistance of shale (transversely isotropic model) in three mutually perpendicular directions, it is necessary to prepare 9 (nine) samples, size (8x8x8) [cm], using the More-Coulomb law, that the rock destruction happens in a plane of the lowest resistance to shear. Definition of the law is possible with minimum three points:

$$\tau = c + \sigma_n \, tg\varphi \qquad (1)$$

- τ, shear resistance of predefined plane
- C, cohesion
- σ_n, normal stress
- φ, angle of internal friction

The prepared shale samples are put into a wet concrete and then in molds, Figure 5, and placed in a chamber for shear, Figure 3 and 4. Concreted shale sample, placed in a chamber for pressure of normal shear stress σn, is in the triaxial stress state. The plane, along which which the shear stress is done, is loaded only with transverse stresses, as the concrete is interrupted at that level. Movement in shearing is monitored with comparator, accuracy 10-2 [mm], and the normal load is kept constant.

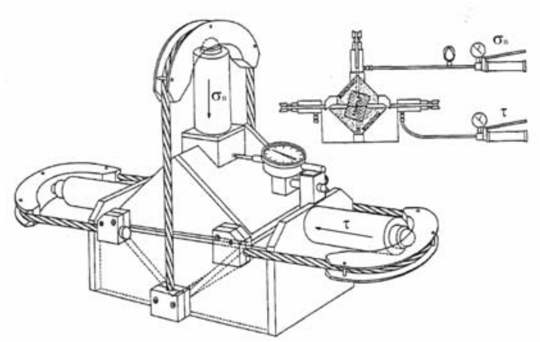

Fig. 3. *Apparatus for shearing along desired plane*

Fig. 4. *Apparatus for shearing*

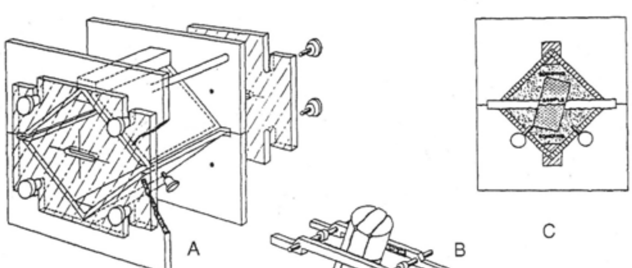

Fig. 5. *Tool for orientation and concrete of samples*

1.2. Results of Shear Resistance Experiment in a Transversely Isotropic Environment

Shearing parameters are defined on three samples of shale along three mutually perpendicular directions. The results are shown by diagrams in Figures 6, 7 and 8.

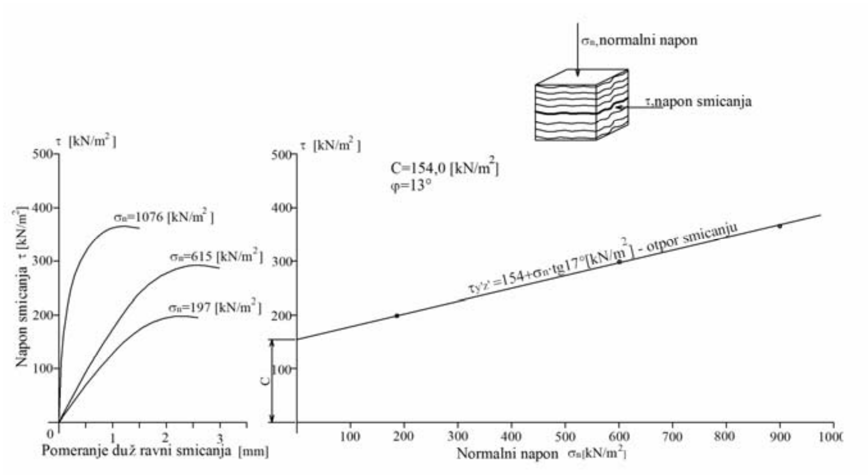

Fig. 6. *Shearing along schistose $\tau_{y'z'}$ plane*

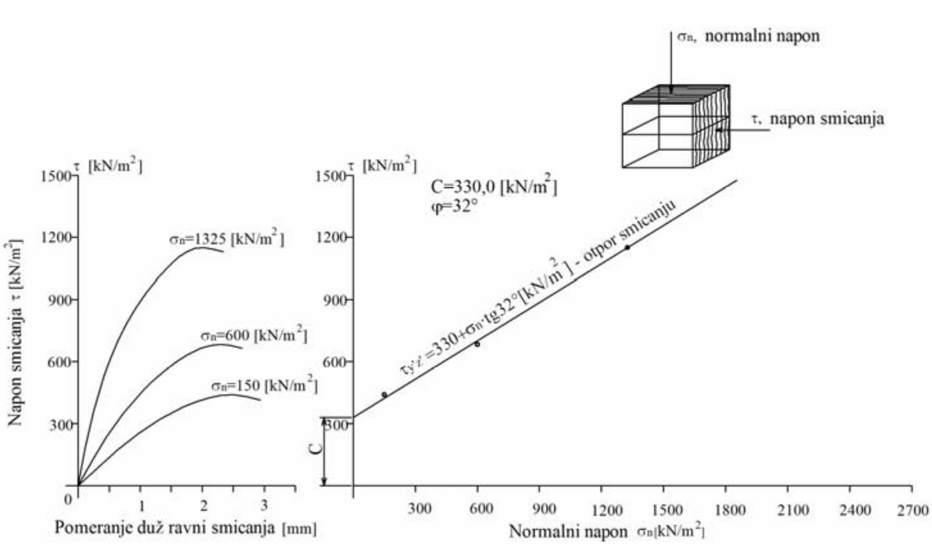

Fig. 7. *Shearing along $\tau_{z'x'}$ plane*

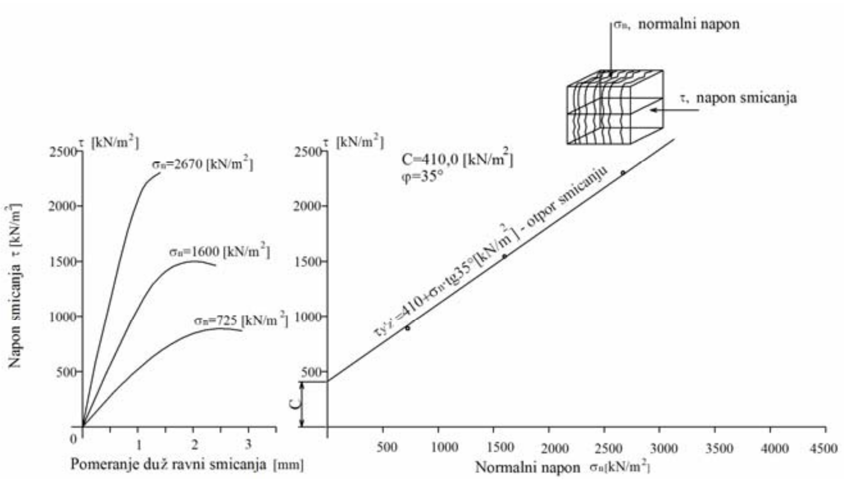

Fig. 8. *Shearing along $\tau_{y'x'}$ plane*

General case of determining the appropriate spatial stress state {σ}, shown in the system of coordinates x, y and z, has to be adapted to the local coordinate system x ', y' and z' and corresponding structure of tested rock, Figure 9.

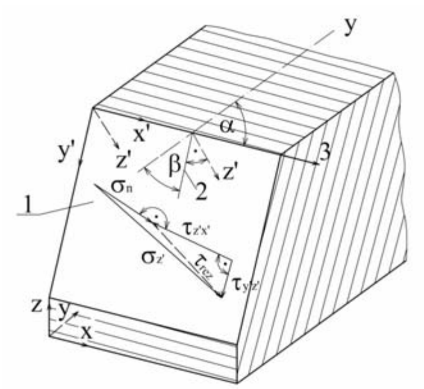

Fig. 9. *Determination of normal σ_n and shear τrez stress*

- 1, plane of stratification
- 2, direction of fall
- 3, strike direction.

In the case, as it shown in Figure 9, the axis z' is perpendicular to the plane of stratification. Connection with the fixed-space coordinate system is: the angle of strike direction α and angle of fall direction β, Figure 9.

$l_1 = \sin\alpha$; $m_1 = \cos\alpha$

$l_2 = \cos\beta\cos\alpha$; $m_2 = -\cos\beta\sin\alpha$; $n_2 = -\sin\beta$

$l_3 = -\sin\beta\cos\alpha$; $m_3 = \sin\beta\sin\alpha$; $n_3 = -\cos\beta$

Stresses τrez. i σn are shown in the coordinate system x', y' i z' by the following expressions:

$$\tau_{rez.} = \sqrt{\tau^2_{y'z'} + \tau^2_{z'x'}} \quad (2)$$

$$\sigma_n = \sigma'_z \quad (3)$$

Shearing and normal stresses are obtained by the following equations:

$$\begin{aligned} \tau_{y'z'} = {} & \sigma_x \cdot l_2 \cdot l_3 + \sigma_y \cdot m_2 m_3 + \\ & + \sigma_z \cdot n_2 \cdot n_3 + \tau_{xy}(l_2 \cdot m_3 + l_3 \cdot m_2) + \\ & + \tau_{yz}(m_2 \cdot n_3 + m_3 \cdot n_2) + \\ & + \tau_{zx}(n_2 \cdot l_3 + n_3 \cdot l_2) \end{aligned} \quad (4)$$

$$\begin{aligned} \tau_{z'x'} = {} & \sigma_x \cdot l_3 \cdot l_1 + \sigma_y \cdot m_3 \cdot m_1 + \\ & + \tau_{xy}(l_3 \cdot m_3 + l_3 \cdot m_1) + \tau_{yz} \cdot m_1 \cdot l_3 + \\ & + \tau_{zx} \cdot n_3 \cdot l_1 \end{aligned} \quad (5)$$

$$\begin{aligned} \sigma'_z = {} & \sigma_x \cdot l_3^2 + \sigma_y \cdot m_3^2 + \sigma_z \cdot n_3^2 + \\ & + 2\tau_{xy} \cdot l_3 m_3 + 2\tau_{yz} \cdot m_3 n_3 + \\ & + 2\tau_{yx} \cdot n_3 l_3 \end{aligned} \quad (6)$$

Considering the destruction caused by the σts stress, i.e. the stress normal on the plane of stratification, according to the expression 3, the following criteria of destruction is applied:

$$\sigma_n = -\sigma_{ts} \quad (7)$$

It must be emphasized that the additional tests are necessary for use the numerical methods for stress-strain analysis of selected structural model (transversely isotropic), which would be carried out on representative samples of shale, and those are defining the movement in three mutually perpendicular directions under uniaxial loading and defining the parameters of deformation in three mutually normal planes.

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RAZMATRANJE MOGUĆNOSTI OTKOPAVANJA PREOSTALIH RUDNIH REZERVI U RUDNOM TELU TILVA ROŠ U JAMI BOR**

Izvod

Otkopavanje u rudnom telu Tilva Roš obustavljeno je 2007. godine usled prodora vode i mulja sa površinskog kopa. Preosrale rudne rezerve procenjene su na 2,5 miliona tona rude. U ovom radu daju se razmatranja mogućnosti otkopavanja ovih rezervi. Rudno telo Tilva Roš je veoma specifično. Iznad njega je stari površinski kop, a ispod njega masivno rudno telo Borska reka, koje se smatra dugoročnom perspektivom podzemne eksploatacije rude bakra u Boru. U slučaju nastavka rudarskih radova, potrebno je posvetiti posebnu pažnju osiguranju podzemnih prostorija i izolaciji rudarskih radova od površinskih voda i mulja.
Ključne reči: *podzemna eksploatacija, borska Jama, sigurnost, prodor vode.*

1. UVOD

Borsko rudno ležište nalazi se u hidrotermalno izmenjenoj zoni sa pravcem pružanja severozapad – jugoistok, zapadno od Borskog raseda. Eksploatacija rude bakra počela je 1903. godine, najpre podzemnim putem, a od 1924. g. i površinskim putem. Između 1934. i 1940. godine podzemnim putem otkopavano je 600.000 do 850.000 tona rude. Podzemna eksploatacija rekordne kapacitete dostiže u drugoj polovini poslednje decenije prošlog veka, kada je godišnja proizvodnja bila blizu 2 miliona tona rude.

Ukupno, do sada je podzemnim putem otkopano oko 50 miliona tona rude i 1,1 miliona tona bakra. [3]

Geološke karakteristike, kao što su dubina i zaleganje ležišta, rezerve rude i sadržaj bakra, bili su odlučujući faktori u strategiji eksploatacije. Površinski kop i jama radili su istovremeno nekoliko decenija. Nakon prestanka rada površinskog kopa, preostale rudne rezerve otkopavane su podzemno, u borskoj Jami. Jama je otvorena izvoznim, servisnim i ventilacionim oknom, uz dva pomoćna ventilaciona okna.

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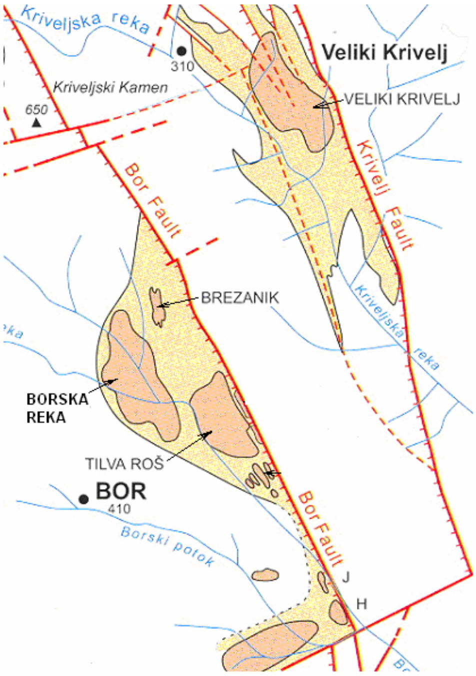
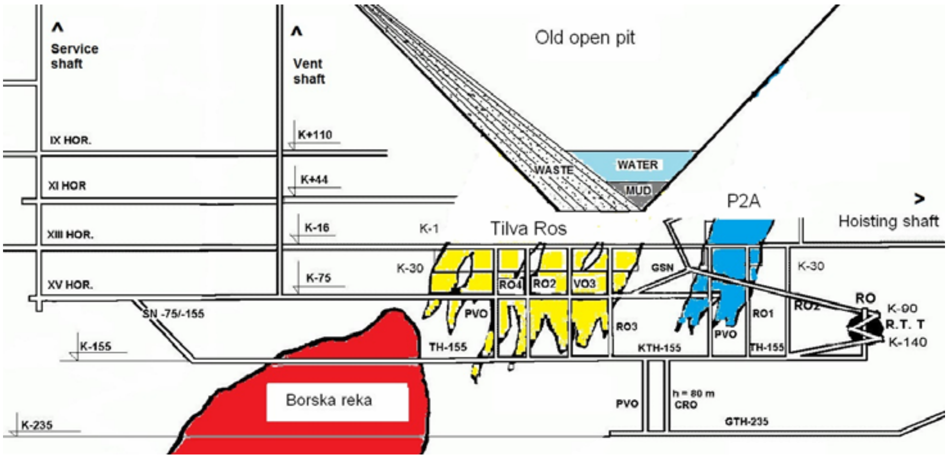
** *Ovaj rad je deo projekta broj TR 33038 koji finansira Ministarstvo za prosvetu i nauku Republike Srbije, a u okviru projekta „Usavršavanje tehnologija eksploatacije i prerade rude bakra sa monitoringom životne i radne sredine u RTB Bor Grupa“*

Borsko ležište čini nekoliko rudnih tela. U prethodnim godinama podzemna eksploatacija uglavnom se oslanjala na rudna tela Tilva Roš i P_2A. Rudno telo Brezanik je takođe aktivno, ali se bliži kraju eksploatacije. Rudno telo T ima relativno male rezerve rude, ali sa izrazito visokim sadržajem bakra, srebra i zlata. Rudno telo Borska reka je masivno rudno telo, sa stotinama miliona tona rudnih rezervi i predstavlja budućnost podzemne eksploatacije na ovim prostorima, ali se nalazi na velikoj dubini i ima relativno nizak sadržaj bakra.



Sl. 1. *Raspored rudnih tela u borskom ležištu*

Podzemna eksploatacija se obavlja primenom različitih metoda otkopavanja. U rudnim telima Tilva Roš i P_2A primenjivana je švedska varijanta podetažnog zarušavanja, u rudnom telu Brezanik komorno – stubno otkopavanje sa zapunjavanjem otkopa, a u rudnom telu T primenjuje se NATM (New Austrian Tunneling Method) metoda. Otkopana ruda transportuje se do rudnih okana, kojima se spušta na nivo XVII horizonta (K-155 m). Na XVII horizontu, ruda se iz rudnih okana toči u jamske kamione, kojima se transportuje do postrojenja za primarno drobljenje. Nakon drobljenja, ruda se centralnim rudnim oknom (CRO) spušta na nivo XIX horizonta (K-235 m), odakle se dalji transport vrši transporterima sa trakom. Na ovaj način ruda se kroz glavni transportni hodnik na nivou XIX horizontal i glavni transportni niskop doprema do bunkera izvoznog okna, na nivou XIII horizonta (K-21 m), odakle se vrši izvoz rude skipovima, nakon čega ruda ide na dalju preradu.



Sl. 2. *Šematski prikaz rudnih tela i prostorija u borskoj Jami*

2. RUDNO TELO TILVA ROŠ

Tilva Roš je drugo po veličini rudno telo u borskom ležištu. Dužina mu je 800 m, širina 150 – 250 m, a visina 350 m. Javljaju se dve osnovne vrste mineralizacije – masivna sulfidna ruda u gornjim delovima i štokverkno – impregnaciona u nižim delovima rudnog tela. Sa porastom dubine, rudno telo se deli u manje delove, nalik korenu drveta (slika 2). Delovi rudnog tela bliži površini terena otkopani su površinskim putem. Glavni rudni minerali su kovelin i enargit. Sadržaj bakra u rudi kreće se od 0,65 do 0,8 %.

Preostale rudne reserve procenjuju se na oko 2,5 miliona tona. U trenutku prestanka radova, otkopavanje se odvijalo na nivou K-31 m, dok je nivo K-46 m bio u pripremi. Na ova dva nivoa ostalo je oko 1,1 miliona tona neotkopane rude. Za otkopavanje nižih nivoa, K-61 m i K-76 m, tek je potrebno izraditi odgovarajuću dokumentaciju. [3]

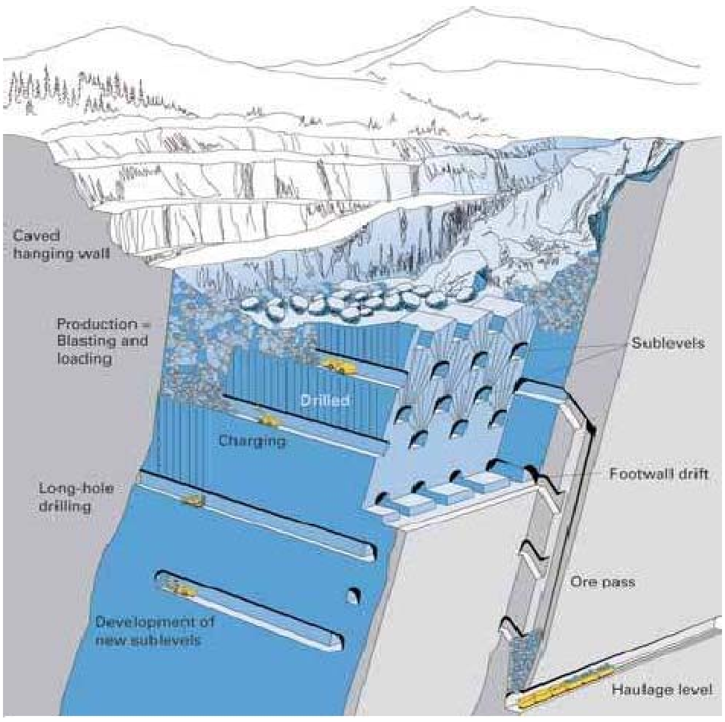
U rudnom telu Tilva Roš primenjuje se švedska metoda podetažnog zarušavanja, sa sledećom geometrijom:

H = 30 m - dužina bušenja,
h = 15 m - visina podetaže,
S = 14 m - rastojanje između podetažnih hodnika, D = 4 m- širina podetažnog hodnika,
V = 3,5 m - visina podetažnog hodnika,
P = 12,7 m^2 - površina podetažnog hodnika,
α = 3 ‰ - pad podetažnog hodnika,
β = 90° - ugao površine miniranja.

U svakoj lepezi buši se 6 – 8 minskih bušotina, a razmak između lepeza je 2 m. Ukupna dužina bušotina u jednoj lepezi je 100 – 120 m. Miniranjem svake lepeze dobija se oko 1200 t rude.

Postavlja se pitanje da li je ova metoda pogodna za primenu i u nižim delovima rudnog tela, gde je njegov oblik prilično nepravilan. Neki autori smatraju da je komorno – stubno otkopavanje sa zapunjavanjem otkopa bolje rešenje, obzirom da je ta metoda fleksibilnija i omogućila bi bolje prilagođavanje geometriji rudnog tela.

Takođe, postoje predlozi za izmenu pripreme, kao i metode za otkopavanje preostalih rezervi, imajući u vidu specifičnosti zaleganja rudnog tela Tilva Roš u nižim delovima [4], [5].



Sl. 3. *Metoda podetažnog zarušavanja rude i pratećih stena [2]*

3. PRODOR VODE I MULJA U RUDARSKE RADOVE

Teška rudarska nesreća dogodila se 2007. godine, kada je došlo do masovnog prodora vode i mulja sa dna površinskog kopa u podzemne rudarske radove, kada je jedan rudar nastradao, a eksploatacija u rudnim telima Tilva Roš i P_2A prekinuta. Analizom ovog događaja utvrđeno je das u se voda i mulj akumulirali na dnu starog površinskog kopa, koji se nalazi iznad podzemnih rudarskih radova. Obzirom na činjenicu da se u rudnim telima Tilva Roš i P_2A primenjivala metoda sa zarušavanjem rude i krovinskih stena, proces zarušavanja uzrokovao oštećenje vodonepropusnog sloja, čime je vodi i mulju omogućeno da kroz zarušene krovinske stene uđu u podzemne otkope i rudničke prostorije.

Sl. 4. *Akumulacija vode i mulja iznad rudnog tela Tilva Roš [3]*

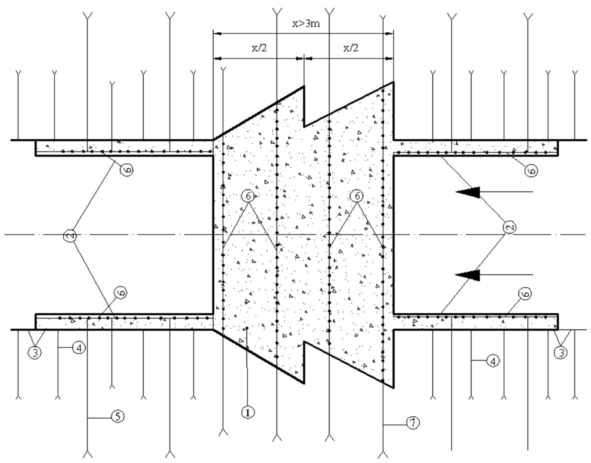
Nakon ove nesreće, eksploatacija rude bakra u borskoj Jami ograničena je na rudna tela Brezanik, sa proizvodnjom od oko 15.000 t mesečno i T, sa projektovanom proizvodnjom od 10.000 t rude mesečno. Rudno telo Borska reka još uvek je u fazi otvaranja.

4. MOGUĆNOSTI OBNAVLJANJA EKSPLOATACIJE

Nakon 2007. godine, urađeno je nekoliko studija vezanih za mogućnosti nastavka eksploatacije u rudnim telima Tilva Roš i P_2A. Zaključeno je da je za nastavak radova potrebno ispuniti dva preduslova: izolacija rudarskih radova izradom određenog broja betonskih baraža i uklanjanje akumulacije vode i mulja sa dna površinskog kopa. Tek nakon ispunjenja ta dva zahteva, moguće je dalje razmatranje nastavka otkopavanja. [3]

Akumulacija vode i mulja na dnu površinskog kopa predstavlja stalnu pretnju za podzemne rudarske radove, naročito u slučaju primene metoda sa zarušavanjem rude i krovinskih stena. Zbog toga je neophodno ukloniti akumulaciju, a zatim i preduzeti korake da se spreči njen ponovni nastanak.

Utvrđeno je da je potrebno ugraditi 27 betonskih baraža radi potpune izolacije rudarskih radova na različitim nivoima u jami.



Sl. 5. *Betonska baraža u jamskoj prostoriji [3]*

5. ZAKLJUČAK

Otkopavanje u rudnim telima Tilva Roš i P_2A zaustavljeno je 2007. godine usled prodora vode i mulja sa dna površinskog kopa u jamu. Obzirom da su u ovim rudnim telima ostale značajne reserve rude, razmatraju se mogućnosti za obnavljanje proizvodnje. Analizama je utvrđeno da je nastavak otkopavanja moguć samo u slučaju potpune izolacije podzemnih prostorija od priliva vode ugradnjom većeg broja betonskih baraža i uklanjanjem vodene akumulacije na dnu površinskog kopa koji se nalazi iznad rudnog tela. Vrednost ovih radova procenjena je na 1,2 miliona US$. [3]

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UDK: 622.272.5(045)=20

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CONSIDERATIONS THE POSSIBILITIES FOR MINING THE REMAINING ORE RESERVES IN THE ORE BODY "TILVA ROS" IN THE JAMA BOR MINE**

Abstract

Mining works in the ore body „Tilva Ros" were suspended in 2007, due to the water and mud irruption from the old open pit. The remaining ore reserves are estimated to 2.5 Mt. Possibilities for excavation of these reserves are considered in this paper. The ore body „Tilva Ros" is a very specific. It is situated bellow the old open pit and above massive ore body „Borska reka", which is considered as a long-term perspective of underground mining in this area. It is necessary to pay a special attention to safety of underground works and their isolation from the surface water and mud.

Key words: *underground mining, mine safety, irruption of water*

1. INTRODUCTION

The Bor copper ore deposit is situated in the hydrothermally altered zone, with northwest – southeast strike direction, west from the Bor fault. Copper ore mining has started in 1903, first by underground mining, and since 1924 by both surface and underground mining. In a period between 1934 and 1940, the underground production reached 600,000 to 850,000 tons of ore. The underground mining reached its maximum production in late 90's, with near 2 Mt of ore per year. In total, some 50 Mt of ore and 1.1 Mt of copper have been excavated by underground mining so far. [1]

Geological properties, such as the deposit depth and inclination, ore reserves and grades, were the most important factor in mine designing. The open pit and underground mine worked simultaneously for several decades. After termination of surface mining, the remaining ore reserves were excavated from the underground mine Jama Bor. This mine was opened by hoisting shaft, service shaft, main ventilation shaft and two auxiliary ventilation shafts.

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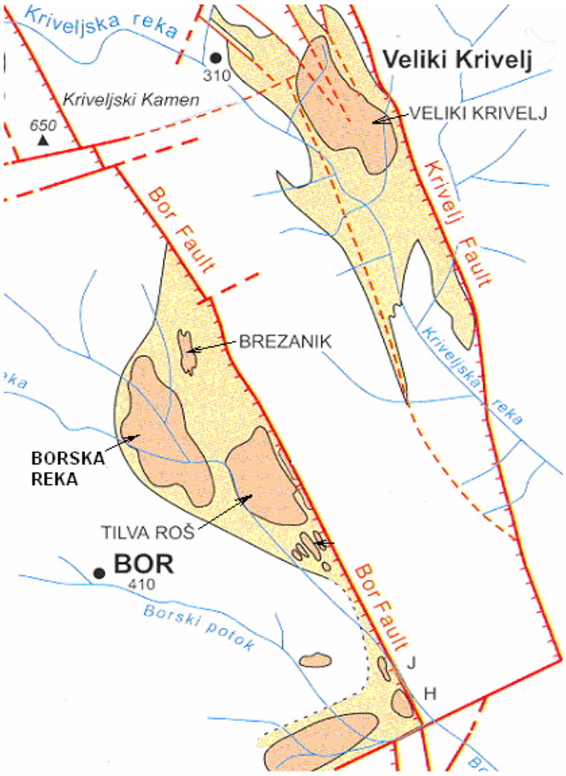

Fig. 1. *Horizontal layout of ore bodies in the Bor deposit*

There are several ore bodies in the Bor deposit. Copper ore production in recent years mainly relied on the ore bodies Tilva Ros and P_2A. The ore body Brezanik is also active, but close to being mined out. The ore body T is relatively small, but very rich ore body with high grades of copper, gold and silver. The ore body Borska reka is very massive, with hundreds million tons of ore reserves and represents the future of underground copper ore mining in this area, although it is situated at significant depths and ore grades are relatively low.

Production process includes different mining methods. Sublevel caving was used in the Tilva Ros and P_2A, cut and fill in the ore body Brezanik and NATM (New Austrian Tunneling Method) in the ore body T. Excavated ore is transported to the ore passes, where it is drawn to a transport level at Horizon XVII (K -155 m). At this level, trucks loaded at ore passes transport the ore to the crusher. After crushing, ore goes into the central ore pass (CRO) and, at Horizon XIX (K - 235 m), is loaded on a belt conveyer, which transports the ore to the bin at Horizon XIII (K -21 m), near the hoisting shaft. From the bin, feeders loads the ore into skips and through hoisting shaft the ore comes out on the surface and goes to the mineral processing plant.

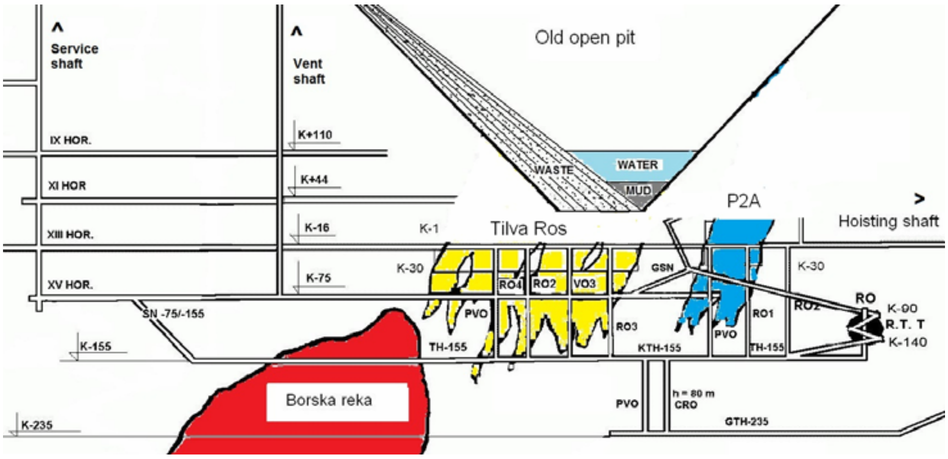

Fig. 2. *Scheme of the Jama Bor underground mine*

2. THE ORE BODY TILVA ROS

Tilva Ros is the second largest ore body in the Bor deposit. It is 800 m long, 150-250 m wide and 350 m deep. There are two major types of mineralization – massive sulphide type at higher levels, and stockwork – impregnated type in the lower sections. As the ore body depth increases, it splits into smaller sections, similar to a tree root (Figure 2). Parts of the ore body closer to the ground surface were mined by surface mining. The main ore minerals are covelline and enargite. Ore grades vary from 0.65 to 0.8 % Cu.

The remaining ore reserves are estimated to 2.5 Mt. At the moment of irruption, the production was at K – 31m level, with K – 46m level in development. There are 1.1 Mt of ore left at these two levels. It is necessary to develop a suitable documentation for mining the lower levels, K - 61m and K – 76m. [3]

The underground mining of the ore body Tilva Ros was carried out using the Swedish variant of sublevel caving. Stope geometry is the following:

H = 30 m - drilling length,

h = 15 m - sublevel height,

S = 14 m - spacing between sublevel drifts,

D = 4 m- sublevel drift width,

V = 3,5 m- sublevel drift height,

P = 12.7 m^2 - sublevel drift area,

α = 3 ‰ - sublevel drift inclination,

β = 90° - angle of blasting plane.

There are 6-8 drill holes in a single ring, and spacing between rings is 2 m. Overall drill hole length is 100-120 m. Blasting of each ring provides 1200 t of ore.

There are considerations whether this mining method is suitable for lower parts of the ore body, regarding to its irregular shape in this area. Some authors suggest a variant of cut and fill mining, which is more flexible and enables adjustments required by the ore body geometry.

Also, in some previous papers, [4], [5], there were several suggestions for changing the development and excavation design in lower parts of the ore body Tilva Ros, due to its specific layout.

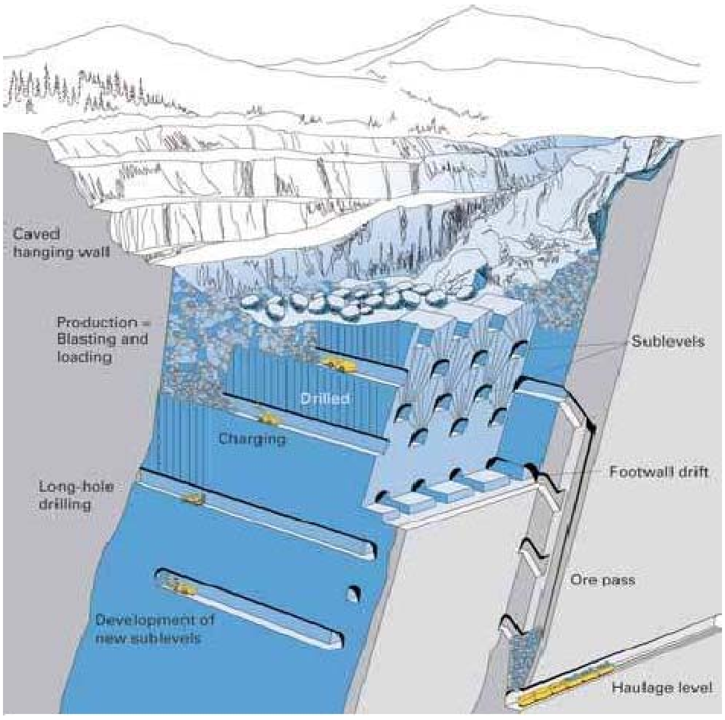

Fig. 3. *Principle of sublevel caving [2]*

3. IRRUPTION OF WATER AND MUD

In 2007, a serious accident happened in the Jama Bor underground mine. Massive breakthrough of water and mud flooded some parts of the mine when a miner died, and, as a consequence, mining in the ore bodies Tilva Ros and P_2A was suspended. The analyses of this accident have shown that water and mud were accumulated on the bottom of the old Bor Open Pit, which is situated above the underground works. Considering the fact that sublevel caving is applied in the ore bodies Tilva Ros and P_2A, caving process caused water-tight layer of rock to disintegrate, thus enabling huge amount of mud and water to pass through the broken rock in the roof and enter into stopes and mine corridors.

Fig. 4. *Water accumulation above the ore body Tilva Ros [3]*

After this accident, the copper ore mining in the Jama Bor was limited to the ore bodies Brezanik, with production of 15,000 t and T, with 10,000 t of ore monthly output. The ore body Borska reka is still in a development stage.

4. RENEWAL OF THE MINING WORKS

After 2007, there were several studies which considered the situation in the ore bodies Tilva Ros and P_2A and possibilities for renewal of mining works. Conclusions suggested that there are two major prerequisites: the water isolation of mining works by number of concrete barrages at several underground levels, and removal of water accumulation on the bottom of the old open pit. Only if these two prerequisites are fulfilled, it would be safe to continue mining works in this area. [3]

Water accumulation on the bottom of the old open pit is a constant threat for underground works, especially when the caving mining methods are used. Caving may cause dislocation of the waterproof seam bellow the accumulation and cause catastrophic consequences in the underground mine. That is why this water accumulation has to be removed and possibility for its re-forming have to be prevented.

Analyses have shown that total of 27 water barrages should be installed in the underground drifts of the Jama Bor, in order to prevent irruption of water, mud and other materials from the ground surface and abandoned mining works. Barrages should be made of reinforced concrete.

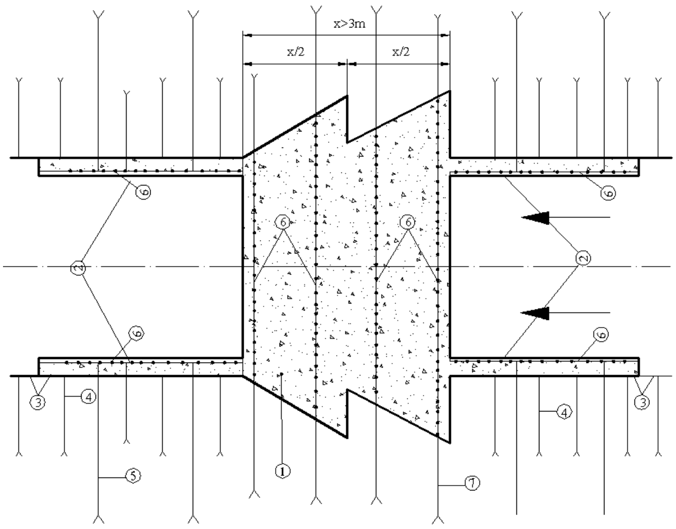

Fig. 5. *Concrete barrage [3]*

5. CONCLUSION

Mining works in the ore bodies Tilva Ros and P_2A were suspended in 2007, due to the irruption of water and mud. Since the significant ore reserves are left in this area, the possibilities for renewal of mining works were taken into consideration. Most of the analyses have shown that it is possible only in a case of complete water isolation of that underground area by numerous barrages and removal of water accumulation on the bottom of the old open pit above the mining works. The value of these works is estimated to 1.2 million US$. [3]

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NOVI TEHNOLOŠKI POSTUPAK ZA ODRŽIVU PRERADU RUDARSKOG TEHNOGENOG OTPADA**

Izvod

Rezultati istraživanja pokazuju, tehnološku mogućnost i ekonomsku opravdanost zajedničke prerade dve tehnogene otpadne sirovine koje nastaju u procesima prerade rude bakra. Šljaka plamene peći Topionice bakra u Boru i flotacijska jalovina Rudnika bakra Bor, Srbija su dve tehnogene otpadne sirovine skladirane na deponijama i u permanentnom nastajanju. One predstavljaju velike zagađivače životne sredine, a shodno sadržaju korisnih komponenti u istim, i potencijalne sirovinske resurse. Zbog specifičnih fizičko-hemijskih i mineraloških karakteristika pojedinačna prerada ovih otpadnih sirovina ne garantuje profitabilnu delatnost. Zajedničku preradu topioničke šljake i stare flotacijske jalovine u masenom odnosu 90 % šljake i 10 % flotacijske jalovine, u originalnom tehnološkom postupku povećava se iskorišćenje bakra u procesu flotiranja za oko 15 %, istovremeno povećava efikasnost procesa mlevenja i klasiranja, time smanjuje potrošnju energije za 10-15 %, čineći da prerada ovih otpadnih sirovina, pored ekološkog, ima i pozitivan ekonomski efekat.

Ključne reči: *topionička šljaka, flotacijska jalovina, tehnogeni otpadi, flotacija, iskorišćenje bakra*

UVOD

Rudarsko-topioničarski basen Bor, u složenim tehnologijama proizvodnje bakra, od eksploatacije do elektrolitičkog prečišćavanja, produkuje nekolino desetina različitih, zagađujućih, tehnogenih otpadnih sirovina [1]. Veći broj istih, u gasovitom, tečnom ili čvrstom stanju, sadrže značajne koncentracije korisnih komponenata, koje se, uz primenu savremenih tehnologija mogu valorizovati. Rezultati istraživanja

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prerade topioničke šljake, najzastupljenijeg tehnogenog otpada u metalurgiji bakra, upućuju na tehnološku mogućnost prerade [2,3]. Zbog povećane otpornosti prema usitnjavanju, povećane gustine i odsustva aluminata [4-7], pulpa formirana od topioničke šljake poseduje smanjenu viskoznost i stabilitet. Ove karakteristike topioničke šljake i pulpe formirane od iste, uslovljavaju brzu sedimentaciju čvrste faze, što negativno utiče na efikasnost najznačajnijih procesa prerade, mlevenje i klasiranje, kao i proces flotacijske koncentracije. Povećana potrošnja energije u procesu mlevenja, odnosno povećanje troškova prerade i smanjeno iskorišćenje korisnih komponenti u procesu flotiranja, dovode u pitanje ekonomičnosti prerade ovog tehnogenog otpada.

Istraživanja poboljšanja viskoziteta i stabilnosti pulpe, dodavanjem flotacijske jalovine u procesu mlevenja šljake trebalo bi da daju pozitivne rezultate. Primena flotacijske jalovine, najzastupljenijeg tehnogenog otpada i najvećeg zagađivača životne sredine u procesima prerade rude bakra, će poboljšati karakteristike pulpe, a time i efikasnost najznačajnijih faza procesa. Zahvaljujući manjoj gustini čvrste faze flotacijske jalovine, kao i prisustvu finozrnih aluminata i alumosilikata, pulpa formirana od definisanog masenog odnosa navedenih otpadnih sirovina, ima povećanu viskoznost i stabilitet što uslovljava veću efikasnost procesa mlevenja i klasiranja, odnosno manju potrošnju energije u procesu prerade, kao i povećano iskorišćenje korisnih komponenti u procesu flotiranja. Povećano iskorišćenje bakra iz mešavine topioničke šljake i flotacijske jalovine moglo bi biti rezultat kako većeg iskorišćenja bakra iz šljake zbog boljih hidrodinamičkih uslova flotiranja čestica veće krupnoće i gustine, tako i dodatnog iskorišćenja bakra sadržanog u flotacijskoj jalovini. Veća viskoznost pulpe pozitivno utiče na efikasnost mlevenja u mlinu sa kuglama, zbog boljeg prijanjanja sirovine za meljuća tela, a povećana efikasnost hidraulične klasifikacije u hidrociklonu, kao posledica veće stabilnosti pulpe smanjuje nepotrebno vraćanje dovoljno usitnjenih čestica za proces flotiranja, kroz pesak hidrociklona na ponovno mlevenje u mlinu sa kuglama.

Smanjenje cirkulativne šarže do optimalne granice, odnosno smanjeno učešće dovoljno usitnjenim česticama u pesku hidrociklona, ima višestruki tehno-ekonomski značaj. Pored povećanja kapaciteta i smanjenja potrošnje energije u procesu mlevenja, kao najvećeg troška u procesu prerade, manje preusitnjavanje čestica korisnih komponenti uslovljava veće iskorišćenje istih u procesu flotiranja.

Pozitivni efekti primene nove tehnologije, zajedničke prerade topioničke šljake i flotacijske jalovine, izraženi kroz smanjenje troškova prerade i povećano iskorišćenje korisnih komponenti, omogućavaju profitabilnu delatnost i ekonomsku održivost zaštite životne sredine.

EKSPERIMENTALNI DEO

Eksperimentalna istraživanja su sadržana u dve faze. U prvoj fazi su vršena laboratorijska ispitivanja karakteristika topioničke šljake i flotacijske jalovine i pulpe formirane od istih, značajnih za predviđene postupke prerade. U drugoj fazi eksperimenata izvedeni su paralelni opiti flotacijske koncentracije bakra iz topioničke šljake, i mešavine šljake i flotacijske jalovine, čiji rezultati treba da potvrde predhodna teoretska razmatranja o tehnološkoj opravdanosti zajedničke prerade ovih tehnogenih otpadnih sirovina.

Sva laboratorijska ispitivanja vršena su u akreditovanim i licenciranim laboratorijama Tehničkog fakulteta u Boru i Institutu za rudarstvo i metalurgiju Bor.

Mikroskopska i mineraloška ispitivanja vršena su na mikroskopu tipa Carl Zeiss-Jena "JENAPOL-U".

Hemijske analize uzoraka vršene su primenom različitih analitičkih metoda (tabela 2).

Karakteristike topioničke šljake

Za odabrani proces koncentracije bakra iz topioničke šljake, flotacijsku koncentraciju, od posebnog značaja su mineraloški i hemijski sastav sirovine. U tabelama 1. i 2. prikazane su ove karakteristike šljake.

Tabela 1. *Mineraloški sastav topioničke šljake*

Minerali	Zastupljenost (%)
Fajalit	60,00
Magnetit	30,00
Bornit	6,50
Halkopirit	1,50
Elem.bakar	0,50
Bakrenac	0,30
Hrizokola	0,15
Kovelin	0,05
Pirit	1,00
$\sum$	100,00

Tabela 2. *Hemijski sastav topioničke šljake*

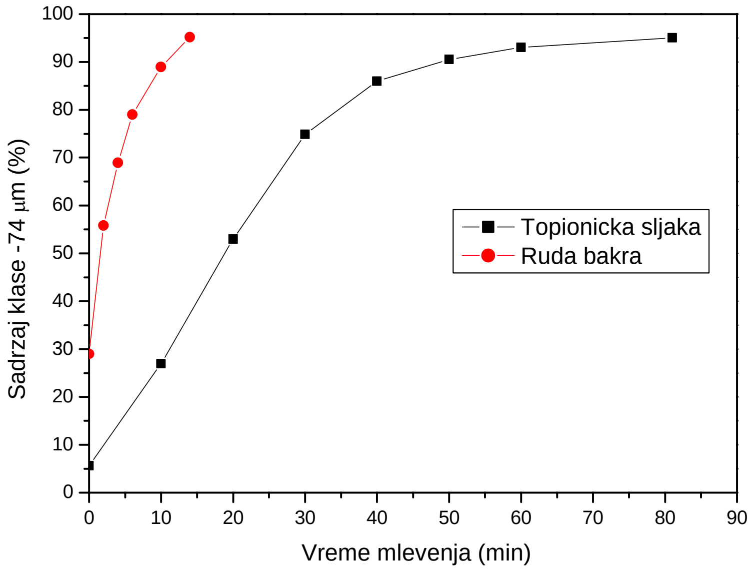
Elementi	Sadržaj (%)	Analitička metoda
$Fe_{uk.}$	37,05	VT
SiO_2	32,77	G
Fe_3O_4	9,68	A-Fe 304
Al_2O_3	4,86	ICP-AES
Fe u Fe_2O_3	3,02	R
S	0,81	G
Ca	3,21	AAS
Mg	0,62	AAS
Cu_{uk}	0,84	SF
Cu_{elem}	0,38	G
Cu_{sul}	0,30	R
Cu_{ox}	0,16	PO
Ag	3,3 g/t	FA
Au	0,5 g/t	FA

Mineraloški i hemijski sastav topioničke šljake potvrđuju mogućnost primene flotacijske koncentracije kao metode za koncentraciju korisnih komponenti. Sadržaj bakra od 0,84 % i plemenitih metala, zlata od 0,5 g/t i srebra od 3,3 g/t, povrđuje konstataciju da ovaj tehnogeni otpad sadrži 2-3 puta veću koncentraciju navedenih korisnih komponenti od sadržaja istih u primarnim rudama bakra, oko 0,25 - 0,35 % Cu, koje se sada eksploatišu. Ova činjenica upućuje na zaključak da ovaj tehnogeni otpad predstavlja sirovinski resurs i opravdava istraživanja mogućnosti prerade istog.

Gustina topioničke šljake određena staklenim piknometrom iznosi $\rho_č \approx 3500$ kg/m^3. Sama činjenica da je gustina ovog tehnogenog otpada za oko 500 – 700 kg/m^3 veća od gustine rude bakra upućuje na zaključak o većoj brzini sedimentacije čestica šljake u pulpi u odnosu na brzinu sedimentacije čestica rude bakra iste krupnoće.

Meljivost, odnosno otpornost prema usitnjavanju, kinetika mlevenja i garnulometrijski sastav su fizičke karakteristike sirovine od kojih u najvećoj meri zavise troškovi prerade iste.

Bond-ov indeks meljivosti u mlinu sa kuglama za topioničku šljaku iznosi W_i= 31 kWh/t, što je oko 2 puta veća vrednost od istog za rudu bakra. Ova činjenica upućuje na očekivanu veliku potrošnju energije u procesu usitnjavanja. Na slici 1, dijagramima su prikazane kinetike mlevenja topioničke šljake i rude bakra, gde se jasno uočava višestruko duže vreme mlevenja topioničke šljake u odnosu na rudu bakra za postizanje iste finoće proizvoda mlevenja.



Sl. 1. *Efikasnost mlevenja topioničke šljake i rude bakra*

Rezultati istraživanja nedvosmisleno ukazuju na veliku otpornost usitnjavanja ovog tehnogenog otpada, što povrđuje predhodnu konstataciju o velikoj potrošnji energije u procesu usitnjavanja, a time i povećanim troškovima prerade.

Karakteristike flotacijske jalovine

Istraživanja su vršena na "staroj" flotaijskoj jalovini produkovanoj u procesu flotiranja rude bakra u prvoj polovini prošlog veka. Za ovu tehnogenu otpadnu sirovinu smo se opredelili s obzirom na povećani sadržaj korisnih komponenti u istoj, 0,15 – 0,4% Cu, uslovljen visokim sadržajem bakra u rudi koje je u tom periodu prerađivana oko 2,5-3% Cu, kao i efikasnošću tada primenjivane tehnologije flotacijske koncentracije.

Mineraloški sastavi flotacijske jalovine određeni na uzorcima iz različitih dubina starog flotacijskog jalovišta prikazani su u tabeli 3, a hemijski sastavi jalovine određeni na uzorcima sa različitih, za flotaciju karakterističnih delova deponije, prikazani su u tabeli 4.

Tabela 3. *Mineraloški sastavi uzoraka stare flotacijske jalovine*

Minerali	Zastupljenost (%)	
	Uzorak 1 (0-16 m)	Uzorak (16-40m)
Halkopirit	0,16	0,12
Kovelin	0,03	0,11
Energit	0,01	0,02
Halkozin	0,01	0,01
Bornit	U tragu	0,01
Azurit	U tragu	0,03
Kuprit	0,01	0,04
Pirit	14,27	22,20
Minerali jalovine	80,63	71,96
Ostalo	4,88	5,50
Σ	100,00	100,00

Tabela 4. *Hemijski sastavi uzoraka stare flotacijske jalovine [3]*

Kom-ponenta	Brana jalovišta	Unutrašnjost jalovišta	Analit. metoda
	Zastupljenost (%) (g/t)*	Zastupljenost (%) (g/t)*	
Cu_{uk}	0,155	0,370	
Cu_{ox}	0,033	0,120	
$Cu_{sulf.}$	0,122	0,250	
S	9,870	12,260	
Fe	8,860	10,270	
SiO_2	58,030	59,710	
Al_2O_3	12,040	12,630	
Au	0,300*	0,600	
Ag	2,140*	1,300	

Rezultati istraživanja pokazuju da je u jalovini sadržana značajna koncentracija korisnih komponenti u mineraloškom obliku pogodnom za primenu procesa flotiranja.

Međutim, shodno osnovnom razlogu korišćenja jalovine u procesu prerade topioničke šljake, a ista je poboljšanje karakteristika pulpe, odnosno povećanje viskoznosti i stabiliteta do optimalnih vrednosti, od većeg značaja su fizičke karakteristike ovog tehnogenog otpada.

Gustina flotacijske jalovine određena je na većem broju uzoraka uzorkovanim na različitim mestima flotacijskog jalovišta. Potreba uzimanja uzoraka sa više različitih mesta flotacijskog jalovišta, od brane do plaže istog, uslovljena je načinom formiranja flotacijskog jalovišta i distribucije peska i preliva hidrociklona. Vrednosti gustine pojedinih uzoraka kao i srednja gustina flotacijske jalovine koja je korišćena u daljim istraživanjima date su u tabeli 5.

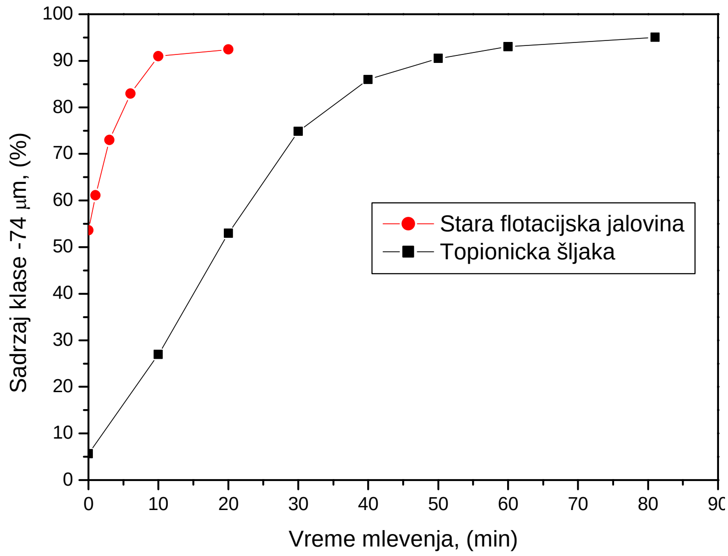
Tabela 5. *Vrednosti gustine uzoraka stare flotacijske jalovine [3]*

Flotacijska jalovina	Uzorak 1 (brana jalovišta)	Uzorak 2 (centralni deo jalovišta)	Uzorak 3 (obodni deo jalovišta)	Kompozitni uzorak
ρ (kg/m^3)	2 650	3 070	2 814	2 844

Znatno manja gustina flotacijske jalovine od gustine topioničke šljake uslovljena je različitim mineraloškim i hemijskim sastavom istih. Dok u topioničkoj šljaci dominantno učešće imaju fero-silikati većih gustina, cca 3700-3900 kg/m^3, dotle u flotacijskoj jalovini najveće učešće imaju alumo-silikati znatno manjih gustina, oko 2500-2600 kg/m^3.

Otpornost prema usitnjavanju finozrnog materijala flotacijske jalovine nije moguće određivati standardnom Bond-ovom metodom, međutim, ista se može poistovetiti sa otpornošću prema usitnjavanja primarne rude bakra, koja je za prerađivanu rudu u tom periodu iznosila od 10 – 15 kWh/t.

Shodno praktičnoj primeni rezultata istraživanja, a ista je zajednička prerada topioničke šljake i flotacijske jalovine, koja započinje mlevenjem mešavine, na slici 2, dijagramima su prikazane uporedne vrednosti kinetike mlevenja ova dva tehnogena otpada sa realnim – industrijskim početnim krupnoćama materijala.



Sl. 2. *Efikasnost mlevenja topioničke šljake i stare flotacijske jalovine realnih početnih krupnoća*

Znatno kraće vreme potrebno za ostvarivanje određene finoće proizvoda mlevenja flotacijske jalovine u odnosu na vreme potrebno za ostvarivanje iste finoće mlevenja topioničke šljake, upućuju na logičan zaključak o očekivanoj boljoj meljivosti mešavine ova dva tehnogena otpada u odnosu na meljivost čiste topioničke šljake.

EKSPERIMENTI MLEVENJA TOPIONIČKE ŠLJAKE I MEŠAVINE TOPIONIČKE ŠLJAKE I FLOTACIJSKE JALOVINE

Istraživanja meljivosti sirovina izvršena su postupkom mokrog mlevenja u mlinu sa kuglama. U prvoj fazi istraživanja izvedene su tri serije laboratorijskih opita, na topioničkoj šljaci i mešavini topioničke šljake i stare flotacijske jalovine u određenim masenim odnosima, sa ciljem definisanja efikasnosti procesa mlevenja pojedinih uzoraka.

Druga faza istraživanja odnosila se na ispitivanje uticaja gustine pulpe, odnosno odnosa Č:T, na efikasnost procesa mlevenja u mlinu sa kuglama. Opiti su izvedeni na topioničkoj šljaci, kao i na mešavinama topioničke šljake i stare flotacijske jalovine definisanih masenih odnosa.

Oprema korišćena u eksperimentima

Opiti su izvođeni u laboratorijskom mlinu tipa „Denver“, pri sledećim tehničko-tehnološkim uslovima:

- karakteristike mlina
  - zapremina mlina: 13,8 l
  - masa šarže: 10,5 kg
  - broj obrtaja 56 min^{-1}
  - dimenzije mlina: DxL=370x130 mm
  - granulometrijski sastav meljuće šarže kugli

Tabela 6. *Granulometrijski sastav šarže meljućih kugli*

Krupnoća kugli (mm)	Masa (%)	D (%)
+50	15,30	100,00
-50+40	43,99	84,70
-40+30	27,64	40,71
-30+0	13,07	13,07
	100,00	-

- karakteristike mlina karakteristike sirovine
  - izdrobljena šljaka do GGK – 3,327 mm
  - flotacijska jalovina GGK – 0,833mm sa procentualnim učešćem 5% i 10%

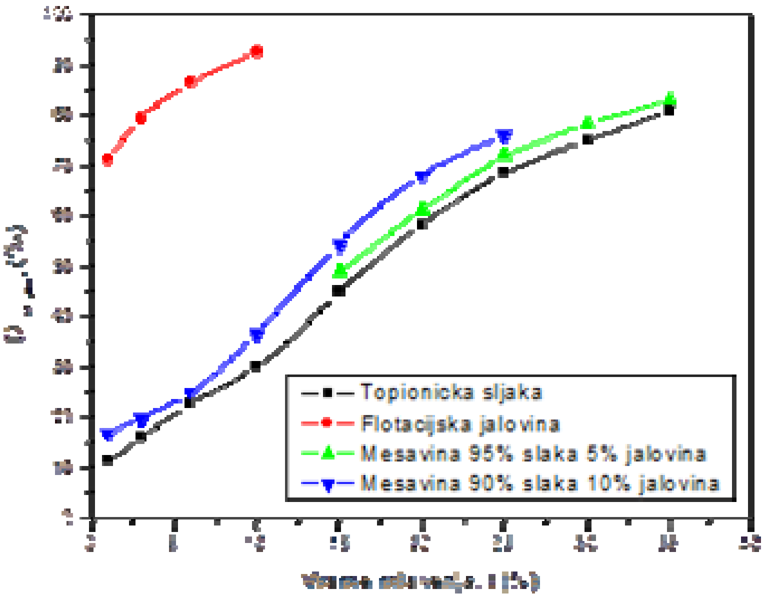
Tehnološki režim laboratoriskih opita mlevenja

Svi eksperimenti su izvodjeni pri pret-hodno definisanim režimom opita mleve-nja koji podrazumevaju:

- masa čvrstog u opitu mlevenja m=1,0 kg
- sadržaj čvrstog u pulpi u prvoj fazi istraživanja je p=75%
- sadržaj čvrstog u pulpi u drugoj fazi istraživanja je p=70-80%
- koeficijent punjenja mlina šaržom φ=15%

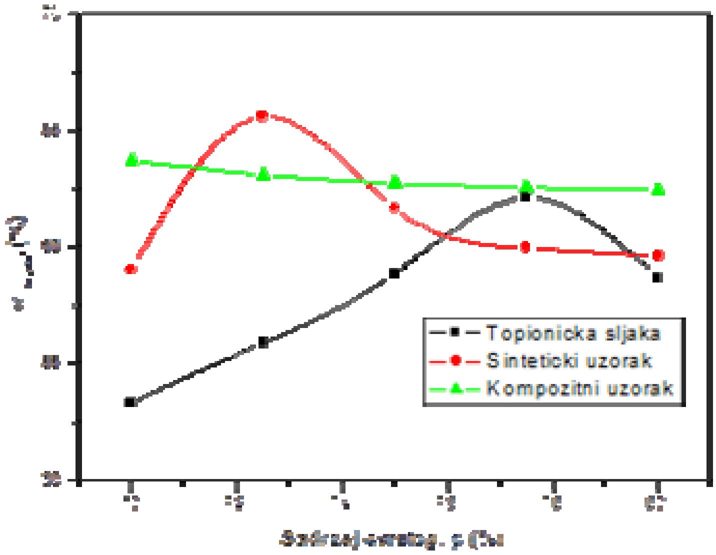
Rezultati istraživanja

Efikasnost mlevenja topioničke šljake i mešavine topioničke šljake i stare flota-cijske jalovine u definisanim masenim odnosima prikazana je na dijagramima na slici 3.



Sl. 3. *Efikasnost mlevenja sirovine u mlinu sa kuglama*

Na slici 4, dijagramom je prikazana zavisnost efikasnosti procesa mlevenja u mlinu sa kuglama od gustine pulpe, odnosno odnosa Č:T, pri mlevenju topioničke šljake i mešavine čljake i stare flotacijske jalovine u trajanju od 20 minuta.



Sl. 4. *Efikasnost mlevenja sirovine u funkciji gustine pulpe*

Rezultati istraživanja efikasnosti mlevenja topioničke šljake i mešavine šljake i flotacijske jalovine u različitim masenim odnosima, prikazani dijagramima na slici 3, pokazuju pretpostavku o pozitivnom uticaju flotacijske jalovine, odnosno prisustvu aluminata na bolju efikasnost mlevenja. Za ostvarivanje finoće proizvoda mlevenja mešavine šljake i flotacijske jalovine od 60 % klase - 0,074 mm, potrebno je 7 % kraće vreme mlevenja za mešavinu 95 % šljake i 5 % jalovine, odnosno 18 % za mešavinu 90 % šljake i 10 % jalovine, u odnosu na isto vreme potrebno za mlevenje topioničke šljake, čime se vrši adekvatna ušteda energije. Različit uticaj sadržaja čvrstog u pulpi, kao i različite vrednosti istog pokazatelja pri kojem se ostvaruju najbolji efekti mlevenja topioničke šljake i mešavina iste sa flotacijskom jalovinom, potvrđuju konstataciju o pozitivnom uticaju aluminata na efikasnost mlevenja.

EKSPERIMENTI ODREĐIVANJA STABILITETA PULPE

Istraživanja uticaja flotacijske jalovine na stabilitet pulpe formirane od topioničke šljake vršena su paralelnim opitima merenja brzine sedimentacije, odnosno visine istaložene čvrste faze u menzuri, u pulpama formiranim od topioničke šljake i mešavina različitih masenih odnosa šljake i flotacijske jalovine.

Oprema korišćena u eksperimentima

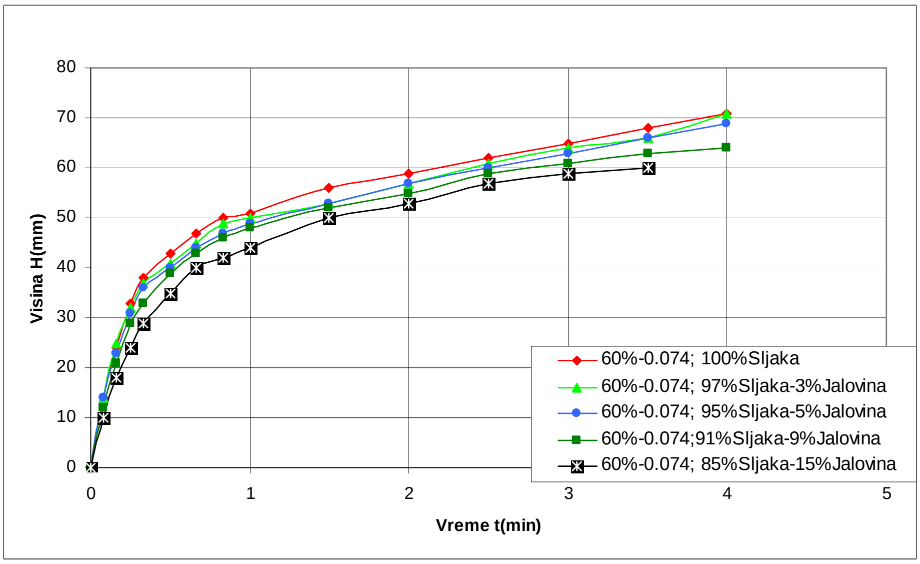
Opiti su izvođeni na indentičnoj opremi, što podrazumeva staklenu menzuru od 1000 ml i mehaničku mešalicu tipa TEHNICA ŽELEZNIKI UM 405.

Tehnološki režim izvođenja opita

Eksperimenti su izvođeni na sobnoj temperaturi od 20^0 C, na uzorcima iste finoće proizvoda mlevenja, 60 % učešća klase – 0,074 mm, kao i istom odnosu čvrste i tečne faze 30% Č, sa konstantnim brojem obrtaja mešalice, 1000 o/min i pH vrednosti pulpe 8.

Rezultati istraživanja

Rezultati istraživanja brzine taloženja čvrste faze u pulpi formiranoj od topioničke šljake i mešavina šljake i flotacijske jalovine u različitim masenim odnosima, prikazani su dijagramima na slici 5.



Sl. 5. *Zavisnost visine formiranog taloga od vremena taloženja topioničke šljake, bez i sa dodatkom jalovine*

Zbog specifičnosti procesa flotiranja koji zahteva održavanje čvrste faze pulpe u disperznom stanju, odnosno obezbeđivanju hidrodinamičkih uslova koji onemogućuju taloženje čvrste faze, eksperimenti određivanja stabiliteta pulpe formirane od topioničke šljake i mešavina iste sa flotacijskom jalovinom izvođeni su merenjem visine istaložene čvrste faze u funkciji vremena taloženja. Rezultati istraživanja prikazani dijagramima na slici 5, nedvosmisleno potvrđuju teoretsku pretpostavku da će prisustvo flotacijske jalovine, odnosno finozrnih aluminata i alumosilikata u pulpi topioničke šljake usloviti povećanu stabilnost iste. Povećanje dužine vremena potrebnog da se u statičkim uslovima procesa sedimentiranja istaloži ista visina taloga čvrste faze u svim vremenskim intervalima i za sve visine istaložene čvrste faze sa sigurnošću potvrđuju pozitivan uticaj flotacijske jalovine na stabilnost pulpe.

Očekivane posledice ove pojave su pre svega veće efikasnosti procesa hidraulične klasifikacije u hidrociklonu i flotacijske koncentracije sa svim prethodno objašnjenim pozitivnim efektima.

EKSPERIMENTI FLOTIRANJA

Istraživanja perocesa flotacijske koncentracije bakra iz topioničke šljake vršena su u pet serija laboratorijskih opita.

Pri istom, prethodno utvrđenom optimalnom tehnološkom režimu, laboratorijski opiti flotacijske koncentracije vršeni su na uzorku topioničke šljake i četiri uzorka mešavine šljake i stare flotacijske jalovine u različitim masenim odnosima.

Oprema korišćena u eksperimentima

Eksperimenti flotiranja vršeni su u laboratorijskoj flotacionoj mašini tipa „DENVER DR-12“ zapremine ćelije 2,6 l. Broj obrtaja rotora je 1600 o/min, sa samousisanom zapreminom vazduha 260 l/min, odnosno 4,33 l/s.

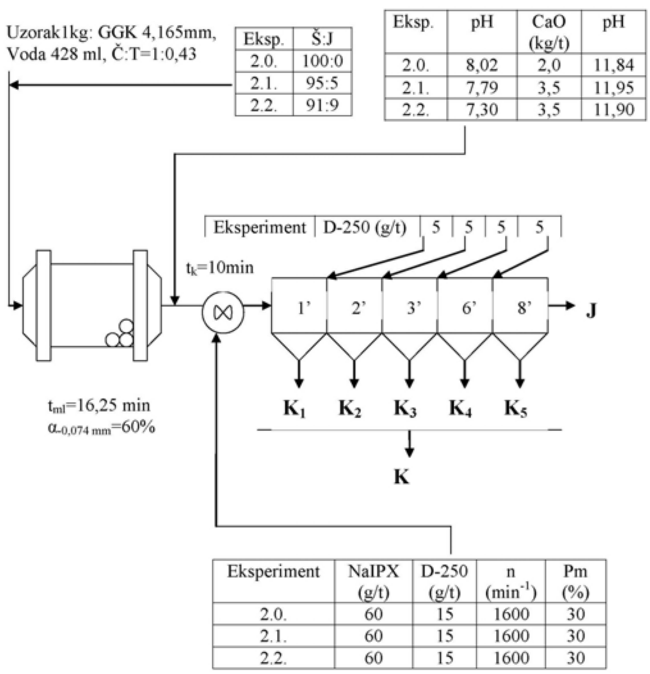
Kontrola pH vršena je pH metromtipa MA 5705, Iskra Kranj, sa kombinovanom elektrodom, tip SENTIX 50.

Tehnološki režim opita flotiranja

Tehnološka mogućnost zajedničke prerade tehnogenih otpada, topioničke šljake i flotacijske jalovine eksplicitno se može povrditi rezultatima koncentracije korisnih komponenti iz istih, kinetikom procesa flotiranja i tehnološkim iskorišćenjem korisnih komponenti.

Efekat uticaja flotacijske jalovine na tehnološke pokazatelje procesa flotiranja topioničke šljake biće sagledani upoređivanjem rezultata paralelnih laboratorijskih opita flotiranja topioničke šljake i mešavina šljake i flotacijske jalovine, u različitim masenim odnosima, pri indentičnoj šemi i režimu flotiranja.

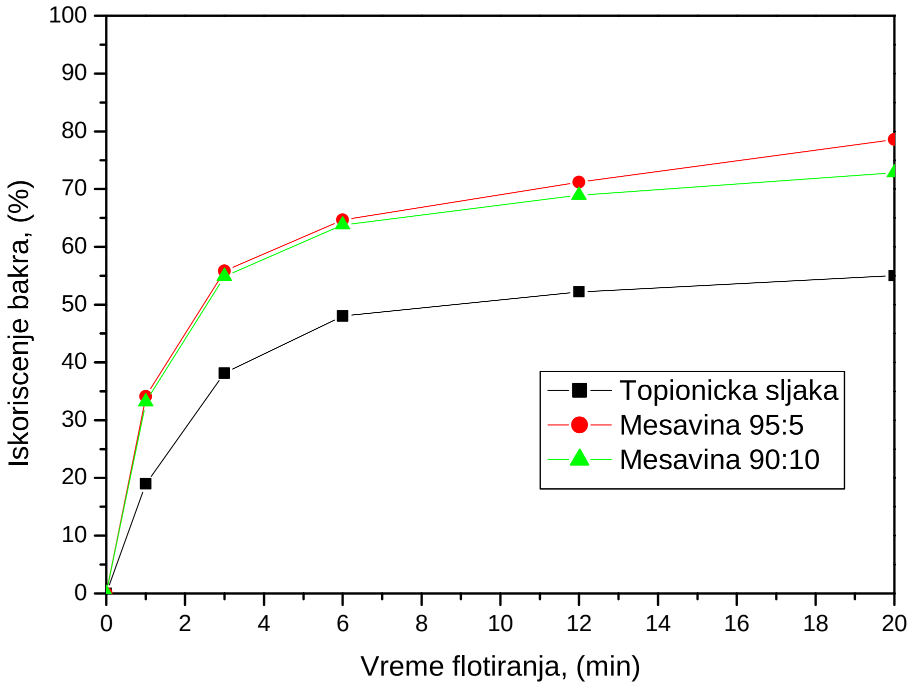
Šema i režim laboratorijskih opita osnovnog flotiranja, prikazani na slici 6, definisani su na osnovu obimnih istraživanja optimizacije parametara procesa.



Sl. 6. *Tehnološka šema izvodjenja eksperimenata*

Rezultati istraživanja

Na slici 7, dijagramima su prikazane kinetike flotacijske koncentracije bakra iz topioničke šljake i mešavina šljake i flotacijske jalovine različitih masenih odnosa, za svaki laboratorijski opit posebno.



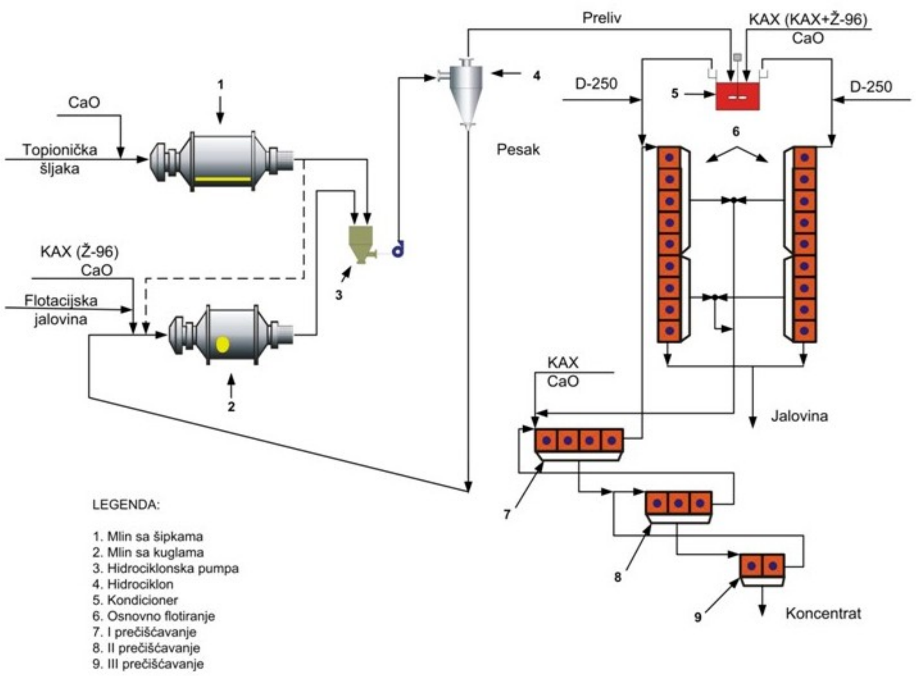
Sl. 7. *Kinetike flotiranja topioničke šljake i mešavina šljake i jalovine u odnosima 95:5 i 90:10*

Rezultati istaživanja kinetike flotiranja bakra iz topioničke šljake i mešavina šljake i flotacijske jalovine, prikazani na slici 7, potvrđuju pozitivan uticaj stabilnosti i viskoziteta pulpe na iskorišćenje bakra. Razlika u iskorišćenju osnovne korisne komponente, bakra za oko 17%, odnosno 24%, pri opitu flotiranja u trajanju od 20 minuta su iznad svih očekivanja. Neophodno je istaći, da su uzrok povećanom iskorišćenju bakra kako bolji hidrodinamički uslovi flotiranja kao posledica veće stabilnosti i viskoziteta pulpe, tako i dodatno iskorišćenje bakra iz same flotacijske jalovine.

Rezultati laboratorijskih istraživanja i zaključci doneti na osnovu istih, potvrđeni su snimanjem industrijskog procesa prerade topioničke šljake u RTB-u Bor, Srbija. Nezadovoljavajuća efikasnost klasifikacije u hidrociklonu definisana preko iskorišćenja obračunske klase krupnoće – 0,074 mm u prelivu od 60 %, kao i vrednost cirkulativne šarže od oko 750–950 % to najbolje potvrđuju. Zbog male stabilnosti pulpe i neodgovarajućeg hidrodinamizma iste u flotacionim mašinama, iskorišćenje bakra u procesu osnovnog flotiranja je takođe nezadovoljavajuća i iznosi oko 56 %.

Sadržaj obračunske klase krupnoće u pesku hidrociklona 6%.

Rezultati prikazanih i drugih istraživanja [2,8,9] bili su osnova za kreaciju idejnog rešenja nove tehnologije za zajedničku preradu topioničke šljake i stare flotacijske jalovine. Nova tehnologija zajedničke prerade dva tehnogena otpada, velika zagađivača životne sredine, prikazana je na slici 8.



Sl. 8. *Tehnološka šema zajedničke prerade topioničke šljake i flotacijske jalovine [2,8,9]*

ZAKLJUČAK

Rezultati eksperimentalnih istraživanja pojedinih fenomena i pojava koji utiču na najznačajnije faze procesa topioničke šljake, potvrđuju naša teoretska razmatranja.

Na osnovu ostvarenih rezultata u laboratorijskim istraživanjima, a koji su prezentovani u radu, u industrijskim uslovima prerade topioničke šljake, primenom predložene tehnologije zajedničke prerade šljake i stare flotacijske jalovine, mogu se očekivati znatno bolji tehno-ekonomski pokazatelji procesa. Smanjenje potrošnje energije i čelika u industrijskom procesu mlevenja, i povećanje kapaciteta prerade, doprineli bi smanjenju troškova za oko 10 – 15 %, a povoljniji granulometrijski sastav proizvoda mlevenja i veća stabilnost pulpe u procesu flotiranja uslovili bi povećanje iskorišćenja za oko 15 %. Ostvareni efekti značajno bi uticali na povećanje ekonomičnosti prerade navedenih tehnogenih otpadnih sirovina, i učinili bi je ekonomski održivom.

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UDK: 622.7:666.952:658.5671(045)=20

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NEW TECHNOLOGICAL PROCEDURE FOR SUSTAINABLE PROCESSING OF MINING TECHNOLOGICAL WASTE**

Abstract

The results show technological possibility and economical feasibility for joint processing of two technological waste materials generated in the process of copper ore treatment. Smelter slag and flotation tailings from the Mining and Smelting Basin - RTB Bor, Serbia, are two technological waste materials stored on dumps and emerging permanently. They represent the major polluters of the environment, but also the potential raw material resources, according to the content of useful components. Due to the specific physical, chemical and mineralogical characteristics, individual processing of these waste materials do not guarantee a profitable business. Joint processing of smelter slag and old flotation tailings, in mass ratio 90% of slag and 10% of flotation tailings, in the original process technology, increases recovery of copper in the flotation process for approximately 15%, efficiency of grinding and classification processes are also increasing, thereby power consumption is reduced 10 -15%, making the processing of these waste materials to have both the positive economic and ecological effect.

***Key words:** smelter slag, flotation tailings, technological wastes, flotation, copper recovery*

INTRODUCTION

Mining and Smelting Basin RTB Bor, produces dozens of different polluting, technological waste materials in complex technologies of copper production, from exploitation to the electrolytic refining [1]. A number of them, in gaseous, liquid or

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solid form, contain significant concentrations of useful components, which could be valorized using the modern technologies. The research results of processing the smelter slag, the most common waste in copper metallurgy, indicate the possibility of technological processing [2,3]. Due to the increased resistance to comminution, increased density and lack of aluminates [4-7], the formed pulp from smelter slag has the reduced viscosity and stability. These characteristics of smelter slag and pulp, formed from slag condition, have caused the rapid sedimentation of the solid, which negatively affects the efficiency of the most important processes, grinding and classification, as well as the process of flotation concentration. Increased energy consumption in grinding process, i.e. increased processing costs and reduced recovery of useful components in the flotation process have called a question to the economy of this technological waste processing.

Researches to improve viscosity and stability of the pulp, by adding flotation tailings in the process of grinding slag should yield the positive results. The use of flotation tailings, the most common technological waste and also the biggest polluter in the process of copper ore processing, will improve the pulp characteristics, and thus the efficiency of the most important phases of the process. Thanks to lesser density of flotation tailings, and the presence of fine particles of aluminates and aluminosilicates, the pulp formed from 90% of slag and 10% of tailings, has the increased viscosity and stability which creates higher efficiency and lower power consumption in the processes of grinding and classification, and also increased recovery of useful components in the flotation process. Increased recovery of copper from mixture of smelter slag and flotation tailings could be the result of higher recovery of copper from slag because of better hydrodynamic conditions of flotation, and also recovery of copper from flotation tailings. Higher pulp viscosity has the positive effect on grinding efficiency in the ball mill, due to better adhering of material on grinding balls. Increased efficiency of hydraulic classification in hydrocyclones, as the result of higher pulp stability, reduces the unnecessary circulation of enough comminuted particles for the flotation process, through hydrocyclone underflow to regrinding in the ball mill.

Reducing the circulative charge to the optimal limit, i.e. reducing the share of enough grained particles in the hydrocyclone underflow, has multiple technical and economic importance. Besides increasing capacity and reducing energy consumption in the grinding process, less over grinding of useful components particles causes their better recovery in the flotation process.

The positive effects of applying the new technology for common processing of smelter slag and flotation tailings, expressed through reduction of processing costs and increased recovery of useful components, allow a profitable activity and economic sustainability of the environment.

EXPERIMENTAL PART

Experimental studies were carried out in two phases. The first phase was aimed to determine the characteristics of smelter slag and flotation tailings and the pulp formed from these materials, which are significant for intended processing procedures. In the second phase, parallel experiments of flotation concentration of copper from smelter slag, and mixtures of slag and flotation tailings, were carried out. The results should be verified by previous theoretical considerations of technological feasibility the common processing of these technological waste materials.

Smelter Slag Characteristics

For selected process of copper concentration from smelter slag, the flotation concentration, the mineralogical and chemical compositions of materials have particular importance.

Microscopic and mineralogical studies were carried out on microscope Carl Zeiss-Jena "JENAPOL-U". The results of these studies are shown in Table 1.

Chemical analysis of the samples was carried out using different analytical methods (Table 2).

Table 1. *Mineralogical composition of smelter slag*

Minerals	Content (%)
Fayalite	60.00
Magnetite	30.00
Bornite	6.50
Chalcopyrite	1.50
Metallic copper	0.50
Copper matte	0.30
Chrysocolla	0.15
Covellite	0.05
Pyrite	1.00
∑	100.00

Table 2. *Chemical composition of smelter slag*

Element Compound	Content (%)	Analytical method
Fe_{Σ}	37.05	VT
SiO_2	32.77	G
Fe_3O_4	9.68	A-Fe 304
Al_2O_3	4.86	ICP-AES
Fe in Fe_2O_3	3.02	R
S	0.81	G
Ca	3.21	AAS
Mg	0.62	AAS
Cu_{Σ}	0.84	SF
Cu_{elem}	0.38	G
Cu_{sul}	0.30	R
Cu_{ox}	0.16	PO
Ag	3.3 g/t	FA
Au	0.5 g/t	FA

Mineralogical and chemical compositions of smelter slag confirm the possibility of use the flotation concentration as the method for concentration of useful components. The contents of copper 0.84% and precious metals, gold 0.5 g/t and silver 3.3 g/t, confirms the statement that this technological waste contains 2-3 times higher concentration of these useful components comparing to their contents in the primary copper ores which are now exploited (0,25 - 0,35% Cu). This fact indicates that this technological waste represents the raw material resource and justifies the research possibilities for its processing.

Density of smelter slag is approximately 3500 kg/m^3. The fact that density of this technological waste is about 500-700 kg/m^3 higher than density of copper ore leads to a conclusion that the sedimentation rate of slag particles in the pulp is higher compared to the sedimentation rate of copper ore particles of the same size.

Grindability, i.e. the resistance to comminution, grinding kinetics and grain size composition are physical characteristics of the material that largely influence the processing costs.

Bond’s work index in the ball mill for smelter slag is Wi = 31 kWh/t, what is about two times higher comparing to the copper ore. This fact points out to the expected high energy consumption in the grinding process. Diagrams in Figure 1 show the grinding kinetics for smelter slag and copper ore. It can be seen that much longer time is needed for grinding the smelter slag compared to copper ore, in order to achieve the same fineness of grinding products.

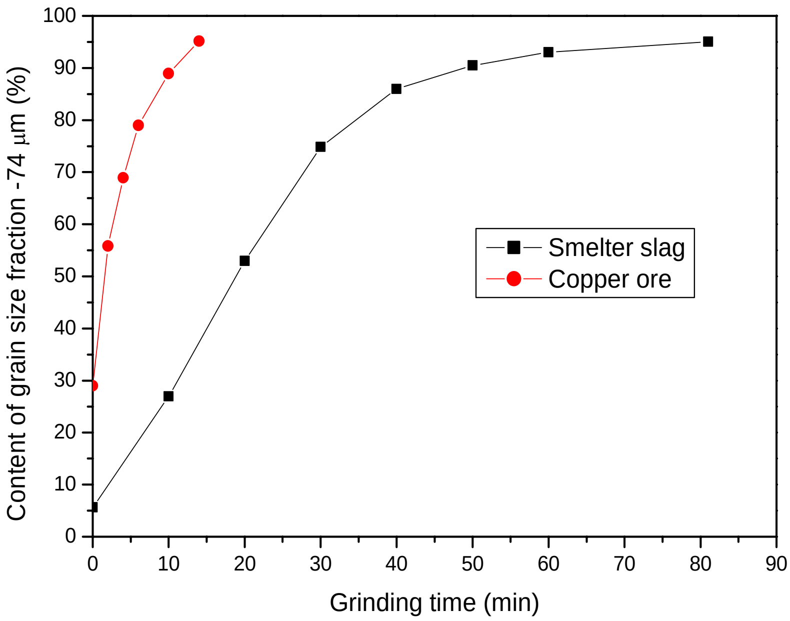

Fig. 1. *Efficiency of grinding smelter slag and copper ore*

The research results clearly indicate the high resistance to comminution of this technological waste that confirms the previous observation of high energy consumption in the process of fragmentation and, hence, the increased processing costs.

Flotation Tailings Characteristics

Studies were performed on the "old" flotation tailings produced in the process of copper ore flotation in the first half of the last century. This technological waste material was chosen due to the increased content of useful components, 0.15 – 0.4% Cu, caused by high content of copper in the ore that was processed in that period (2.5-3% Cu), as well as the efficiency of flotation concentration technology, applied at that time.

Mineralogical compositions of samples from the old flotation tailings from different depths of the old flotation tailing dump are shown in Table 3, and the chemical composition of samples from different parts of dump, characteristic for the flotation process, are shown in Table 4.

Table 3. *Mineralogical composition of samples from the old flotation tailings*

Minerals	Content (%)	
	Sample 1 (0-16 m)	Sample 2 (16-40 m)
Chalcopyrite	0.16	0.12
Covellite	0.03	0.11
Enargite	0.01	0.02
Chalcocite	0.01	0.01
Bornite	in traces	0.01
Azurite	in traces	0.03
Cuprite	0.01	0.04
Pyrite	14.27	22.20
Tailings minerals	80.63	71.96
Other minerals	4.88	5.50
Σ	100.00	100.00

Table 4. *Chemical composition of samples from the old flotation tailing [3]*

Element Compo-nent	Dam of flotation tailing dump	Inside of flotation tailing dump	Analytical method
	Content (%) (g/t)*	Content (%) (g/t)*	
Cu_{Σ}	0.155	0.370	SF
Cu_{ox}	0.033	0.120	PO
Cu_{sul}	0.122	0.250	R
S	9.870	12.260	G
Fe	8.860	10.270	VT
SiO_2	58.030	59.710	G
Al_2O_3	12.040	12.630	ICP-AES
Au	0.300*	0.600	FA
Ag	2.140*	1.300	FA

The results show that flotation tailings contain the significant concentrations of useful components in mineralogical form suitable for use in the flotation process.

However, according to the basic reason for using the flotation tailings in the process of smelter slag processing, to improve the characteristics of the pulp, i.e. to increase the viscosity and stability to the optimal value, the physical characteristics of this technological waste have a great importance.

Density of flotation tailings was determined on a number of samples taken from different places of the flotation tailing dump. The need of taking samples from different places of the flotation tailing dump, from dam to its beach, was caused by the way of formation the flotation tailing dump and distribution of hydrocyclone overflow and underflow. Density value of individual samples as well as the mean density of flotation tailings, used in further researches, is given in Table 5.

Table 5. *Density values of samples of the old flotation tailings [3]*

Flotation tailings	Sample 1 (dam of flotation tailing dump)	Sample 2 (central part of flotation tailing dump)	Sample 3 (peripheral part of flotation tailing dump)	Composite sample
ρ (kg/m³)	2 650	3 070	2 814	2 844

Much lower density of flotation tailings from the density of smelter slag is caused by different mineralogical and chemical composition. While in the smelter slag, the ferrosilicates with higher densities (3700-3900 kg/m^3) have dominant participation, then aluminum silicates with significantly lower densities (2500-2600 kg/m^3) have the highest participation in the flotation tailings.

Resistance to the comminution of fine-grained flotation tailings material is not possible to determine by the standard Bond's method; however, the same can be identified with resistance to the commi-nution of the primary copper ore, that was 10-15 kWh/t for processed ore in that period.

Diagrams in Figure 2 show the comparative value of grinding kinetics of two types of technological waste.

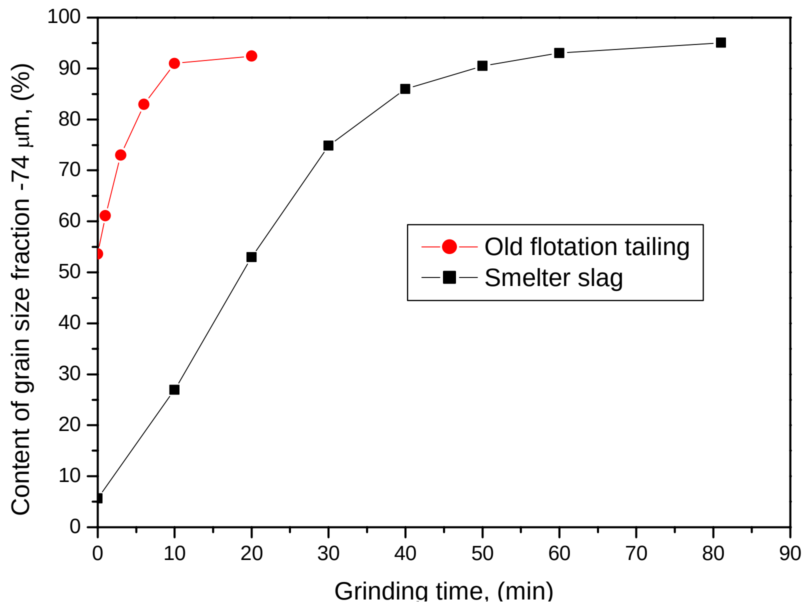

Figure 2. *Efficiency of grinding smelter slag and old flotation tailings*

Much shorter time, required to achieve the certain fineness of flotation tailing grinding product in relation to the required time to achieve the same fineness of grinding smelter slag, points out to a logical conclusion on expected better grindability of these two types of technological waste mixture in relation to the grindability of pure smelter slag.

EXPERIMENTS OF GRINDING THE SMELTER SLAG AND MIXTURE OF SLAG AND FLOTATION TAILINGS

Grindability study was done by the wet grinding method in a ball mill. In the first phase of the study, three series of laboratory experiments were carried out: on smelter slag and mixture of slag and old flotation tailing in the certain mass ratios, with the aim of defining the grinding efficiency of individual samples.

The second phase of study was aimed to investigate the effect of pulp density, or solid/liquid ratio on the efficiency of grinding process in the ball mill. Experiments were carried out on smelter slag, as well as mixtures of slag and old flotation tailing.

The used equipment in experiments

Experiments were carried out in a laboratory mill, type "Denver", with the following technical and technological conditions:

- Characteristics of the mill
  - Mill capacity: 13.8 l
  - Charge mass: 10.5 kg
  - Speed: 56 min^{-1}
  - Mill size: DXL = 370x130 mm
  - Grain size distribution of grinding balls charge (Table 6.)

Table 6. *Grain size composition of charge of grinding balls*

Coarseness of balls (mm)	Mass (%)	D (%)
+50	15.30	100.00
-50+40	43.99	84.70
-40+30	27.64	40.71
-30+0	13.07	13.07
	100.00	-

- Characteristics of raw materials
  - Crushed smelter slag – 3.327 mm
  - Flotation tailings – 0.833 mm with percentage participatio of 5% and 10%

Technological regime of laboratory grinding experiments

All experiments were carried out at predefined regime of grinding:

- Mass of solid, m = 1.0 kg
- Solid content in the pulp in the first phase of the study, p = 75%
- Solid content in the pulp in the second phase of the study, p = 70-80%
- Coefficient of mill charging with ball charge, φ = 15%

Research results

The efficiency of grinding smelter slag and mixture of slag and old flotation tailings in defined mass ratio is shown in the diagrams in Figure 3.

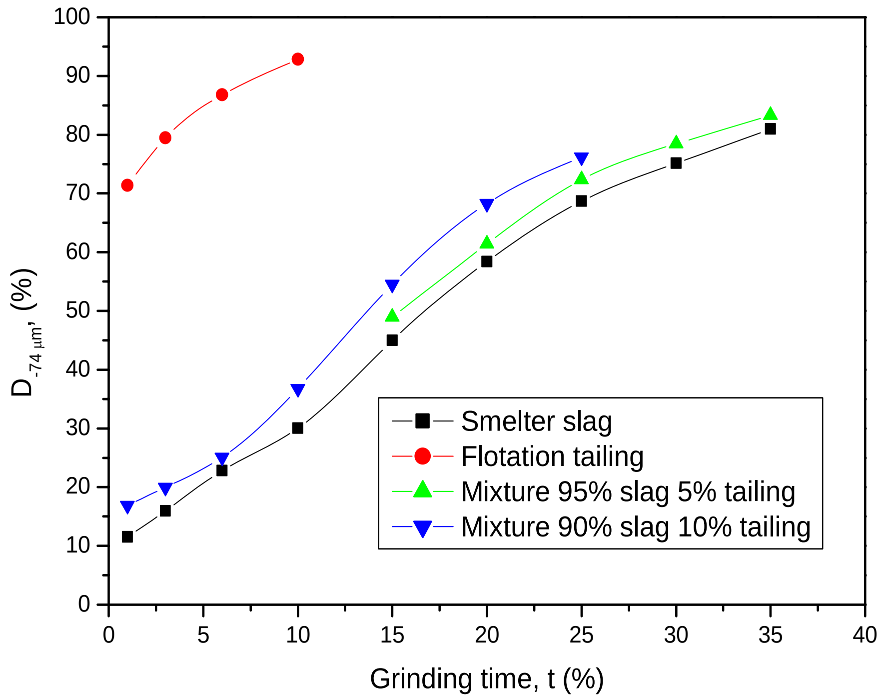

Figure 3. *Efficiency of grinding raw materials in ball mill*

Diagram in Figure 4 shows the dependence of grinding efficiency in the ball mill on pulp density, or solid/liquid ratio in grinding the smelter slag and mixture of slag and old flotation tailings for 20 minutes.

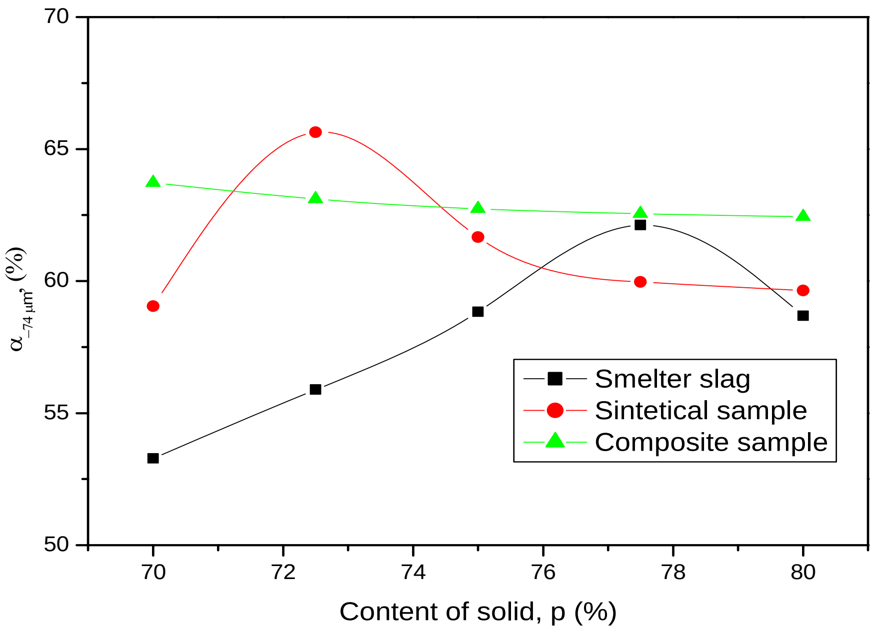

Figure 4. *Efficiency of grinding the raw materials in the function of pulp density*

Study results on the efficiency of grinding smelter slag and mixture of slag and flotation tailings in different mass ratios, Figure 3, show the positive impact of flotation tailings, or presence of aluminates, on better grinding efficiency. In order to achieve 60% of grain size fraction – 0.074 mm in the grinding product of slag and flotation tailing mixture, 7% less grinding time is needed for mixture 95% of slag and 5% of tailings, and 18% for mixture 90% of slag and 10% of tailings, compared to the required time for only smelter slag grinding, which made the adequate energy savings. Different impact of solid content in the pulp, as well as various values of the same indicator at which the best effects of grinding smelter slag and mixture of slag and flotation tailings are achieved, confirm the conclusion on the positive impact of aluminates on grinding efficiency.

EXPERIMENTS OF DETERMINING THE PULP STABILITY

Investigations the impact of flotation tailings on stability of formed pulp from smelter slag were carried out by parallel measurements of deposition rate, i.e. the height of deposited solid phase, in the formed pulp from smelter slag and mixtures of slag and flotation tailings in different mass ratios.

The used equipment in experiments

Experiments were carried out on identical equipment, including a glass measuring cylinder of 1000 ml and a mechanical stirrer, type TECHNIQUE ŽELEZNIKI UM 405.

Technological regime of experiments

Experiments were carried out at room temperature of 20°C, on samples with same grinding product fineness, 60% of grain size fraction – 0.074 mm, and with 30% of solid phase in the pulp, with a constant rate of stirrer, 1000 rpm, and pH value of the pulp of 8.

Experimental results

Experimental results of deposition rate of solid phase in the formed pulp from smelter slag and mixture of slag and flotation tailings in different mass ratios are shown on diagrams in Figure 5.

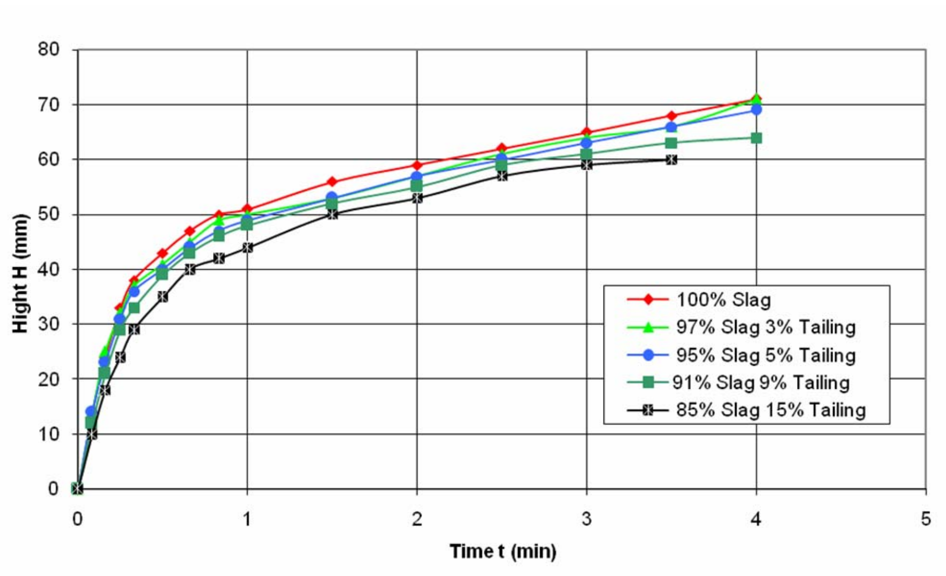

Figure 5. *Dependence of the height of formed deposit on deposition time in the formed pulp from smelter slag, with and without addition of flotation tailings*

Due to a specificity of flotation process that requires a solid phase in the pulp in dispersed state, i.e. to provide hydrodynamic conditions that disable the sedimentation of solid phase, the experiments of determining the stability of formed pulp from smelter slag and mixture of slag flotation tailings were carried out measuring the amount of deposited solid phase in the function of deposition time. Research results, shown on diagrams in Figure 5, unambiguously confirm the theoretical assumption that the presence of flotation tailings, or fine-ground aluminates and aluminosilicates in the formed pulp from smelter slag will increase its stability. Increasing the time required to settle down the same amount of solid phase in all intervals and all deposited amounts of solid phase in the static conditions of sedimentation process, with certainty confirm the positive impact of flotation tailings on the pulp stability.

Expected consequences of this phenomenon are primarily higher efficiency in the process of hydraulic classification in hydrocyclones and flotation concentration with all positive effects, described above.

FLOTATION EXPERIMENTS

Research of flotation concentration of copper from smelter slag was carried out in five series of laboratory experiments.

Laboratory flotation concentration experiments were carried on a smelter slag sample and four samples of mixture of slag and old flotation tailings in different mass relations, with the same, previously determined optimal technological regime.

The used equipment in experiments

Flotation experiments were conducted in laboratory flotation machine, type DENVER DR-12, with 2.6 l volume of a cell. Impeller rate of the rotor was 1600 rpm, and the volume of added air 260 l/min or 4.33 l/s.

Control of pH was performed by pH meter, type MA 5705, Iskra Kranj, with a combined electrode, type SENTIX 50.

Technological regime of flotation experiments

Technological capabilities for mutual processing of technological wastes, smelter slag and flotation tailings, can be confirmed by the results of concentration of useful components from them, kinetics of flotation process and technological recovery of useful components.

The effect of flotation tailings impact on indicators of technological process of smelter slag flotation will be viewed by comparison the results of parallel laboratory experiments of flotation smelter slag and mixture of slag and flotation tailings, in different mass ratios, with identical flow sheet and regime of flotation.

The flow sheet and regime of laboratory experiments of basic flotation, shown in Figure 6, were defined based on extensive research of optimization the process parameters.

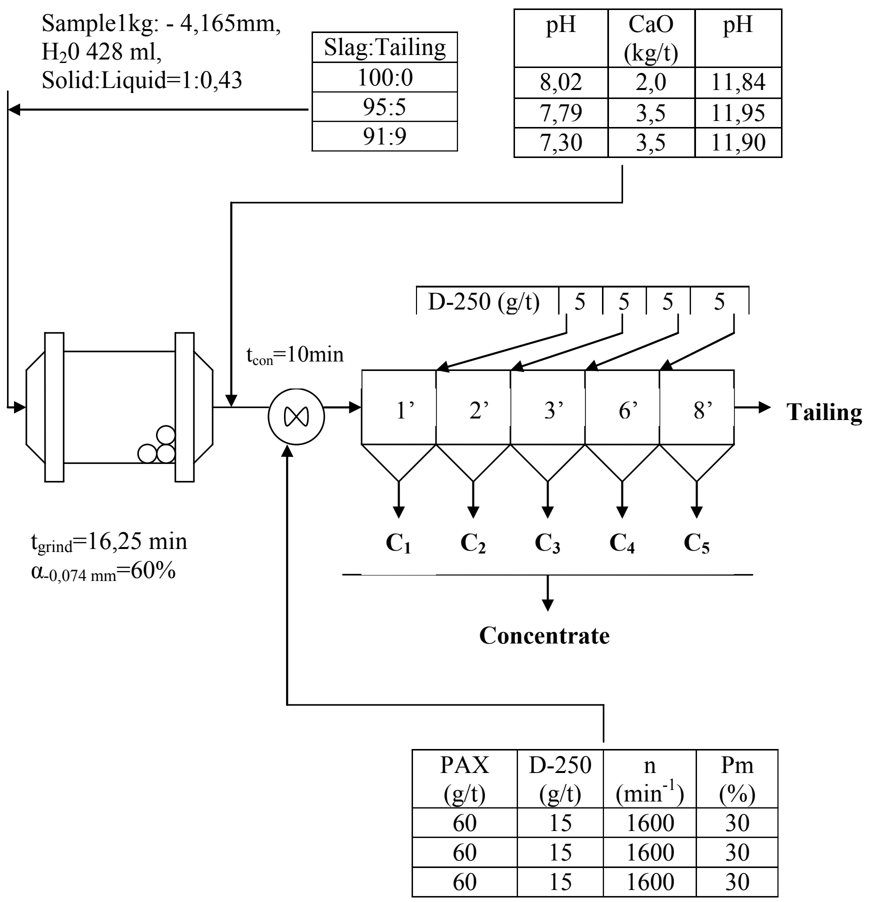

Figure 6. *Flow sheet of flotation experiments*

Research results

Diagrams in Figure 7 show the kinetics of copper flotation concentration from smelter slag and mixture of slag and flotation tailings in different mass ratios for each laboratory experiment separately.

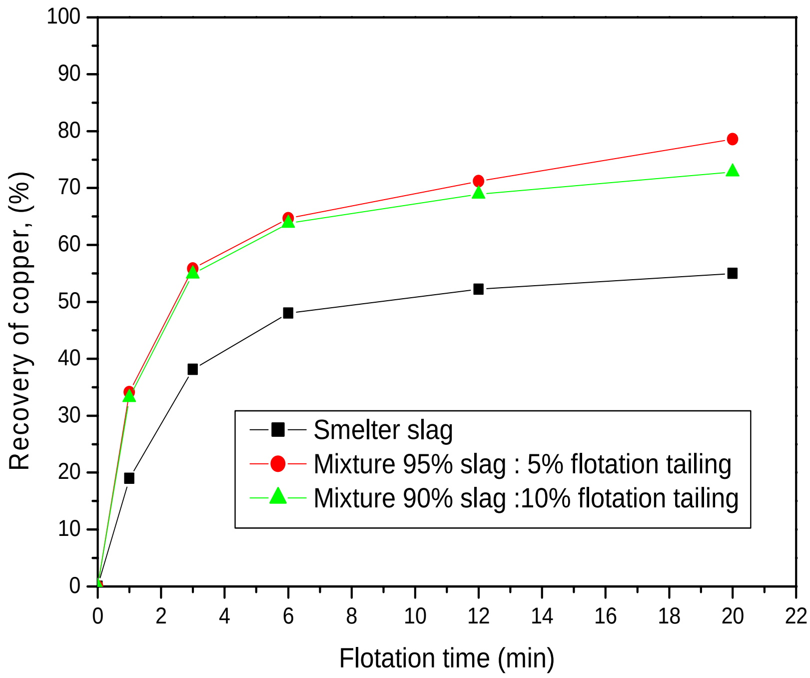

Figure 7. *Flotation kinetics of smelter slag and mixture of slag and flotation tailings*

Testing results of copper flotation kinetics from smelter slag of and mixture of slag and flotation tailing, shown in Figure 7, confirm the positive impact of stability and viscosity of pulp on copper recovery. The difference in copper recovery for approximately 17%, i.e. 24%, for the flotation experiment lasting 20 minutes, were beyond expectations. It is necessary to point out that the cause of increased copper recovery are better hydrodynamic flotation conditions as the result of higher pulp viscosity and stability, as well as additional copper recovery from flotation tailings.

The results of laboratory testing and conclusions, based on them, were confirmed observing the industrial process of smelter slag processing in RTB Bor, Serbia. Unsatisfactory, the efficiency of classification in hydrocyclones, defined by recovery of grain size fraction – 0.074 mm in hydrocyclone overflow of 60% and the value of circulative charge of approximately 750-950%, confirm this. Due to poor pulp stability and its inadequate hydrodinamics in the flotation machines, copper recovery in the basic flotation process was also unsatisfactory, approximately 56%.

Content of grain size fraction – 0.074 mm in hydrocyclone underflow was 6%.

Present results and other studies [2, 8, 9] were basis for development the conceptual design for the new technology for mutual processing of smelter slag and old flotation tailings. The new technology for mutual processing two types of technological waste, large polluters, is shown in Figure 8.

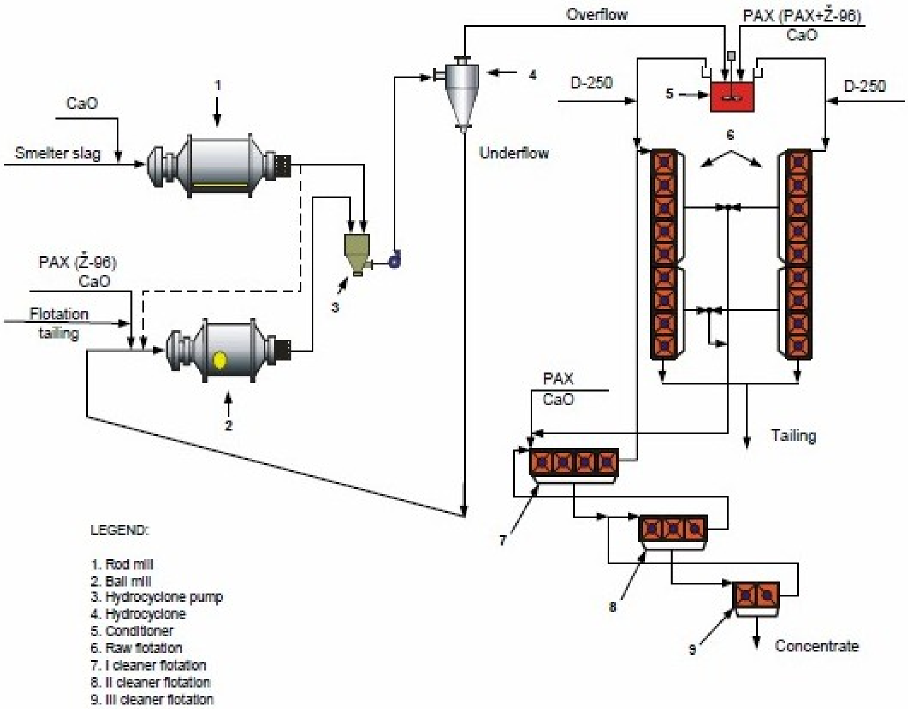

Figure 8. *Technological flow sheet for mutual processing of smelter slag and old flotation tailings [2,8,9]*

CONCLUSION

Results of experimental studies of specific phenomena, that affect the most important stages in the process of smelter slag flotation, confirm our theoretical considerations.

Based on the results of laboratory testing, presented in this paper, in the industrial conditions of smelter slag processing, much better techno-economic indicators of the process can be expected applying the proposed technology for mutual processing of slag and old flotation tailings. Reducing energy and steel consumption in the industrial grinding process, and increasing the processing capacity, can contribute to the reduced costs by 10 - 15%, and more appropriate grain size fraction of grinding product, and increased stability of pulp in the flotation process could cause the increased recovery by 15%. The achieved effects would significantly affect decrease in costs of processing these technological waste materials and would do it economically viable.

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FAZNO OTVARANJE RUDNOG TELA "BORSKA REKA"[**]

Izvod

U radu se govori o osnovnim paramerima otvaranja i razrade rudog tela „Borska reka". Rudno telo je otvoreno do kote K -235 m, a konture rudnog tela sa graničnim sadržajem od 0,3 % idu i do kote K – 995 m [1]. Otvaranje dubljih delova može se vršiti etažno za visinu jednog horizonta ili fazno kada se više horizonata mogu objediniti u jedan koncentracioni ili sabirni horizont. Primenom matematičko-analitičke metode u radu se odredjuje optimalna visina izmedju koncentacionih horizanata odnosno odredjuju se faze otvaranja dubljih delova Borskog ležišta.

Ključne reči: *otvaranje ležišta, koncentracioni horizonti, troškovi, optimizacija*

UVOD

Područje rudnog tela "Borska reka" predstavlja nastavak borske hidrotermalno izmenjene zone u pravcu severozapada na dužini oko 1520 m i po padu oko 800 m. Zaleže prema zapadu pod uglom približno 45-55^{o}. Ovo zaleganje je saglasno zaleganju borskih peščara i konglomerata koje od borske hidrotermalno izmenjene zone odvaja borski rased [2].

Rudno telo se nalazi vrlo duboko ispod površine terena. Dubina mu se povečava prema zapadu i severozapadu. Generalno gledano, dubina rudnog tela u konturi 0,3 % Cu kreće se izmedju 600 i 850 m, s tim što je srednja močnost preko 300 m.
Najplića mineralizacija rudnog tela je u bušotini B - 38,398 m, a najdublja na 1402 m u bušotini B -33 [3].

Dosadašnjim istraživanjem je konstantovano da rudno telo "Borska reka" pripada grupi vrlo velikih ležišta. Dužina po pružanju iznosi oko 1520 m, širina i do 635 m, prosečna močnost prelazi 300 m, a maksimalna površina na K - 560 m u konturi 0,3 % iznosi preko 500.000 m^2 [3].

Rezerve rude u rudnom telu "Borska reka" proračunate su po metodi horizontalnih profila. Površine u rudnom telu obračunate su na nivoima, koji predstavljaju nivoe budućih horizonata, čija visinska razlika iznosi 80 m. U konturi ležišta sa 0,3 % Cu, ukupna količina rezervi rude iznosi 623.637.494 t, sa srednjim

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sadržajem bakra u rudi 0,608 % Cu, a ukupna količina bakra je 3. 799. 447 t [4].

OTVARANJE RUDNOG TELA BORSKA REKA DO K-235 M

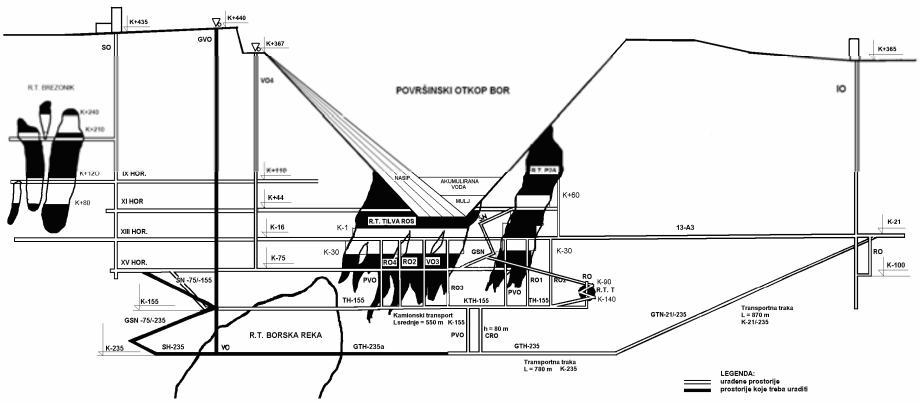
Rudno telo "Borska reka" se nalazi ispod sadašnjeg nivoa eksploatacije rude u jami Bor i dostiže dubinu od oko 1200 m. Rudno telo je delimično otvoreno do nivoa XIX horizonta odnosno do K – 235 m.

Sa XIII horizonta iz novog izvoznog okna na K-21 m izrađen je glavni izvo-zni niskop GIN do K-235 m pod nagi-bom od 16,5°, dužine 750 m. Na nivou K-235 m izrađen je Glavni transportni hodnik (GTH-235 m) do Centralnog rudnog okna u dužini od 780 m [1].

Veza rudnog tela sa servisnim oknom ostvarena je izradom je istražnog niskopa pod uglom od oko 8° do K -155 m u dužini od 560 m. Iz istražnog ni-skopa urađeni su istražni hodnici na nivou K -155 m kroz rudno telo do izlaska iz konture orudnjenja [1].

Na nivou XVII horizonta, K -155 m, urađen je Glavni transportni hodnik u celoj dužini rudnog tela. Služi za transport iskopina pri izradi objekata otvaranja i razrade. U fazi eksploa-tacije služiće za potrebe servisiranja XVII horizonta.

Veza izmedju XIX i XVII horizonta ostvarena je izradom prolazno-ventilaciog okna (PVO) i centralnog-rudnog okna (CRO), slika br.1.



Sl. 1. *Šema objekata otvaranja ležišta Borska reka do K-235 m*

Na osnovu detaljnog sagledavanja i analize faktora koji utiču na otvaranje rudnog tela "Borska reka" došlo se do zaključka da otvaranje rudnog tela "Borska reka" treba rešavati u sklopu postojećih objekata, radi manjih investicionih ula-ganja, a da je potrebno još uraditi:

- Glavni servisni niskop (GSN-75/-235) sa K-75 iz zaobilaznog hodnika navozišta XV horizonta pod nagibom od 11,3° u dužini od 816 m.
- Glavni transportni hodnik (GTH-235a) dužine 756 m (ovim hodnikom otvara se XIX horizont). Izrađuje se na K-235m, kao nastavak GTH-235

pored rudnog tela Borska reka do SH-235. i kojim se povezuje sa GSN-75/-235.

- Glavno ventilaciono okno od K-155 do K-235 (GVO-155/-235). Za potrebe ventilacije treba da se izradi Glavno ventilaciono okno (GVO) od K+110 (na IX horizontu) do površine, koje bi zamenilo VO_4 koje je ugroženo u kosini napuštenog površinskog kopa Bor. Za potrebe otkopavanja rudnog tela “Borska reka” GVO bi se nastavilo sa K+110 do K-155 gde bi se ventilacionim hodnikom spojilo sa hodnikom u blizini rudnog tela, jer se VO_4 od XV do XIII horizonta zarušava zato što se nalazi u zoni zarušavanja.

Navedeni objekti otvaranja i razrade poduhvataju rudno telo do kote –235. Niži deo rudnog tela “Borska reka” ispod kote –235, otvaraće se u narednoj fazi eksploatacije.

OTVARANJE RUDNOG TELA BORSKA REKA ISPOD K-235 M

Za odredjivanje visine narednog koraka otvaranja, odnosno visine koncentracionog horizonta primeniće se matematičko - analitička metoda. Suština metode svodi se na to da se za dati način otvaranja ležišta postavljaju, u opštem matematičkom obliku, izrazi za troškove radova po jedinici proizvodnje, ali samo za radove koji zavise od načina otvaranja ležišta. Ti troškovi se predstavljaju funkcijom T = f (n,H). Ova funkcija predstavlja zbir tri vrste troškova, i imaće minimum koji odredjuje optimalnu visinu koncentracionih horizonata, $H_k = n \cdot H$

Pri rešavanju postavljenog zadatka pošlo se od neophodne predpostavke koja podrazumeva uspostavljanje funkcionalnih zavisnosti. Neophodno je sve vertikalne rudarske objekte definisati kroz poznati parametar, visinu horizonta H, koja u matematičkom smislu predstavlja korak iteracije [5].

Dubina vertikalnih rudarskih objekata (H) izmedju dva susedna sabirna horizonta (koncentraciona horizonta) projektovana za visinu horizonta (H), može se aritmetički definisati relacijama:

Za 1 horizont: $H_{k_1} = H$

Za 2 horizonta: $H_{k_2} = 2 \cdot H$

Za 3 horizonta: $H_{k_3} = 3 \cdot H$

Za (n – 1) horizonta:
$H_{k_{(n-1)}} = (n-1) \cdot H$

Za n horizonta: $H_{k_n} = n \cdot H$

Poslednji izraz pokazuje da je dubina objekata otvaranja izražena kao proizvod visine horizonta (H) i broja horizonata objedinjenih na jednom koncentracionom (sabirnom) horizontu, pa se optimizacija svodi na definisanje "n" [5].

U osnovi odredjivanja optimalne visine koncentracionih horizonata polazi se od predpostavke da se sva investiciona ulaganja kao i ostali tehnološki troškovi daju u vidu specifičnih troškova ulaganja, odnosno kao troškovi po jedinici proizvoda (po toni dobijene rudne mase).

Specifični troškovi ulaganja se dobijaju iz odnosa ukupne sume investicionih ulaganja i zahvaćenih rudnih rezervi. Ove rezerve biće izražene preko poznatih parametara jednog horizonta, koji predstavlja korak iteracije u matematičkom modelu.

Eksploatacione rezerve rude na jednom horizontu iznose [6]:

$$Q_e = \frac{Q_g \cdot K_{ir}}{1 - K_{or}} = Q_g \cdot K_{rm}, \text{ t} \ldots\ldots (1)$$

gde su:

Q_g – geološke rezerve na jednom horizontu, t

K_{ir} – koeficijent iskoriščenja rude,

K_{or} – koeficijent osiromašenja rude,

K_{rm} – koeficijent rudne mase.

Geološke rezerve na jednom horizontu iznose:

$$Q_{e1} = P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm} \text{ , t ………(2)}$$

gde su:

P_{sr} – srednja površina izmedju dva horizonta, m^2

H – visina horizonta, m

γ_r – zapreminska masa rude, t/m^3

Rezerve u jednoj fazi eksploatacije definišu se kao proizvod broja horizonta (n), objedinjenih koncentracionim ili sabirnim horizontom i rezervi rude na jednom horizontu (Q_{e1}). Odnosno:

$$Q_e = n \cdot Q_{e1} = n \cdot P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm} \text{ , t …(3)}$$

Analitički izraz za specifične troškove ulaganja (jedinične troškove) iznosi:

$$T_i^{'} = \frac{T_i}{Q_e} \text{ , US \$/t …………(4)}$$

gde su:

$T_i^{'}$ – specifični troškovi ulaganja, US $/t

T_i – ukupna suma investicionih ulaganja, US $

Q_e – eksploatacione rezerve koncentracionog horizonta, t

ODREDJIVANJE TROŠKOVA OTVARANJA U JEDNOJ FAZI

Troškovi izrade prostorija drobiličnog postrojenja

Kompleks drobiličnog postrojenja sa pratećim objektim odlikuje se odredjenom složenošću objekata, koji sinhronizuju deo tehnološkog procesa drobljenja i izvoza rude.

Troškovi izrade prostorija u okviru drobiličnog kompleksa predstavljaju zbir troškova izbijanja i troškova podgradjivanja.

Imajući u vidu da se drobljenje rude vrši na svakom koncentracionom horizontu, onda se analitički izraz za jedinične troškove ulaganja u drobilični kompleks formira iz odnosa ukupnih troškova i zahvaćenih rudnih rezervi koncentracionog horizonta [5]:

$$T_1^{'} = \frac{T_{isk} + T_{pod}}{n \cdot P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}} \text{ , US \$/t ……(5)}$$

gde su:

T_1'–jedinični troškovi izrade, US $/t

T_{isk} – troškovi iskopa, US $

T_{pod} – troškovi podgradjivanja, US $

Troškovi izrade rudnih okana

Rudna okna se rade sa nivoa nižih horizonata prema višim pri čemu se obezbedjuje veza sa OH na svakom horizontu. Troškovi izrade rudnih okana su u direktnoj zavisnosti od karakteristika stena u kojoj je okno locirano, koeficijenta čvrtoće, hidrogeoloških prilika, poprečnog preseka okna, brzine izrade i visine okna. Neosporno je da u tehnici izrade objekata odozdo naviše, važan faktor, od koga zavise troškovi izrade, je visina, odnosno u konkretnom slučaju rastojanje izmedju koncentracionih horizonata.

Uzimajući u obzir i koeficijent sukcesivnog povećanja troškova (q) sa visinom okna, aritmetički izraz ukupnih troškova se formira kao suma parcijalnih troškova po horizontima:

Za I horizont:

$$t_1 = H \cdot N_o \cdot k_2 \text{ , US \$}$$

Za II horizont:

$$t_2 = q \cdot H \cdot N_o \cdot k_2 \text{ , US \$}$$

Za III horizont:

$$t_3 = q^2 \cdot H \cdot N_o \cdot k_2 \text{ , US \$}$$

Za n-ti horizont:

$$t_n = q^{n-1} \cdot H \cdot N_o \cdot k_2 \text{ , US \$}$$

Ukupni troškovi predstavljeni su sumom parcijanih troškova:

$$T_2 = \sum_{i=1}^{n} t_i = \frac{1-q^n}{1-q} \cdot H \cdot N_o \cdot k_2 \text{ , US \$.....(6)}$$

gde su:

q - koeficijent sukcesivnog povećanja troškova sa povećanjem dubine horizonta H ,

N_o – broj potrebnih rudnih okana

k_2 - jedinična cena izrade rudnih okana, US $/m'

Ukupni troškovi izrade rudnih okana predstavljeni su kao zbir članova geometrijske progresije.

Sredjivavanjem predhodnog izraza dobija se aritmetički izraz za specifične troškove ulaganja:

$$T_2^{'} = \frac{(0{,}925 + 0{,}075 \cdot n) \cdot k_2 \cdot N_o}{P_{sr} \cdot \gamma_r \cdot K_{rm}}, \text{ US \$/t ..(7)}$$

Troškovi izrade glavnog izvoznog niskopa (GIN), spiralnog niskopa (SN) i kosog ventilacionog okna (KVO)

Primenom postupka koji je analogan predhodnom dobijaju se izrazi za:

1. ***Troškove izrade glavnog izvoznog niskopa (GIN)***

$$T_3^{'} = \frac{(0{,}925 + 0{,}075 \cdot n) \cdot k_3}{P_{sr} \cdot \gamma_r \cdot K_{rm} \cdot \sin\alpha}, \text{ US \$/t ....(8)}$$

2. ***Troškove izrade spiralnog niskopa(SN)***

$$T_4^{'} = \frac{(0{,}925 + 0{,}075 \cdot n)}{P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}} \cdot \left(\frac{H}{\sin\alpha} \cdot k_4 + L_{SH} \cdot k_4^{'} \right), \text{ US \$/t......(9)}$$

3. ***Troškove izrade kosog ventilacionog okna (KVO)***

$$T_5^{'} = \frac{(0{,}925 + 0{,}075 \cdot n)}{P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}} \cdot \left(\frac{H}{\sin\gamma} \cdot k_5 + L_{PVH} \cdot k_5^{'} \right), \text{ US \$/t....(10)}$$

gde su:

α, β, γ – uglovi nagiba kosih prostorija,°

k_3, k_4, $k_4^{'}$, k_5, $k_5^{'}$ – jedinične cene izrade kosih prostorija (GIN, SN, KVO), US \$/m'

L_{SH}, L_{PVH} – prosečna dužina smernih i prečnih ventilacionih hodnika, m

Troškovi izrade prostorija odvodnjavanja

Jedinični troškovi izrade prostorija u okviru pumpne stanice odnose se na 1 t dobijene rudne mase sa jednog koncentracionog (sabirnog) horizonta i mogu se iskazati izrazom:

$$T_6^{'} = \frac{\sum V_{isk} \cdot C_{isk} + U \cdot L \cdot d \cdot C_b}{n \cdot P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}}, \text{ US \$/t ...(11)}$$

gde su:

V_{isk}, C_{isk} – zapremina, m^3 i cena iskopa, US \$/m^3

U, L – obim i dužina prostorije, m

d – debljina ugradjenog betona, m

C_b – cena armiranog betona, US \$/m^3

Odredjivanje troškova održavanja prostorija

Troškovi održavanja rudarskih prostorija u zavisnosti od jedinične cene odžavanja 1m prostorije u jedinici vremena mogu biti veoma uticajni kod odredjivanja koraka faznog otvaranja. Zavisnost troškova održavanja može se iskazati opštim izrazom [5]:

$$T_{održ.} = f(r, L, t, F, K_s, K_d, K_f), \text{ US \$ ..(12)}$$

gde su:

r – jedinična cena održavanja po 1m' u jedinici vremena, US \$/m'god

L – dužina prostorija, m

t – vreme održavanja, god

F – koristan presek prostorije, m^2

K_s – koeficijent uticaja blizine otkopavanja,

K_d – koeficijent uticaja dubine objekata,

K_f – koeficijent uticaja čvrstoće bočnih stena.

Troškovi održavanja mogu biti predstavljeni kao troškovi ponovnog podgradjivanja i troškovi konstantnog održavanja. Prikazuju se kroz jediničnu cenu održavanja u jedinici vremena.

Troškovi drobljenja rude

Pri faznom otvaranju rudnih ležišta kada se izvoz rude vrši transpoterima sa trakama i skipovima primarno drobljenje rude obavlja se u samoj jami na nivou koncentracionog horizonta.

Troškovi drobljenja za jedan koncentracioni svedeni na tonu dobijene rudne mase iznose [7]:

$$T'_{drob} = \frac{P \cdot n_1 \cdot n_2 \cdot n \cdot t \cdot C_{el}}{Q_e}, \text{ US \$/t} \quad ...(13)$$

gde su:

P – maksimalna snaga drobilice, kW,
n_1 - broj radnih časova u toku 24 h,
n_2 - broj radnih dana u godini,
n - broj objedinjenih horizonata
t – vreme otkopavanja jednog horizonta, god.
C_{el} - cena električne energije, US $/kWh

Imajući u vidu da je vreme eksploatacije jednog horizonta:

$$t = \frac{P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}}{Q_{god}}, \text{ god}$$

i da su eksploatacione rezerve na jednom sabirnom horizontu:

$$Q_e = n \cdot P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}, \text{ t}$$

Jedinični troškovi drobljenja iznose:

$$T'_{drob.} = \frac{P \cdot n_1 \cdot n_2 \cdot t \cdot C_{el}}{Q_{god}}, \text{ US \$/t} \quad ..(14)$$

Zadnji izraz pokazuje da troškovi drobljenja nemaju uticaja na visinu koncentracionog horizonta.

Troškovi izvoza rude

Izvoz rude iz rudnog tela „Borska reka“ vršiće se transporterom sa gumenom trakom. Troškovi izvoza predstavljaju zbir troškova energije sredstava i rada i mogu biti fiksni i promenljivi.

Jedinični troškovi izvoza predstavljaju odnos ukupnih troškova izvoza prema rezervama jednog koncentracionog horizonta [8]. Odnosno:

$$T'_{izv.} = t_f + (t_e + t_a + t_o) \cdot \frac{n \cdot H}{\sin\alpha}, \text{ US \$/t} \quad ..(15)$$

gde su:

t_f – fiksni troškovi izvoza, US $/t
t_e, t_a, t_o – promenljivi troškovi (troškovi energije, amortizacije i održavanja), US $/tm'

Zadnji izraz pokazuje da fiksni troškovi ne utiču na visinu koncentracionog horizonta i oni će u odredjivanju optimalne visine biti konstantne veličine. Znajući to njihovi vrednosni pokazatelji neće biti tretirani u istraživanju konkretnog problema.

Troškovi provetravanja

Pri faznom otvaranju i razradi ležišta mineralnih sirovina kod analize troškova provetravanja najveće učešće ima elektroenergija koja se utroši za kretanje vazduha po rudarskim prostorijama.

S obzirom da će se provetravati samo horizont koji je u eksploataciji, očigledno je da troškovi provetravanja, svedeni na tonu dobijene rudne mase (jedinični) troškovi neće uticati na visinu koncentracionog horizonta. Medjutim, provetravanje nivoa koncentracionog horizonta (hale za drobljenje i ostalih objekata) imaće uticaja na visinu koncentracionog horizonta.
Jedinični troškovi provetravanja iznose:

$$T'_{prov.} = \frac{p \cdot Q^3 \cdot [n \cdot (L_1 + L_2) + L_o]}{Q_{god}}, \text{ US \$/t} \quad (16)$$

gde su:

p – koeficijent koji u sebi sadrži cenu električne energije, koeficijent aerodinamičkog otpora prostorija, oblik i veličinu poprečnog preseka prostorija i ukupni koeficijent korisnog dejstva ventilatora,
Q – količina vazduha potrebna za provetravanje, m^3/s
$L_1 = \frac{H}{\sin\alpha}$ - dužina GIN koja odgovara visini jednog horizonta, m

$L_2 = \dfrac{H}{\sin\gamma}$ - dužina KVO koja odgovara visini jednog horizonta, m.

L_o - ukupna dužina prostorija na koncentracionom horizontu koje treba provetravati, m

Troškovi odvodnjavanja

Pri projektovanju rudnika, ovodnjenost rudnih ležišta izražava se prognoznim koeficijentom ovodnjenosti. Ovakva ocena je prilično uslovna, ali opšte usvojena kod projektovanja jama.

Eksploataciona cena odvodnjavanja je skoro isključivo predstavljena cenom energije, jer amortizacija i lični dohodci za opsluživanje odvodnjavanja obično predstavljaju neznatne veličine. Zbog toga se uzima u obzir samo energija za odvodnjavanje kao stavka troškova koja zavisi od dubine radova, pa je prema tome vezana s visinom koncentracionih horizonata.

Po usvojenoj metodologiji jedinični troškovi odvodnjavanja mogu se sračunati iz izraza:

$$T'_{odv.} = \frac{T_{odv}}{Q_e} = n \cdot H \cdot K_{ov} \cdot K \text{, US \$/t} \quad (17)$$

gde su:

K_{ov} - koeficijent ovodnjenosti, m^3/t

K – pokazatelj, koji u sebi sadrži parametre na osnovu kojih se odredjuje cena energije za odvodnjavanje, svedena na $1m^3$ ispumpane vode na 1m' visine.

ANALIZA DOBIJENIH REZULTATA

Ukupni troškovi, koji su parcijalno razmatrani u predhodnim poglavljima, mogu se izraziti opštom-matematičkom zavisnošću:

$$T_{uk} = T_I(n,H) + \frac{T_{II}}{(n,H)} + T_{III} \text{, US \$/t}$$

Obzirom da se u konkretnom slučaju polazi od usvojene visine H, odredjivanje visine koncentracionog horizonta svodi se na definisanje zavisnosti T = f(n,H) za $H_k = n \cdot H$. Odnosno:

$$f(n) = T_{uk} = T_I \cdot n + T_{II} \cdot \frac{1}{n} + T_{III} \text{, US \$/t} \quad \ldots\ldots\ldots(18)$$

Prvu grupu troškova (T_I) čine tehnološki troškovi i njihove dobijene vrednosti su rezultat projektovanih tehničkih rešenja. Karakterišu se proporcionalnim porastom sa povećanjem visine koncentracionog horizonta. Ovde se posebno ističu troškovi izvoza rude sa nivoa koncentracionog horizonta, zatim troškovi odvodnjavanja, dok je uticaj ostalih troškova manje izražen.

Drugu grupu troškova (T_{II}) predstavljaju neophodna investiciona ulaganja za projektovana rešenja otvaranja i razrade ležišta. Njihova opšta karakteristika je da se sa povećanjem visine koncentracionog horizonta smanjuju. Dominantni uticaj imaju troškovi izrade objekata drobiličnog postrojenja i troškovi izrade objekata odvodnjavanja.

Troškovi T_{III} nemaju uticaja na visinu koncentracionog horizonta i njihova vrednost je predstavljena konstantnom veličinom.

Suma ukupnih troškova predstavljena je funkcijom paraboličnog oblika, a jednaka je zbiru funkcija linearnog i hiperboličnog oblika.

Za uslove otvaranja dubljih delova rudnog tela ”Borska Reka”, zamenom vrednosti u izvedenim formulama, dobijena je matematička zavisnost [5]:

$$F(n) = T_{uk} = 0{,}049 \cdot n + 0{,}072 \cdot \frac{1}{n} + 0{,}076 \text{, US \$/t} \quad \ldots\ldots(19)$$

Funkcija je neprekidna i ima prvi izvod:

$$f'(n) = \frac{df(n)}{dn} = T_I - \frac{T_{II}}{n^2}$$

Drugi izvod funkcije je:

$$f''(n) = \frac{2 \cdot T_{II}}{n^3}$$

Znak drugog izvoda pokazuje da funkcija ima minimum za sledeću vrednost (n):

$$T_I - \frac{T_{II}}{n^2} = 0; \Rightarrow n_o = \sqrt{\frac{T_{II}}{T_I}} = 1{,}212 \ldots (20)$$

Optimalna visina koncentracionog horizonta, odnosno korak nove faze otvaranja dubljih delova rudnog tela ”Borska reka” iznosi:

$$H_k = n_o \cdot H = 1{,}212 \cdot 80 = 96{,}96\, m$$

Odnosno, za otvaranje dubljih delova rudnog tela „Borska reka“, svaki horizont može biti koncentracioni ili sabirni horizont, ali sa uvećanom visinom od 96,96 m.

ZAKLJUČAK

U radu je primenom matematičko – analitičke metode razmatrana mogućnost faznog otvaranja dubljih delova rudnog tela ”Borska reka“. Izbor najpovoljnijeg rešenja je tražen na bazi optimizacije svih značajnih troškova, a pre svega troškova izrade jamskih prostorija, njihovog održavanja, a potom i troškova izvoza, provetravanja, odvodnjavanja i dr. Neki od ovih troškova su proporcionalni, a neki obrnuto proporcionalni visini koncentracionog horizonta što stvara mogućnost njihove optimizacije.

Dobijeni matematički izrazi, koji predstavljaju funkcionalne zavisnosti najznačajnih parametara i ukupnih troškova, koji su uzeti kao relevatni, imaju dovoljno uopšten oblik da se mogu lako primeniti i u drugim prilikama različitim od uslova u rudnom telu ”Borska reka“.

Za konkretne uslove u rudnom telu ”Borska reka“ dobijena je optimalna visina koncentracionog horizonta od 1,212 H, odnosno 97 m. Očigledno je da se ne može usvojiti visina koncentracionog horizonta jednaka dvostrukoj visini horizonta, ali bi korisno bilo da se poveća visina horizonta, ukoliko bi to odgovaralo tehnologiji eksploatacije.

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PHASE DEVELOPMENT OF THE BORSKA REKA ORE BODY**

Abstract

This paper gives the main parameters of opening and development of the "Borska reka" ore body. It has been developed to K-235 m level, while its bottom contour, within 0.3 % Cu limit grad, reaches K-995 m level [1]. Development of lower parts of the ore body could be performed in levels, block by block, or in phases, with several blocks joined into a single main level. Using the mathematical-analytical method, the optimum level height is determined for phase development of lower part of the Bor deposit.

Key words: *development, main levels, costs, optimization*

INTRODUCTION

The area of the "Borska reka" ore body is a continuation of the Bor hydrothermally altered zone of Bor in the northwest direction. This 1,500 m long and 800 m deep area is inclined towards west at 45° to 55° dip angle. This inclination matches the inclination of sandstones and conglomerates in this zone, separated from hydrothermally altered zone by the Bor Fault [2].

The "Borska reka" ore body lies very deep below the terrain surface. Its depth increases towards west and northwest. In general, the ore body depth in 0.3 % Cu contour varies between 600 and 850 m, with average thickness of 300 m. According to the exploration drilling data, the closest mineralization to the surface was at 398 m depth, in a drillhole B-38, while the deepest one was at 1,402 m, in a drillhole B-33 [3].

Previous exploration works have shown that the Borska reka ore body belongs to a group of massive ore bodies. It is 1,520 m long, 635 m wide with average thickness of 300 m. Maximum area, at level K-560 m in a contour 0.3% Cu reaches over 500,000 m^2. [3]

The ore reserves were calculated using the horizontal profile method. The areas were calculated by 80 m high levels which matched the supposed excavating levels. In 0.3 % Cu contour, total ore reserves are 623,637,494 t, with average grade of

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0.608 % Cu, while total copper reserves are 3,799,447 t. [4]

DEVELOPMENT OF THE ORE BODY BORSKA REKA TO K-235 M LEVEL

The ore body “Borska reka” is situated bellow current mining works in the underground mine “Jama” Bor and reaches the depth of 1,200 m. It is partially developed in the Main Level XIX, at K-235 m level.

From the XIII level of the new hoisting shaft at K-21 m level, 750 m long hoisting slope (GIN) was driven down to K-235 m level, with 16.5^{o} inclination. At K-235 m level, the slope is connected to the main haulage drift (GTH-235m). This drift is 780 m long and represents a connection to the central orepass (CRO) [1].

The ore body is connected with service shaft by 560 m long exploration slope. From the slope, the exploration drifts were driven towards the ore body contours at K-155 m level [1].

At Level XVII (K-155 m), the main transport drift is driven all along the ore body. It is used for transport of excavations from opening and development drifting. In the latter phase of mining, this drift will be used as a service drift for Level XVII.

Levels XVII and XIX are connected by ventilation shaft (PVO) and central orepass (CRO) (Figure 1).

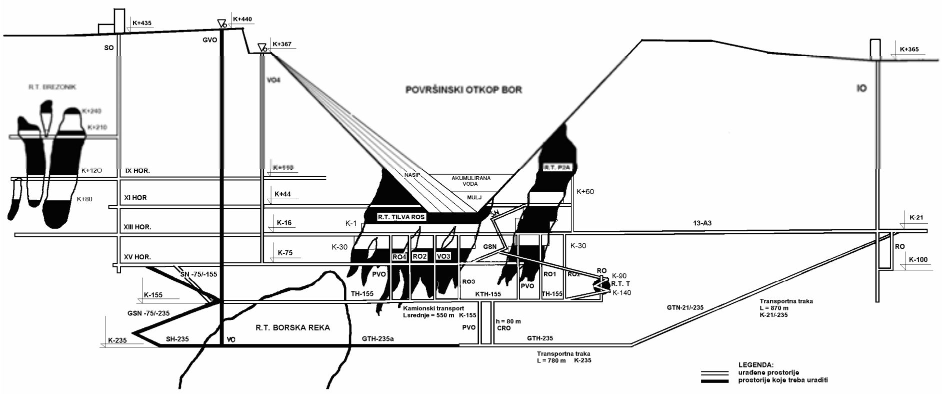

Fig. 1. *Scheme of development the ore body “Borska reka” to K-235 m level*

The basic idea in opening and development design is to use the existing drifts as much as possible, in order to reduce the investments. However, there are some new drifts that have to be driven, such as:

- Main service slope (GSN-75/-235), from the service shaft discharge point at K-75 m level, 816 m long with 11.3^{o} inclination;
- Main haulage drift (GTH-235a), 756 m long. This drift will be used for development theLevel XIX, and it will be driven at K-235 m level, as a continuation of GTH-235 drift, designed to pass near the ore body and connect to GSN-75/-235 slope;
- Main ventilation shaft (GVO-155/-235). Ventilation of the ore body includes the new main ventilation shaft

from K+110 level at Level IX to the ground surface. This shaft should replace the existing VO_4 shaft, which is situated in the slope of the old open pit and endangered by the ground movement process. After that, the new shaft would be deepened from K+110 m to K-155 m and connected to drifts driven from the ore body at that level. Between Levels XV and XIII, VO_4 would be in the influence zone of mining works in the "Borska reka" and endangered by caving.

These facilities include the opening and development of the ore body up to the level -235. Lower part of the ore body "Borska reka", below the level -235, will be opened in the next phase of exploitation

DEVELOPMENT OF THE ORE BODY BORSKA REKA BELLOW K-235 M LEVEL

In order to determine the height of next development stage, e.g. block height, the mathematical-analytical method will be used. The principle of this method is to establish the equations for calculation the costs of mining works per ton of ore, but only the mining works which are dependable on development design. These costs are represented by function T = f (n, H). This function represents the sum of three types of costs, and its off – peak will determine the optimum block height, $H_k = n\,H$, m.

Solving of this problem has started from the necessary assumption related to the establishment of functional relations. It is necessary to define each vertical mining facilities (shaft) through the known parameter, Level height H, which is, in a mathematical sense, a step of iteration [5].

Depth of vertical mining facilities (H_k) between two adjacent levels, for designed level height (H), could be defined by the following relations:

- For single level, $H_{k_1} = H$
- For two levels, $H_{k_2} = 2 \cdot H$
- For three levels, $H_{k_3} = 3 \cdot H$
- For (n-1) levels, $H_{k_{(n-1)}} = (n-1) \cdot H$
- For n levels, $H_{k_n} = n \cdot H$

The last relation shows that a depth of vertical facilities is defined as the product of Level height (H) and number of Levels included in the Main Level, which means that optimization is focused to determination of number "n" [5].

In determination of optimum height of the Main Level, the investments and other costs are given as specific investment costs, per production unit (e.g. per ton of ore).

Specific investment costs are calculated as a relation between sum of investments and developed ore reserves. The ore reserves will be defined by parameters of Level, and they present a step of iteration in mathematical model. Exploitable reserves of a single Level are [6]:

$$Q_e = \frac{Q_g \cdot K_{ir}}{1 - K_{or}} = Q_g \cdot K_{rm} \text{, t……(1)}$$

where:

Q_g – geological ore reserves of Level, t,

K_{ir} – ore recovery ratio,

K_{or} – ore dilution ratio,

K_{rm} – ore mass ratio.

Geological reserves of Level are:

$$Q_{e1} = P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm} \text{, t…………(2)}$$

where:

P_{sr} – average area between two Levels, m^2,

H – Level height, m,

γ_r – ore density, t/m^3.

The ore reserves in a specific phase of extraction are defined as a product of number of Levels (n) included in the Main Level and ore reserves in a single Level (Q_{e1}):

$$Q_e = n \cdot Q_{e1} = n \cdot P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm} \text{, t…(3)}$$

Specific investments costs, or costs per unit, are:

$$T_i^{'} = \frac{T_i}{Q_e} \text{ , US \$/t} \qquad (4)$$

where:

T'$_i$ – specific investment costs, US $/t,

T_i – sum of investments, US $,

Q_e – mining reserves of the Main Level, t.

COSTS OF A SINGLE DEVELOPMENT PHASE

Crusher room costs

The rusher room is a specific facility, designed to match demands of ore crushing and hoisting. Crusher room costs are a sum of excavation costs and supporting costs.

Since the ore crushing takes place in each Main Level, the investment costs per unit are defined as a relation between sum of costs and ore reserves in the Main Level [5]:

$$T_1^{'} = \frac{T_{isk} + T_{pod}}{n \cdot P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}} \text{ , US \$/t……(5)}$$

where:

T_1' – crusher room costs per unit, US $/t,

T_{isk} – excavation costs, US $/t,

T_{pod} – support costs, US $/t.

Orepass driving costs

The ore passes are driven from lower levels towards upper, thus providing connection with operational drifts at each Level. Costs of orepass driving are directly dependable on rock properties of a location, hydrogeology, orepass diameter, driving speed and orepass height.

Since the most important factor is the orepass height, calculation of costs takes the coefficient of successive cost increase with height into account. Total costs are the sum of costs by Levels:

For Level I

$$t_1 = H \cdot N_o \cdot k_2 \text{ , US \$/t,}$$

For Level II

$$t_2 = q \cdot H \cdot N_o \cdot k_2 \text{ , US \$/t,}$$

For Level III

$$t_3 = q^2 \cdot H \cdot N_o \cdot k_2 \text{, US \$/t,}$$

For Level n

$$t_n = q^{n-1} \cdot H \cdot N_o \cdot k_2 \text{ , US \$/t.}$$

Total costs are the sum of partial costs:

$$T_2 = \sum_{i=1}^{n} t_i = \frac{1-q^n}{1-q} \cdot H \cdot N_o \cdot k_2 \text{, US \$/t...(6)}$$

where:

q – coefficient of successive cost increase with increase of Level height (H),

N_0 – number of ore passes,

k_2 - unit price of orepass driving per meter, US $/m'.

Total costs of orepass driving are presented as a sum of relations of geometric progression.

The arithmetic expression of the specific investment costs per unit is derived after development the previous expression:

$$T_2^{'} = \frac{(0{,}925 + 0{,}075 \cdot n) \cdot k_2 \cdot N_o}{P_{sr} \cdot \gamma_r \cdot K_{rm}} \text{,US/t...(7)}$$

Driving costs for the main hoisting slope (GIN), ramp (SN) and inclined ventilation shaft (KVO)

With the similar procedure as previous, the following expressions are derived:

1. Driving costs for the main haulage slope (GIN)

$$T_3^{'} = \frac{(0.925 + 0.075 \cdot n) \cdot k_3}{P_{sr} \cdot \gamma_r \cdot K_{rm} \cdot \sin\alpha} \text{ , US \$/t…(8)}$$

2. Driving costs for the ramp (SN)

$$T_4^{'} = \frac{(0.925 + 0.075 \cdot n)}{P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}} \cdot \left(\frac{H}{\sin\alpha} \cdot k_4 + L_{SH} \cdot k_4^{'} \right) \text{, US \$/t…(9)}$$

3. Driving costs for the inclined ventilation shaft (KVO)

$$T_5^{'} = \frac{(0.925 + 0.075 \cdot n)}{P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}} \cdot \cdot \left(\frac{H}{\sin \gamma} \cdot k_5 + L_{PVH} \cdot k_5^{'} \right), \text{ US \$/t.... (10)}$$

where:

α, β, γ – inclination angles of drifts, °,

k_3, k_4, k_4’, k_5, k_5’ – driving costs per meter, US $/m’,

L_{SH}, L_{PVH} – average length of ventilation drifts, m.

Driving costs for dewatering drifts

Costs per unit for driving of drift within the pump station are related to 1 t of the obtained ore mass from Main the Level and they can be calculated using the expression:

$$T_6^{'} = \frac{\sum V_{isk} \cdot C_{isk} + U \cdot L \cdot d \cdot C_b}{n \cdot P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}}, \text{ US \$/t..(11)}$$

where:

V_{isk} – volume of excavation, m^3,

C_{isk} – costs of excavation, US $/$m^3$,

U, L – circumference and length of a working, m,

d – thickness of built concrete, m,

C_b – price of reinforced concrete, US $/$m^3$.

Maintenance costs

Maintenance costs for mining drifts depend on the unit costs per meter in a specific time. These costs might have a significant influence to total costs and, furthermore, to development design. Determination of maintenance costs can be expressed by the following expression [5]:

$$T_{održ.} = f(r, L, t, F, K_s, K_d, K_f), \text{ US \$ ...(12)}$$

where:

r – maintenance costs per meter in a specific time period, US $/m’ year,

L – drift length, m,

T – maintenance time, year,

F – area of drift cross – section, m^2,

K_s – coefficient of influence of nearby excavations,

K_d – coefficient of influence of drift depth,

K_f – coefficient of influence of side rock.

Maintenance costs could be presented as the costs of re-supporting and costs of constant maintenance.

Ore crushing costs

In a case of phase development, when the ore hoisting is performed by conveyor belts and skips, the primary crushing is carried out underground, on the bottom of the Main Level.

The ore crushing costs for the Main Level per ton of ore are [7]:

$$T_{drob}^{'} = \frac{P \cdot n_1 \cdot n_2 \cdot n \cdot t \cdot C_{el}}{Q_e}, \text{ US \$/t ......(13)}$$

where:

P – maximum crusher power, kW,

n_1 – working hours per day,

n_1 – working hours per year,

n – number of Levels included in the Main Level,

t – excavation time of a single Level, years,

C_{el} – electric energy costs, US $/kWh.

Extraction time of a single Level is:

$$t = \frac{P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}}{Q_{god}}, \text{ years}$$

Mining ore reserves of the Main Level are:

$$Q_e = n \cdot P_{sr} \cdot H \cdot \gamma_r \cdot K_{rm}, \text{ t}$$

Considering the previous relations, the ore crushing costs are:

$$T_{drob.}^{'} = \frac{P \cdot n_1 \cdot n_2 \cdot t \cdot C_{el}}{Q_{god}}, \text{ US \$/t .....(14)}$$

This means that the ore crushing costs have no influence on determination the Main Level height.

Ore hoisting costs

Ore hoisting from the ore body "Borska reka" will be carried out by belt conveyors. Hoisting costs are the sum of energy, material and labor costs and they can be fixed or variable.

Hoisting costs per unit present a relation of total hoisting costs and ore reserves of the Main Level [8]:

$$T'_{izv.} = t_f + (t_e + t_a + t_o) \cdot \frac{n \cdot H}{\sin\alpha}, \text{ US \$/t ..(15)}$$

where:

t_f – fixed hoisting costs, US $/t,

t_e, t_a, t_o – variable costs (energy, depreciation and maintenance), US $/t m'.

The last expression shows that the fixed costs have no influence on determination the Main Level height, as they remain constant. Knowing that, these costs will be excluded from the optimization.

Ventilation costs

In cost analysis of phase mine development, one of the most important parameters is consumption of electric energy, used for underground air flow process.

Since only the active Level will be ventilated, it is obvious that ventilation costs per ton of ore wouldn't influence the Level height determination. On the other hand, ventilation of crusher room and other objects at Main Level has to be taken in consideration.

Ventilation costs per unit are:

$$T'_{prov} = \frac{p \cdot Q^3 \cdot [n \cdot (L_1 + L_2) + L_o]}{Q_{god}}, \text{ US \$/t ...(16)}$$

where:

p – coefficient related to the energy costs, drift aerodynamic resistance, shape and area of drift cross – section and overall coefficient of fan efficiency,

Q – necessary fresh air volume for ventilation, m^3/s,

$L_1 = \frac{H}{\sin\alpha}$ - length of the main hoisting slope in a Level, m,

$L_2 = \frac{H}{\sin\gamma}$ - length of inclined ventilation shaft in a Level, m,

L_o - sum of drift lengths in the Main Level, m.

Dewatering costs

In the mine designing, volume of underground water in deposit is expressed through evaluated coefficient of underground water. Such evaluation is pretty conditional, but it is generally accepted in the mine designing.

In the structure of dewatering costs, the electric energy costs are dominant. Depreciation and labor costs usually have much less influence to overall costs. Therefore, only the energy costs are taken into consideration.

According to the adopted methodology, the unit costs of dewatering can be calculated by the following expression:

$$T'_{odv.} = \frac{T_{odv}}{Q_e} = n \cdot H \cdot K_{ov} \cdot K, \text{ US \$/t (17)}$$

where:

K_{ov} – coefficient of underground water, m^3/t,

K – parameter used to determine the energy costs per unit, e.g. the energy costs needed for pumping 1 m^3 of water to 1 m height.

ANALYSIS OF THE OBTAINED RESULTS

Total costs, partially analyzed in the previous chapters, can be expressed by the general mathematical dependence:

$$T_{uk} = T_I(n,H) + \frac{T_{II}}{(n,H)} + T_{III}\text{, US \$/t}$$

Since the height of a single Level H is fixed, definition of the Main Level height can be expressed through relation T = f(n, H), for $H_k = n \cdot H$.
That is,

$$f(n) = T_{uk} = T_I \cdot n + T_{II} \cdot \frac{1}{n} + T_{III}\text{, US \$/t} \ldots(18)$$

The first group of costs, T_I, is technological costs, related to designed technical solutions. Their increase is proportional to increase of the Main Level height. This group of costs includes the hoisting costs, dewatering costs and some other costs with lesser influence.

The second group of costs, T_{II}, is the necessary investments for designed solutions of opening and development of the deposit. Their general characteristic is to be reduced with the increase of Level height. Costs of crusher room and pump station drifting are dominant in this group.

The third group of costs, T_{III}, has no influence on a Level height and their value is present by the constant.

Total sum of costs can be presented as a parabolic function, and it is similar to a sum of linear and hyperbolic functions.

For parameters of the ore body "Borska reka", mathematical function of costs is [5]:

$$F(n) = T_{uk} = 0.049n + 0.072\frac{1}{n} + 0.076\text{, US\$/t} \ldots(19)$$

This function is a continous and its first derivative is:

$$f'(n) = \frac{df(n)}{dn} = T_I - \frac{T_{II}}{n^2}$$

The second derivative of this function:

$$f''(n) = \frac{2 \cdot T_{II}}{n^3}$$

This means that the function has minimum for the following value (n):

$$T_I - \frac{T_{II}}{n^2} = 0; \Rightarrow n_o = \sqrt{\frac{T_{II}}{T_I}} = 1.212 \ldots(20)$$

Optimal height of the Main Level, respectively a step of development phase of the ore body is:

$$H_k = n_o \cdot H = 1.212 \cdot 80 = 96.96\,m$$

This means that in development of the ore body "Borska reka", each Level could be the Main Level, but with the increased height from 80 m to 96.96 m.

CONCLUSION

This paper gives the mathematical-analytical method for optimum development design in the ore body "Borska reka". All relevant costs were analyzed and optimized. The most important are the costs of drifting, maintenance, hoisting, ventilation and dewatering. Some of these costs are directly proportional to the Level height, while some of them are inversely proportional, what makes the possibility for their optimization.

The final relations are the functional dependences of the most important parameters and overall costs, and they can be generalized and applied to the other mines, with different properties from this case.

For parameters of the ore body "Borska reka", according to this methodology, the optimum Main Level height is 1.212·H, or 97 m. Since it is impossible to involve two 80 m Levels into one Main Level, it would be useful to increase the Level height from 80 m to 97 m, if it is suitable to the mining technology.

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OTKOPAVANJE SIGURNOSNIH PLOČA U RUDNIKU ''TREPČA'' – STARI TRG

Izvod

U podzemnoj eksploataciji problem sekundarne ekspolatacije, tj. otkopavanje sigurnosnih ploča naročito je izražen kod onih ležišta gde se primarna eksploatacija vrši metodama krovnog otkopavanja sa zasipavanjem otkopnih prostora, kao što je slučaj u rudniku ''Trepča'' – Stari Trg. U primarnoj fazi eksploatacije ostavljane su sigurnosne ploče moćnosti 6 – 8 metara. U sekundarnoj fazi eksploatacije planirano je otkopavanje sigurnosnih ploča čije se površine kreću u granicama od 100 – 4.000 m², i one sada predstavljaju najproduktivniju zonu trepčanskog ležišta, kako po sadržaju metala tako i po količini rudne mase.

U ovom radu prikazan je modifikovani način eksploatacije sigurnosnih ploča, uz primenu sistematskog podgrađivanja unakrsnim drvenim slogovima.

Ključne reči: *podzemna eksploatacija, sigurnosne ploče, podgrađivanje.*

UVOD

Za otkopavanje ležišta ''Trepča'' – Stari Trg, u primarnoj fazi eksploatacije primenjuje se ''Trepčanska metoda'' tj. metoda krovnog otkopavanja u horizontalnim etažama sa zasipavanjem otkopnog prostora. Ovom metodom se otkopavaju sva rudna tela od nivoa horizonta sa kojeg počinje otkopavanje do na 6 - 8 metara ispod nivoa višeg horizonta, odnosno do granice sigurnosne ploče.

U početnoj fazi eksploatacije za otkopavanje sigurnosnih ploča primenjivala se metoda kvadratnih slogova u horizontalnim etažama. Intenzitet otkopavanja kod ove metode je izrazito nizak, tako da je s vremenom stvaran sve veći raskorak kod otkopavanja u primarnoj i sekundarnoj fazi eksploatacije.

Ležište Trepča, sastoji se od niza cevastih rudnih tela nepravilnog oblika (Sl. 1). Najveći broj rudnih tela izgrađuju sulfidni minerali, a manji deo ležišta čine tzv. oligonitna rudna tela, izgrađena od gvožđevito – magmatskih karbonata sa većim ili manjim sadržajem olovo – cinkanih sulfida.

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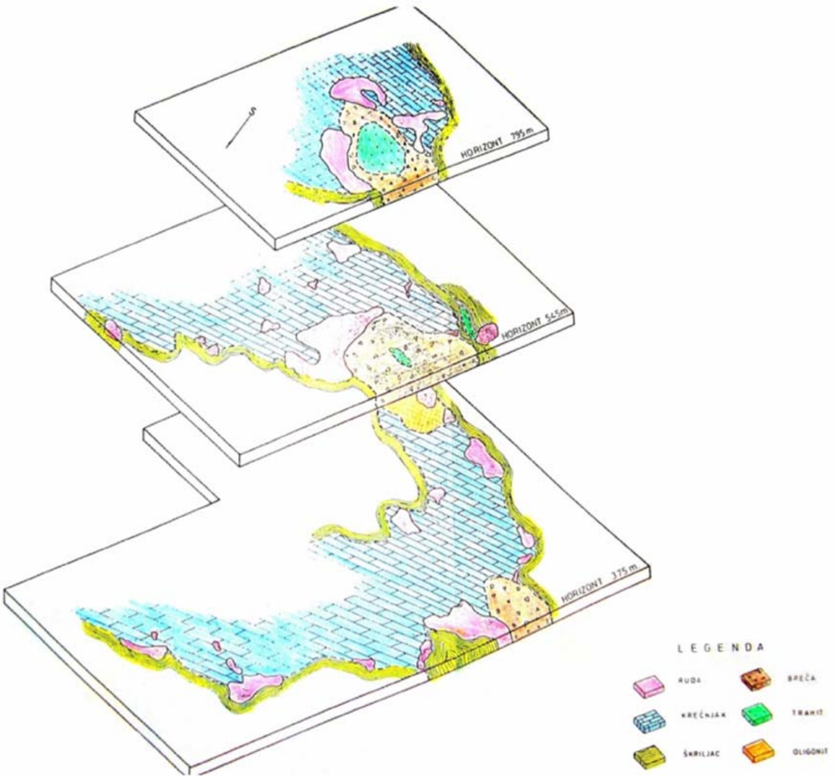


Sl. 1. *Blok prikaz Trepčanskog ležišta*

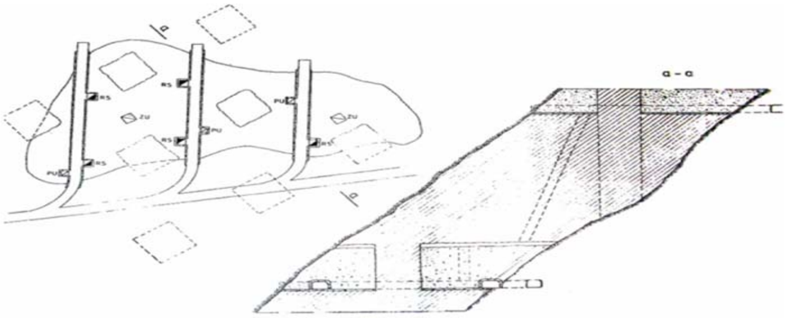
Laboratorijskim ispitivanjem uzoraka, uzorkovanih u jami ''Trepča'' – Stari Trg, utvrđene su sledeće fizičko – mehaničke karakteristike rude i pratećih stena, koje su prikazane u tabeli br. 1.

Tabela 1. *Fizičko – mehaničke karakteristike rude i pratećih stena*

RUDA / STENA	γ_1 [t/m^3]	γ_2 [t/m^3]	σ_c MPa	σ_i MPa	f	c MPa	ν
Sulfidi	4,26	3,70	78,0	5,9	7,80	12,3	0,19
Oligoniti	3,67	3,48	82,1	7,4	7,43	13,6	0,19
Krečnjak	2,86	2,80	49,5	5,0	5,27	8,8	0,17
Škriljac	2,83	2,76	44,1	6,6	4,45	8,6	0,17
Breča	3,00	2,90	60,9	6,4	6,08	10,9	0,17

OTKOPAVANJE LEŽIŠTA

Metoda krovnog otkopavanja u horizontalnim etažama sa zasipavanjem se primenjuje od samog početka eksploatacije ležišta. Njome su se otkopavala sva rudna tela u primarnoj fazi eksploatacije (sl. 2).



Sl. 2. *Trepčanska metoda otkopavanja*

Pripremni radovi tj. glavni izvozni hodnik se izrađuje u podinskom boku, a rudno telo se preseca prečnim hodnicima čiji broj zavisi od veličine otkopa. Prečnim hodnicima rudno telo se preseca do krovinskog kontakta, a zatim se iz njih vrši otkopavanje rudnog tela po čitavoj površini.

Sa nivoa horizonta počinje otkopavanje prve etaže čija visina iznosi 5,5 m. Bušenje kratkih minskih bušotina se vrši bušaćim čekićem RK – 28. Utovar rude se vrši direktno u vagonete sa CAVO utovarnom lopatom. Kada se otkopa ruda po celoj površini, delovi prečnog hodnika koji su bili u rudi izrađuju se u betonskoj oblozi, a ujedno se rade i rudne i rudno – prolazne sipke. Radi zasipavanja otkopa, izrađuju se u rudi zasipni uskopi do višeg horizonta. Sa nivoa višeg horizonta doprema se zasip koji se razastire po otkopu. Pod krovom otkopa ostavlja se otvorena visina od 2 m radi ventilacije i komuniciranja u otkopu. Otkopavanje sledeće etaže počinje od sipke prema krovinskom delu otkopa.

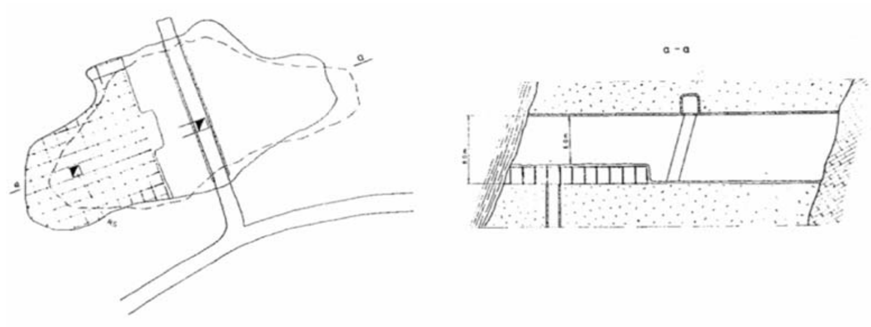
Visinska razlika između horizonata iznosi 60 m. Za rudna tela u rudniku Trepča, velikih površina i čiji je raspon od podinskog do krovinskog dela veliki, kao privremeno sredstvo osiguranja otkopa u primarnoj fazi eksploatacije, ostavjaju se sigurnosni stubovi raspoređeni u šahovskom poretku, dimenzija 10 x 10 m. Rastojanje između redova kreće se od 12 - 16 m, a rastojanje između stubova u redu 16 - 20 m. Visina sigurnosnih stubova kreće se od 10 - 70 m, u zavisnosti od moćnosti rudnog tela.

OTKOPAVANJE SIGURNOSNIH PLOČA

Dugi niz godina sigurnosne ploče su se otkopavale metodom kvadratnih slogova koja ima veoma nizak intenzitet otkopavanja i dovodi do sve većeg zaostajanja sekundarne u odnosu na primarnu fazu otkopavanja.

Metoda je šematski prikazana na (sl. 3), a princip rada se sastoji u tome što se u otkope ugrađuje specijalno pripremljena kvadratna podloga i to neposredno posle otkopavanja jednog manjeg prostora.

Prvi red kvadratne podgrade se postavlja na temeljače, a svaki naredni se nadovezuje na prethodni. Podgrada treba da se slaže i vezuje pravilno, stubovi moraju biti postavljeni vertikalno, a stropnice i prečage moraju nalegati na stubove. Zatezanje podgrade o strop i bokove otkopa vrši se samo na sastavima.



Sl. 3. *Metoda kvadratnih slogova*

Sve ostale radne operacije su iste kao kod metode krovnog otkopavanja sa zasipavanjem samo sa smanjenim obimom radova i učincima. Metodu karakterišu veoma niski učinci u svim fazama procesa rada, a samim tim se postiže nizak intenzitet otkopavanja.

Sa napredovanjem rudarske tehnike i uvođenjem savremene mehanizacije, zadnjih godina se pokušalo sa otkopavanjem sigurnosnih ploča primenom metode unakrsnih drvenih slogova. To je u stvari modifikacija horizontalnog krovnog otkopavanja uz primenu sistematskog podgrađivanja unakrsnim drvenim slogovima. Metoda se pokazala znatno efikasnija u poređenju sa metodom kvadratnih slogova, a takođe i dovoljno sigurna.

Uzimajući u obzir dosadašnje iskustvo pri otkopavanju sigurnosnih ploča, može se konstatovati da je ova metoda vrlo efikasna i sigurna, a po intenzitetu ne zaostaje mnogo za ''trepčanskom metodom'', pošto je moguće mehanizovati radove kako na otkopavanju tako i na zasipavanju otkopa što nije bio slušaj sa metodom kvadratnih slogova.

U konkretnom slučaju, kao podgrada se koriste unakrsni drveni slogovi (sl.4), dimenzija 2x2 m, a njihova visina u momentu kada je etaža otkopana iznosi 2,5 m. Radi nesmetanog kretanja mehanizacije, rastojanje između slogova u redu i između redova treba da iznosi 2,5-3 m. Ako se usvoji pomenuto rastojanje 3,0 m i posmatra slučaj raspona ploče od 18 m, onda se u redu mogu postaviti 4 unakrsna sloga, a na površini 18 x 18 m, to je 4 x 4 odnosno 16 unakrsnih slogova. Ovako raspoređeni slogovi, suprostavljahu se pritisku iz krova otkopa tj. treba da spreče deformaciju ploče.

Ovako izabrana metoda otkopavanja sigurnosnih ploča, u suštini predstavlja nastavak ''trepčanske metode'', pri otkopavanju zadnjih 8 m, rudnog ploče između dva horizonta. Izrada posebnih pripremnih radova za ovu metodu nije potrebna, jer se koriste objekti pripreme izvedeni za primarnu fazu eksploatacije.

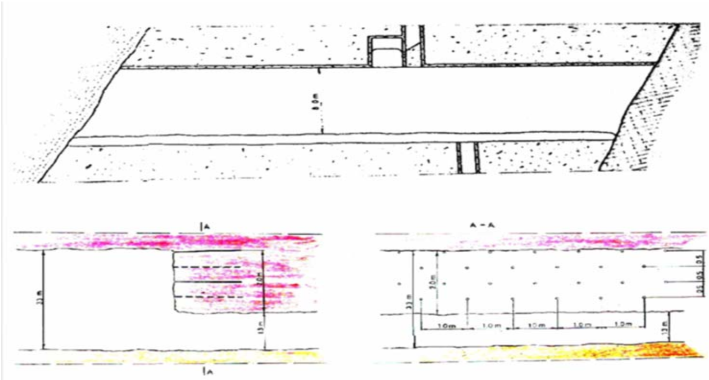
Sve faze tehnološkog procesa otkopavanja obavljaju se istim redosledom kao i kod normalnog rada, a jedina razlika je u fazi podgrađivanja otkopa koji se mora obavljati određenim redosledom i sa posebnom pažnjom. Naime, podgrađivanje mora sukcesivno da prati otkopne radove i maksimalna otvorena površina bez stalne podgrade ne sme da bude veća od 25 m^2.

Otkopavanje će se obavljati u etažama visine 2 m, i ono otpočinje od rudne sipke najbliže krovnom boku. U prvoj fazi napredovanje se vrši ka krovinskom boku, a kada se do njega dođe, onda se sa izvesnim prethođenjem uz bok formira otkopni front za dalje napredovanje.

Napredovanje na otkopnom frontu vrši se horizontalnim bušenjem i miniranjem u otsecima širine 5 m. Napredovanje jednog otseka na frontu, posmatrano po dubini, takođe može iznositi do 5 m, tako da ukupna otvorena površina ne prelazi 25 m^2, posle čega se mora postaviti unakrsni drveni slog.

Obaranje rude vrši se kratkim minskim bušotinama dužine 1,6 m, i prečnika 32 mm. Bušenje se vrši lakim bušaćim čekićima RK – 28.

Raspored minskih bušotina prikazan je na (Sl. 4), u šahovskom rasporedu sa rastojanjem 1,0 m, između bušotina i 0,5 m, između redova. Na frontu sigurnosne ploče širine 5 m, i visine 2 m, zabušićese 4 reda ili ukupno 18 minskih bušotina.



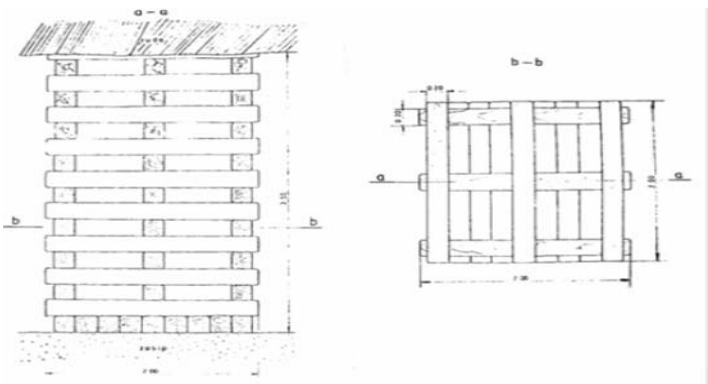
Sl. 4. *Prikaz otkopnog fronta sa rasporedom minskih bušotina*

Odminirana ruda se se utovara i odvozi do sipke samohodno utovarno–transportnom mašinom T_2GH, koja ima zapreminu kašike od 0,12 m^3 i zapreminu sanduka 0,75 m^3.

Sistematsko podgrađivanje unakrsnim drvenim slogovima podrazumeva da se nakon otkopavanja jedne sekcije maksimalne površine od 25 m^2, mora postaviti unakrsni drveni slog, tek posle čega može započeti otkopavanje naredne sekcije.

Slogovi se postavljaju u centar svake otkopne sekcije, tako da se na otkopu formira mreža unakrsnih slogova na međusobnom rastojanju od 3 m, u redu, i istom rastojanju između redova slogova.

Način postavljanja svakog unakrsnog sloga je sledeći: Na zasip prethodne etaže, koji se na mestu postavljanja sloga mora izravnati, poređaju se 9 gusto složenih greda. Zatim se na ovako stvorenu osnovu slažu unakrsno po 3 grede u redu i to sve do stropa otkopa kao što je prikazano na (sl. 5).



Sl. 4. *Podgrađivanje unakrsnim drvenim slogovima*

Zatezanje postavljenog unakrsnog sloga vrši se drvenim klinovima, koje sa svih strana treba podjednako nabijati, da bi se slog ravnomerno zategao o strop otkopa. Slogovi u redu i redovi slogova se međusobno povezuju podvlakama dužine 3,5 m, a podvlake se o strop otkopa zatežu takođe klinovima. Ovako postavljeni i međusobno povezani unakrsni drveni slogovi predstavljaju čvrstu celinu koja se može suprostaviti pritiscima iz stropa otkopa.

Posle završetka otkopnih radova na celoj etaži, ili na delu etaže koji može da predstavlja tehnološku celinu, i nakon podizanja rudnih sipki, prolaznog uskopa i uskopa za zasip, pristupa se fazi zasipavanja otkopa. Zasipni materijal se sa višeg horizonta preko zasipnog uskopa doprema u otkop i skreperima razvlači između redova unakrsnih drvenih slogova. Mreža unakrsnih drvenih slogova, kojima je otkop osiguran, ostaje u zasipu služeći kao osnova novom slogu koji će biti postavljen pri otkopavanju naredne etaže. Otkopavanje preostalog fronta sigurnosne ploče, vrši se na isti način kao što je ovde i opisano.

ZAKLJUČAK

Za otkopavanje sigurnosnih ploča u rudniku ''Trepča'' – Stari Trg, u sekundarnoj fazi eksploatacije ranije se koristila metoda kvadratnih slogova koja je imala veoma nizak intenzitet otkopavanja i dolazilo je do sve većeg zaostajanja sekundarne u odnosu na primarnu fazu otkopavanja.

U radu je prikazan izbor nove modifikovane metode horizontalnog krovnog otkopavanja uz primenu sistematskog podgrađivanja unakrsnim drvenim slogovima. Metoda se pokazala znatno efikasnija u poređenju sa metodom kvadratnih slogova, a takođe i dovoljno sigurna.

Uzimajući u obzir dosadašnje iskustvo pri otkopavanju sigurnosnih ploča, može se zaključiti da je ova metoda vrlo efikasna i sigurna, a po intenzitetu ne zaostaje ni malo za ''trepčanskom metodom'', pošto je moguće mehanizovati sve radove kako na otkopavanju tako i na zasipavanju otkopa, što nije bio slušaj sa metodom kvadratnih slogova.

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MINING OF SAFETY PLATES IN THE MINE ''TREPČA'' – STARI TRG

Abstract

The problem of secondary mining in the underground mining, i.e. excavation of safety plates is particularly expressed in those deposits where the primary mining is done using the methods of roof caving with back filling, as in the case in the mine "Trepča" – Stari Trg. In the primary stage of exploitation, the plates thickness 6 – 8 m were left. In the secondary stage of exploitation, the excavation of safety plates was planned with surface areas within the limits of 100- 4,000 m^2, and they now represent the most productive zone of the Trepča deposit, both in terms of metal content and the amount of ore mass.

This paper presents the modified mining method of safety plates using the systematic supporting with wooden chocks.

Key words: *underground mining, safety panels, supporting*

INTRODUCTION

The "Trepča" method, i. e. method of roof caving in horizontal floors with backfilling, was used for excavation the deposit "Trepča"– Stari Trg in the primary stage of mining. This method is used for excavation all ore bodies from the level of horizon from which the excavation starts up to 6-8 m below the level of upper horizon, i.e. up to the boundary of safety plate.

The method of square chocks in horizontal floors was used in the initial stage of mining for excavation the safety plates. The intensity of excavation in this method is extremely low, so that over time more and more gap was formed between the excavation in the primary and secondary stage of mining.

The Trepča deposit consists of a series of pipe ore bodies of irregular shape (Figure 1). The largest number of ore bodies is built of sulfphide minerals and smaller part of deposits consists of so called oligonite ore bodies, built of ferruginous – magmatic carbonates with higher or lower content of lead - zinc sulphides.

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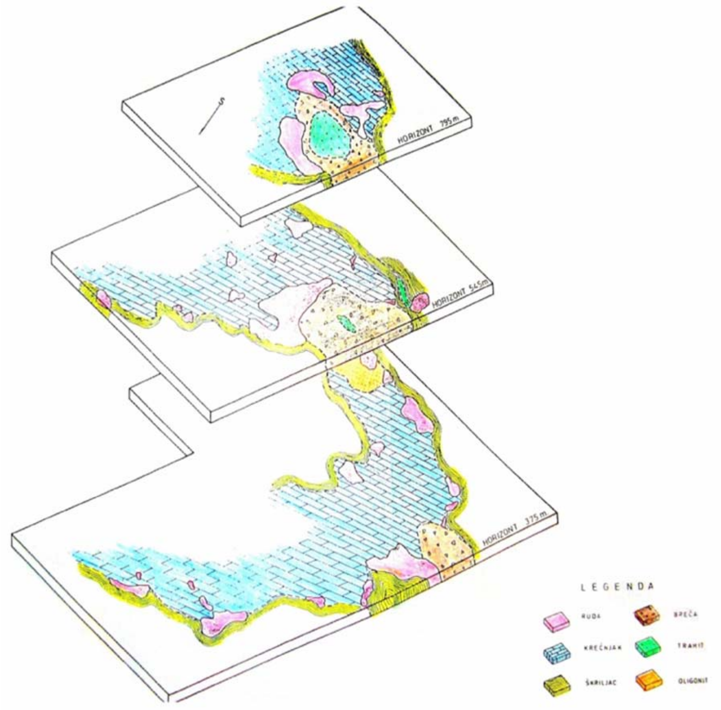

Fig. 1. *Block review of the Trepča deposit*

By laboratory testing of samples, sampled in the pit "Trepča"– Stari Trg, the following physical - mechanical characteristics of the ore and associated rocks, were found, which are shown in Table 1.

Table 1. *Physical – mechanical characteristics of ore and associated rocks*

ORE / ROCK	γ_1 [t/m^3]	γ_2 [t/m^3]	σ_c MPa	σ_i MPa	f	c MPa	ν
Sulphides	4.26	3.70	78.0	5.9	7.80	12.3	0.19
Oligonites	3.67	3.48	82.1	7.4	7.43	13.6	0.19
Limestone	2.86	2.80	49.5	5.0	5.27	8.8	0.17
Shale	2.83	2.76	44.1	6.6	4.45	8.6	0.17
Breccia	3.00	2.90	60.9	6.4	6.08	10.9	0.17

MINING OF DEPOSIT

The method of roof caving in horizontal floors with backfilling is used from the beginning deposit mining. It was used for excavation all ore bodies in the primary stage of exploitation (Figure 2).

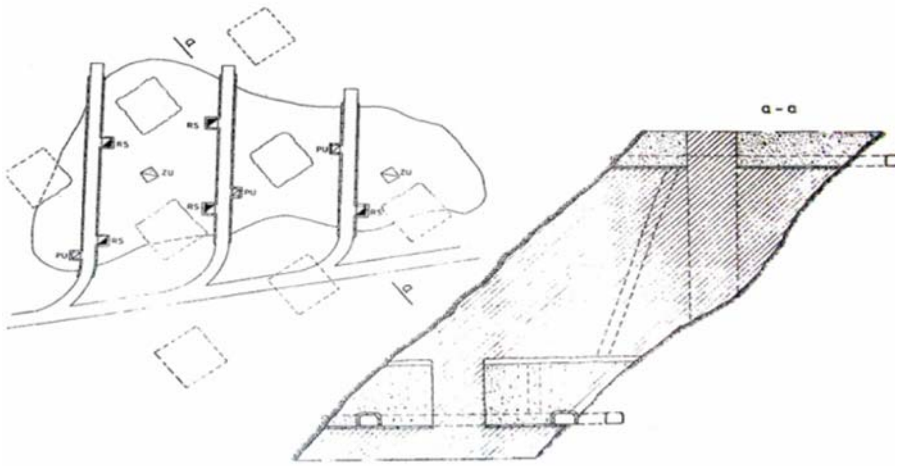

Fig. 2. *The Trepča mining method*

Preparatory works, i.e. the main haulage drift is made in the floor block, and the ore body is crosscut by intersected drifts whose number depends on the stope size. The ore body is crosscut by intersected drifts to the roof contact, and then the excavation of ore body is carried out near them along the entire surface.

Excavation of the first floor, height of 5.5 m, begins at the level of horizon. Drilling of blast holes is carried out by pneumatic drill RK – 28. Loading of ore is done directly into wagons by CAVO loading shovel. When the ore is excavated around the perimeter, the parts of drift, which were in the ore, are produced in the concrete lining, and also the ore chutes. The backfilling material is transported from the level of higher horizon and spread along the stope. An open height of 2 m for ventilation and communication in the stope is left under the roof of stope. Excavation of the next floor begins from the chute to the roof part of the stope.

Height difference between the horizons is 60 m. For the ore bodies in the Trepča mine of large areas from floor to the roof part, the safety pillars are left as temporary stope insurance in the primary stage of mining, arranged in a chess order, size 10 x 10 m. Distance between rows is from 12 – 16 m, and distance between pillars in a row of 16 – 20 m. Height of safety pillars is from 10 – 70 m, depending on the ore body thickness.

MINING OF SAFETY PLATES

The safety plates were mined for many years using the method of square chocks that has very low intensity of excavation and leads to growing delay of secondary stage to the primary stage of excavation.

The method is schematically shown in Figure 3, and the working principle consists in embedding the specially prepared square base into stope, immediately after excavation of a small space.

The first row of square support is placed on the foundation, and any subsequent is attached to the previous. The support has to be laid and connected properly, the pillars have to be placed vertically, and overheads and flights have to test against the pillars. Support tightening on the ceiling and sides on the stope is only done on joints.

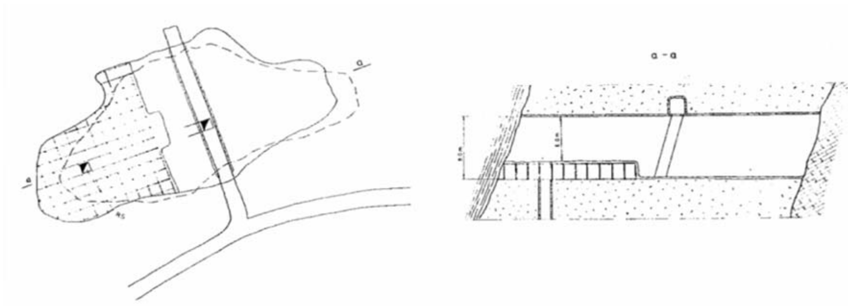

Fig. 3. *Method of square chocks*

All other working operations are the same as the method of roof mining with backfilling, but with only a reduced volume of works and effects. The method is characterized by very low effects in all phases of working process, and therefore ensuring low intensity of mining.

With the advancement of mining techniques and introduction of modern mechanization in recent years, it has tried to excavate the security plates using the method of cross wooden chocks. This is actually a modification of the horizontal roof mining with the use of systematic supporting with cross wooden chocks. The method proved significantly more effective in comparison with the method of square chocks, and also safe enough.

Taking into account the previous experience in the mining of safety plates, it can be concluded that this method is very effective and safe, and the intensity is not far behind the “Trepča" method, since it is possible to mechanize works in the excavation and backfilling of the excavated stope what was not a case with the method of square chocks.

In this case, the cross wooden chocks are used as support (Figure 4), size 2x2 m, and their height, at the time when the floor is excavated, is 2.5 m. To ensure the free movement of mechanization, the distance between the chocks in a row and between rows should be 2.5 to 3 m. If the said distance of 3.0 m is adopted and considered a case of the plate range of 18 m, then four crossed chocks can be placed in a row, on the area of 18 x 18 m, it is 4 x 4 or16 crossed chocks. Such arranged chocks are opposed to the roof, i.e. they have to prevent deformation of the plate.

Thus selected mining method of security plates, is essentially a continuation of the “Trepča” method" in excavation of the last 8 m, the ore plate between two horizons. Making the special preparatory works for this method is not required due to the use of facilities for primary stage of mining.

All stages of technological process of mining are carried out in the same order as in the normal operation; the only difference is in the supporting stage of stope that has to be done in the certain order and with special care. Namely, supporting has to follow successively the excavation works and maximum open area without permanent support not exceeding 25 m^2.

Excavation will be done in the floors, height 2 m, and it starts from an ore chute that is the nearest to the roof side. In the first stage, advance is made towards the roof side, and when it occurs, then with the certain precedence along the side for face formation for further advance.

Advance on a face is carried out by horizontal drilling and blasting in cuts, width 5 m. The advance of a cut at the face, looking in a depth, also can be up to 5 m, so that total open area does not exceed 25 m^2, after which the wooden crossed chock has to be installed.

Blowing down of ore is done with short blast holes, length 1.6 and diameter32 mm. Drilling is done by light pneumatic drill RK – 28.

Disposition of blast holes is shown in Figure 4 in the chess arrangement with distance 1.0 m between the blast holes and 0.5 m between rows. On the safety panel face, width 5 m and height 2 m, 4 rows or in total 18 blast holes will be drilled.

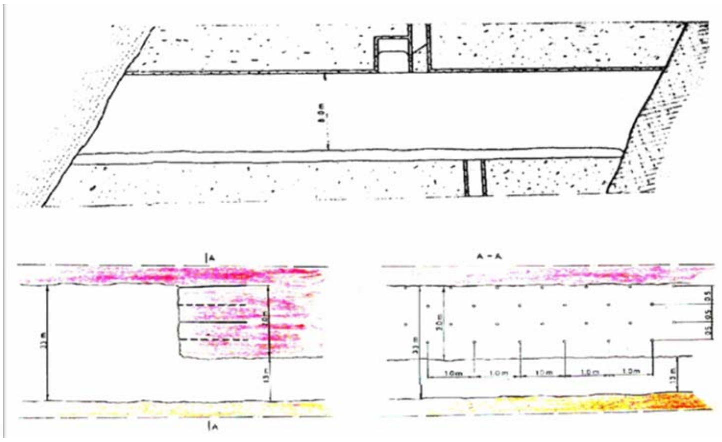

Fig. 4. *Review of excavation face with disposition of blast holes*

Blasted ore is loaded and transported to the chute of self-propelled loading - transport T_2GH machine, which has a bucket capacity of 0.12 m^3 and box volume of 0.75 m^3.

Systematic supporting with wooden crossed chocks means that after excavation of a section, maximum area of 25 m^2, a wooden crossed chock has to be installed, after which the excavation of the next section could start.

Chocks are placed in the center of each excavation sections, so that a network of crossed chocks is formed at mutual distance of 3 m, in a roe, and at the same distance between rows of chocks. The positioning way of each crossed chock is as follows: on a filling material of the previous floor, which in the place of the chock positioning has to be leveled, line up 9 dense placed beams. Then, on a such created base, 3 beams are placed criss-cross in a row all the way to the ceiling, as shown in (Figure 5).

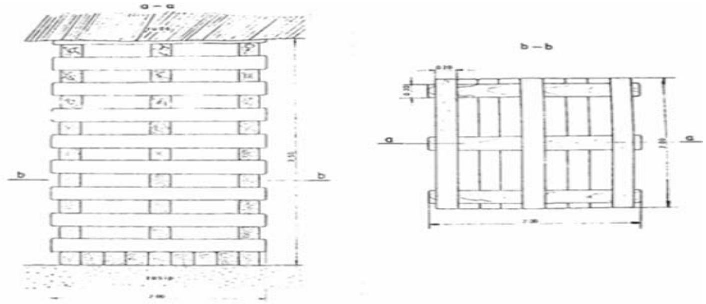

Fig. 5. *Supporting with wooden crossed chocks*

Tightening the placed crossed chock is made with wooden wedges which should be equally tamped on all sides, that a schocks would be equally tightened on the ceiling of stope. Chocks in a row and the rows of chocks are interconnected by stringers, length 3.5 m, and stringers are also tightened by wedges. Such placed and interconnected wooden crossed chocks present a solid unit which can stand up pressures from the ceiling of stope.

After completion of excavation works on entire floor, or a part of floor that can be a technological unit, and after raising the ore chutes, open raise and raise for filling, the stage of backfilling the stope starts.

Backfill material is, from higher horizon over filling raise, transported into stope and stretched with scrapers between the rows of wooden crossed chocks. The network of wooden crossed chocks, which secure the stope, remains in a filling serving as a basis for the new chock that will be placed in the excavation of the next floor. Excavation of the remaining security front of security panel is carried out by the same way as it was described here.

CONCLUSION

The method of square chocks was previously used for mining the safety plates in the mine "Trepča" – Stari Trg in the secondary stage of mining that had previosly very low intensity of excavation and there were growing imbalance of secondary to the primary stage of excavation.

This paper presents a choice of the modified mining method of horiyontal roof excavation using the systematic supporting with wooden chocks. The method has proved as significantly more effective in comparison with the method of square chocks, and also safe enough.

Taking into account the previous experience in the mining safety plates it can be concluded that this method is very effective and safe, and its intensity does not fall behind the "Trepča" method, since it is possible to mechanise all works both on excavation and backfilling of stope, what was not a case with the method of square chocks.

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POJAVA SUPROTNOSMERNOG KRETANJA TRAKASTOG TRANSPORTERA ZA RUDU T.109 I SPREČAVANJE OVE POJAVE

Izvod

U ovom radu je, na primeru trakastog transportera za rudu na površinskom kopu "Veliki Krivelj" sa kapacitetom 2000 t/h i sa usponom u smeru kretanja materijala, data analiza suprotnosmernog kretanja trakastog transportera u slučaju njegovog zaustavljanja.

Analiza je urađena računskim putem metodom "obilaska po konturi" i predstavlja univerzalni metod proračuna trakastih transportera čiji su rezultati neophodni za izbor uređaja za sprečavanje suprotnosmernog kretanja trakastog transportera.

Takođe su izloženi način rada i tehničke karakteristike usvojenog uređaja za sprečavanje suprotnosmernog kretanja.

Ključne reči: *trakasti transporter za rudu, suprotnosmerno kretanje, metod "obilaska po konturi", uređaj za sprečavanje suprotnosmernog kretanja*

1. UVOD

Suprotnosmerno kretanje trakastih transportera sa nagibom u smeru kretanja materijala predstavlja štetnu pojavu koja može da nastane u momentu zaustavljanja transportera kada pod dejstvom gravitacione sile na materijal koji se nalazi na traci transporter teži da se kreće u suprotnom smeru pri čemu može doći do rasipanja materijala sa trake i zasipanja utovarnog uređaja materijalom što uzrokuje zastoje i gubitke materijala.

Za transport primarno izdrobljene rude bakra na površinskom kopu "Veliki Krivelj" kod Bora od drobilice (poz. T.102.100.2) do otvorenog sklada za rudu projektovan je trakasti transporter (poz. T.109) [2]. Zahtev investitora u toku izrade projekta je bio da se proveri mogućnost iskorišćenja opreme postojećeg trakastog transportera koji je van funkcije na površinskom kopu u Majdanpeku. Za sprečavanje suprotnosmernog kretanja ovog trakastog transportera bio je predviđen ustavljač proizvođača Falk sa oznakom 1105NRT. U svrhu provere da li ovaj uređaj zadovoljava u novim radnim uslovima urađen je proračun suprotnosmernog kretanja trakastog transportera T.109 metodom "obilaska po konturi".

2. TEHNIČKI OPIS TRAKASTOG TRANSPORTERA T.109

Trasa trakastog transportera data je slici 1.

Tehničke karakteristike transportera:

1. Podaci o materijalu

* *Institut za rudarstvo i metalurgiju Bor*

1.1. Vrsta materijala: ruda bakra
1.2. Maksimalna dimenzija komada:
$a_{\max} = 250\,[mm]$
1.3. Nasipna gustina: $\rho = 1600\left[\frac{kg}{m^3}\right]$
1.4. Ugao prirodnog pada materijala u kretanju: $\varphi = 20\left[^o\right]$

2. Podaci o transporteru

2.1. Proizvođač: FOD Bor
2.2. Kapacitet transportera:
$Q_m = 2000\left[\frac{t}{h}\right]$
2.3. Dužina transportera:
$L = 466{,}076\,[m]$
2.4. Brzina transportera: $v = 3.8\left[\frac{m}{s}\right]$
2.5. Uslovi rada transportera: Rad van prostorije, vlažan vazduh

Trakasti transporter (vidi sliku 1) je sa gumenom trakom sa čeličnim užadima St 1250 širine 1200 mm koritastog poprečnog preseka sa tri noseća valjka pod nagibom od 36^o na radnoj grani i sa dva povratna valjka pod nagibom od 10^o na povratnoj grani.

Konstrukcija transportera se sastoji od utovarne stanice, standardnih sekcija i istovarno-pogonsko-zatezne stanice.

Utovarna stanica sastoji se od noseće konstrukcije, utovarnog levka, povratnog bubnja, nosećih amortizacionih valjaka i povratnih valjaka.

Standardnih sekcija ima dva tipa koji se razlikuju u dužinama tako da su duže sekcije (tip „A“) predviđene na ravnim delovima trase transportera a kraće (tip „B“) u krivinama.

Istovarno-pogonsko-zatezna stanica se sastoji od noseće konstrukcije („strele“) transportera, istovarnog bubnja, pogonskih bubnjeva, otklonskih bubnjeva i zateznog bubnja, pogona transportera, uređaja za zatezanje i nosećih i povratnih valjaka.

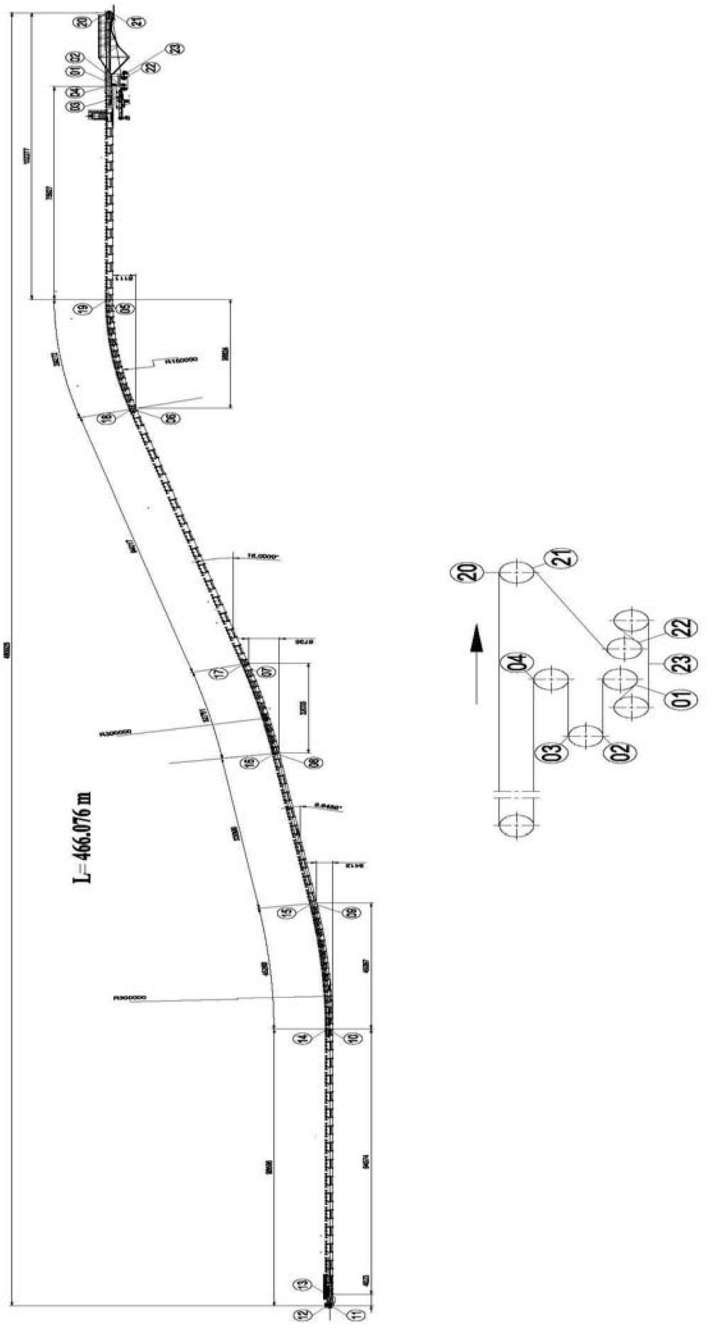
Pogon transportera je preko dva bubnja i sastoji se od dve identične pogonske grupe koje se sastoje od elektromotora, spojnice između elektromotora i reduktora, reduktora, spojnice između reduktora i pogonskog bubnja i samog pogonskog bubnja.

Zatezanje trake vrši se preko zateznog bubnja smeštenog u okviru istovarno-pogonsko-zatezne stanice koji se zateže tegom pomoću užeta i sistema užetnjača.

Istovar se vrši preko istovarnog bubnja-bubnja za odbacivanje koji se nalazi na gornjem kraju trakastog transportera.

Za sprečavanje suprotnosmernog kretanja trake transportera predviđena je kočnica koja je smeštena na istovarnom bubnju.

Za brisanje trake od naslaga materijala predviđena je stružna guma na istovarnom bubnju.



Sl. 1. *Trakasti transporter T.109*

3. PRORAČUN SUPROTNO - SMERNOG KRETANJA TRAKASTOG TRANSPORTERA T.109

1. Proračun ima za svrhu određivanje potrebnog kočionog momenta uređaja za sprečavanje suprotnosmernog kretanja trakastog transportera smeštenog na vratilu bubnja za odbacivanje preko koga se obavlja istovar materijala na kraju trakastog transportera.

Proračun se vrši metodom obilaska po konturi prema [1] za suprotnosmerno kretanje transportera. Karakteristične tačke u kojima se računaju sile u traci date su na slici 1. Proračun počinje od tačke 3 u kojoj traka pri suprotnosmernom kretanju nailazi na zatezni bubanj i u kojoj je sila uslovljena težinom zateznog tega. Proračun se odvija od tačke 3 od tačke 21 u smeru koji odgovara suprotnosmernom kretanju trake, a od tačke 4 od tačke 20 u suprotnom smeru. Razlika izračunatih sila u tačkama 20 i 21, koje predstavljaju tačke nailaska i silaska trake bubnja za odbacivanje na čije vratilo se montira kočioni uređaj, određuje veličinu kočionog momenta.

2. Sile opterećenja po dužnom metru transportera

2.1. Od transportovanog tereta

$$q_{ter} = \frac{Q_m \cdot g}{3,6 \cdot v}\left[\frac{N}{m}\right] = 1434,211\left[\frac{N}{m}\right]$$

2.2. Od težine obrtnih delova rolni radne grane

$$q_{r.rg.A} = \frac{m_{r.rg.A} \cdot g}{l_{r.rg.A}}\left[\frac{N}{m}\right] = 228,9\left[\frac{N}{m}\right]$$

$$q_{r.rg.B} = \frac{m_{r.rg.B} \cdot g}{l_{r.rg.B}}\left[\frac{N}{m}\right] = 343,35\left[\frac{N}{m}\right]$$

2.3. Od težine obrtnih delova rolni povratne grane

$$q_{r.pg.A} = \frac{m_{r.pg.A} \cdot g}{l_{r.pg.A}}\left[\frac{N}{m}\right] = 127,857\left[\frac{N}{m}\right]$$

$$q_{r.pg.B} = \frac{m_{r.pg.B} \cdot g}{l_{r.pg.B}}\left[\frac{N}{m}\right] = 191,785\left[\frac{N}{m}\right]$$

2.4. Od težine trake

$$q_{tr} = m_{tr} \cdot g\left[\frac{N}{m}\right] = 241,326\left[\frac{N}{m}\right]$$

gde su:

$m_{r.rg} = 28[kg]$ - masa obrtnih delova nosećih rolni

$l_{r.rgA} = 1,2[m]$ - rastojanje između nosećih rolni tipa „A“

$m_{r.pg} = 39,1[kg]$ - masa obrtnih delova povratnih rolni

$l_{r.pgA} = 3[m]$ - rastojanje između povratnih rolni tipa „A“

$l_{r.pgB} = 2[m]$ - rastojanje između povratnih rolni tipa „B“

$l_{r.rgB} = 0,8[m]$ - rastojanje između nosećih rolni tipa „B“

$m_t = 24,6[kg]$ - masa trake po dužini trake

3. Proračun sila u traci u karakterističnim tačkama transportera:

$$F_3 = \frac{F_z \cdot \eta_z}{2} = 62152[N]$$

gde su:

$F_z = 130847[N]$ – zatezna sila

$\eta_z = 0,95[-]$ – stepen korisnosti zateznog uređaja

$$F_2 = F_3 \cdot k_p = 65259{,}6\,[N]$$

$$F_1 = F_2 \cdot k_p = 68718{,}3\,[N]$$

$$F_{23} = F_1 \cdot k_p = 72360{,}4\,[N]$$

$$F_{22} = F_{23} \cdot k_p = 76195{,}5\,[N]$$

$$F_{21} = F_{22} \cdot k_p = 80233{,}9\,[N]$$

$$F_4 = \frac{F_3}{k_p} = 59.192{,}4\,[N]$$

$$F_5 = F_4 - W_{4-5} = 58071{,}2\,[N]$$

$$W_{4-5} = q_{tr} \cdot L_{4-5} \cdot w + q_{r.pg.A} \cdot L_{4-5} \cdot w = \\ = 1121{,}2\,[N]$$

$$F_6 = F_5 - W_{5-6} = 56161{,}7\,[N]$$

$$W_{5-6} = q_{tr} \cdot (L_{H5-6} \cdot w + L_{V5-6}) + \\ + q_{r.pg.B} \cdot L_{5-6} \cdot w = 1909{,}5\,[N]$$

$$F_7 = F_6 - W_{6-7} = 48932\,[N]$$

$$W_{6-7} = q_{tr} \cdot L_{6-7} (\cos\beta \cdot w + \sin\beta) + \\ + q_{r.pg.A} \cdot L_{6-7} \cdot w = 7229{,}9\,[N]$$

$$F_8 = F_7 - W_{7-8} = 46746{,}2\,[N]$$

$$W_{7-8} = q_{tr} \cdot (L_{H7-8} \cdot w + L_{V7-8}) + \\ + q_{r.pg.B} \cdot L_{7-8} \cdot w = 2185{,}8\,[N]$$

$$F_9 = F_8 - W_{8-9} = 44001{,}1\,[N]$$

$$W_{8-9} = q_{tr} \cdot L_{8-9} (\cos\beta \cdot w + \sin\beta) + \\ + q_{r.pg.A} \cdot L_{8-9} \cdot w = 2745{,}1\,[N]$$

$$F_{10} = F_9 - W_{9-10} = 42395{,}1\,[N]$$

$$W_{9-10} = q_{tr} \cdot (L_{H9-8} \cdot w + L_{V9-10}) + \\ + q_{r.pg.B} \cdot L_{9-10} \cdot w = 1606\,[N]$$

$$F_{11} = F_{10} - W_{10-11} = 40939{,}1\,[N]$$

$$W_{10-11} = q_{tr} \cdot L_{10-11} \cdot w + q_{r.pg.A} \cdot \\ \cdot L_{10-11} \cdot w = 1456\,[N]$$

$$F_{12} = \frac{F_{11}}{k_p} = 38989{,}6\,[N]$$

$$F_{13} = F_{12} + W_{ut} = 38989{,}6\,[N]$$

$$F_{14} = F_{13} - W_{13-14} = 37210{,}8\,[N]$$

$$W_{13-14} = q_{tr} \cdot L_{13-14} \cdot w + q_{r.pg.A} \cdot L_{13-14} \cdot \\ \cdot w = 1778{,}8\,[N]$$

$$F_{15} = F_{14} - W_{14-15} = 39283{,}5\,[N]$$

$$W_{14-15} = (q_{tr} + q_{ter}) \cdot \\ \cdot (L_{H14-15} \cdot w - L_{V14-15}) + \\ + q_{r.rg.B} \cdot L_{14-15} \cdot w = -2072{,}7\,[N]$$

$$F_{16} = F_{15} - W_{15-16} = 48793{,}5\,[N]$$

$$W_{15-16} = (q_{tr} + q_{ter}) \cdot \\ \cdot L_{15-16} \cdot (\cos\beta \cdot w - \sin\beta) + \\ + q_{r.rg.A} \cdot L_{15-16} \cdot w = -9510\,[N]$$

$$F_{17} = F_{16} - W_{16-17} = 57481{,}4\,[N]$$

$$W_{16-17} = (q_{tr} + q_{ter}) \cdot \\ \cdot (L_{H16-17} \cdot w - L_{V16-17}) + \\ + q_{r.rg.B} \cdot L_{16-17} \cdot w = -8687{,}9\,[N]$$

$$F_{18} = F_{17} - W_{17-18} = 91305{,}6\,[N]$$

$$W_{17-18} = (q_{tr} + q_{ter}) \cdot \\ \cdot L_{17-18} \cdot (\cos\beta \cdot w - \sin\beta) + \\ + q_{r.rg.A} \cdot L_{17-18} \cdot w = -33824{,}2\,[N]$$

$$F_{19} = F_{18} - W_{18-19} = 96727{,}9\,[N]$$

$$W_{18-19} = (q_{tr} + q_{ter}) \cdot \\ \cdot (L_{H18-19} \cdot w - L_{V18-19}) + \\ + q_{r.rg.B} \cdot L_{18-19} \cdot w = -5422{,}3\,[N]$$

$$F_{20} = F_{19} - W_{19-20} + W_{str} = 88348{,}7\,[N]$$

$$W_{19-20} = (q_{tr} + q_{ter}) \cdot \cdot L_{19-20} \cdot w + q_{r.rg.A} \cdot L_{19-20} \cdot w = = -7791{,}2[N]$$

gde su:

$k_p[-]$ - koeficijent povećanja sile zatezanja pri obavijanju doboša

$w = 0{,}04[-]$ – koeficijent otpora trak. transportera za olučasti profil trake i rad spolja

$L_{a-b}[m]$ – dužina karakteristične deonice

$\beta[^o]$ – ugao nagiba karakteristične deonice

$L_{Ha-b}[m]$ – dužina karakteristične deonice u horizontalnoj projekciji

$L_{Va-b}[m]$ – dužina karakteristične deonice u vertikalnoj projekciji

$W_{ut} = 0[N]$ – otpor na utovarnom delu trakastog transportera

$W_{str} = 490 \cdot B = 490 \cdot 12[N]$ - otpor stružne gume brisača

Potrebna sila kočenja na bubnju za odbacivanje izražena je jednačinom:

$$F_k = F_{20} - F_{21} = 8114{,}8[N]$$

Moment kočenja na sporohodom vratilu bubnja za odbacivanje dat je jednačinom:

$$M_k = \beta_k \cdot \frac{F_k \cdot D}{2} = 5629{,}6[Nm]$$

gde je:

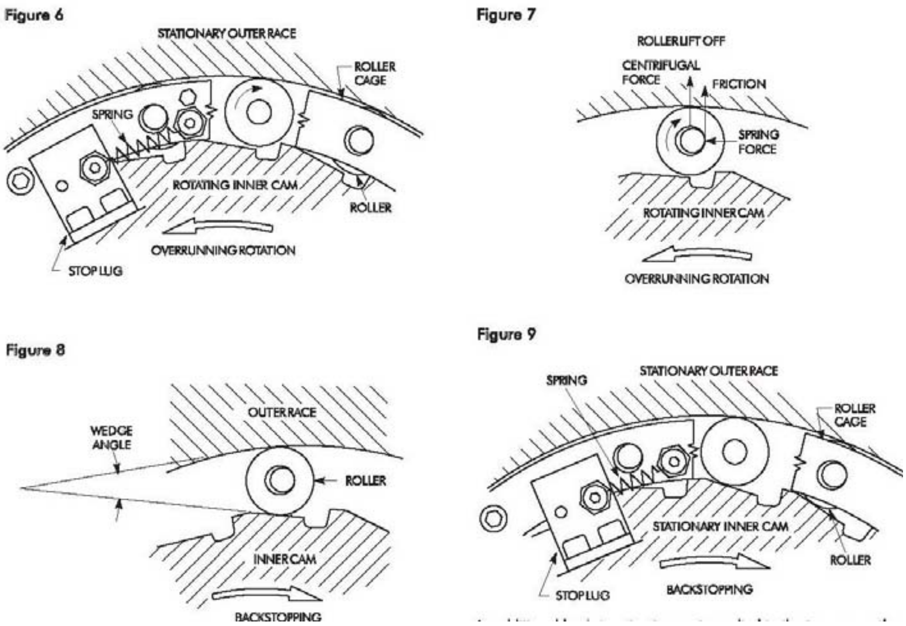
$\beta_k = 1{,}25$ – stepen sigurnosti kočenja

$D = 1{,}11[m]$ – spoljni prečnik bubnja za odbacivanje

4. DISKUSIJA PRORAČUNA

Proračunom dobijena pozitivna vrednost sile kočenja pokazuje da je moguće suprotnosmerno kretanje trakastog transportera pri njegovom zaustavljanju.

Da bi se sprečilo suprotnosmerno kretanje trakastog transportera predviđen je odgovarajući uređaj (vidi sl. 2) koji se sastoji od nepokretnog spoljnjeg prstena koji je preko poluge fiksiran za spoljnu konstrukciju i pokretne glavčine sa kosim ispustima između kojih se kotrljaju rolne smeštene u odgovarajućem kavezu vezanom preko opruga za obrtnu glavčinu. U toku kretanja trake u radnom smeru rolne se slobodno kotrljaju po spoljnjem prstenu dok pri suprotnosmernom kretanju dolazi do njihovog zaklinjavanja između odgovarajuće profilisane površine glavčine i spoljnjeg prstena. Obrtni moment kočenja se preko spoljnjeg prstena i poluge prenosi na spoljnu konstrukciju (vidi sl. 3).



Sl. 2. *Princip rada uređaja za sprečavanje suprotnosmernog kretanja trakastog transportera*

Sl. 3. *Uređaj za sprečavanje suprotnosmernog kretanja trakastog transportera ugrađen na vratilu bubnja za odbacivanje*

5. ZAKLJUČAK

Prema kataloškim podacima proizvođača Falk [3], uređaj za sprečavanje suprotnosmernog kretanja trakastog transportera sa oznakom 1105NRT ima maksimalni dozvoljeni kočioni moment od 60975 Nm koji je veći od potrebnog, što znači da postojeći uređaj za kočenje zadovoljava.

Iako sporohodi uređaj za zaustavljanje trakastog transportera pri suprotno smernom kretanju, obrađen u ovom radu, predstavlja samo jedan način rešavanja ovog problema, uz uređaje za kočenje sa skakavicom i brzohode uređaje za kočenje, on ima nesumnjive prednosti pre svega u jednostavnoj montaži, tihom i pouzdanom radu i dugom radnom veku.

6. LITERATURA

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[3] Katalog: Falk True Hold Low Speed Backstops

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Branislav Rajković, Zoran Ilić*, Radomir Mijović**

PHENOMENON OF REVERSAL MOTION OF BELT CONVEYOR FOR ORE T.109 AND PREVENTION OF THIS PHENOMENON

Abstract

This paperwork gives an analysis of reversal motion of the belt conveyor in the case of its stoppage in the example of a belt conveyor for ore at the open pit "Veliki Krivelj", with capacity of 2000 t/h with inclination in material transport direction.

The analysis was made by the calculation procedure using the "contour bypass" method, which is a universal calculation method for conveyor belt calculation, whose results are indispensable for the selection of backstopping device of belt conveyor.

Also, the principal of operation and technical characteristics the selected backstopping device are stated.

***Key words**: belt conveyor for ore, reversal motion, "contour bypass" method, backstopping device*

1. INTRODUCTION

Reversal motion of belt conveyors with inclination in material transportation direction is a harmful phenomenon which may occur at the moment of stoppage the belt conveyor when, under the influence of gravitational force on the material on belt, the belt conveyor tends to move in opposite direction whereby the effusion of material from the belt conveyor may occur as well as the backfilling of loading device by material, causing intermissions and material loss.

The belt conveyor was designed for transport of primary crushed copper ore at the open pit "Veliki Krivelj" in Bor from the crusher (pos. T.102.100.2) to the open storage for ore [3]. The request of investor during designing was to check the possibility of usage the existing equipment of a similar belt conveyor, which was out of operation at the Majdanpek open pit. The backstopping device, made by Falk with label 1105 NRT, was predicted in order to prevent the reversal motion of the belt conveyor. For the purpose of this device verification in the new operating conditions, the calculation of reversal motion of the belt conveyor T.109 was done using the "contour bypass" method.

2. TECHNICAL DESCRIPTION OF BELT CONVEYOR T.109

The route of the belt conveyor is shown on figure 1.

Technical characteristics of belt conveyor:

1. Material data

** Mining and Metallurgy Institute Bor*

1.1. Sort of material: copper ore

1.2. Maximum size of lumps: $a_{\max} = 250\,[mm]$

1.3. Bulk density: $\rho = 1600\left[\frac{kg}{m^3}\right]$

1.4. Surcharge angle in motion: $\varphi = 20\left[^o\right]$

2. Belt conveyor data

2.1. Manufacturer: FOD Bor

2.2. Capacity: $Q_m = 2000\left[\frac{t}{h}\right]$

2.3. Length: $L = 466.076\,[m]$

2.4. Speed: $v = 3.8\left[\frac{m}{s}\right]$

2.5. Operating condition: Operation outside the room, humid air

Belt conveyor (Figure 1) has a rubber belt with steel cords St 1250, width of 1200 mm, and trough cross section made of three carrying rolls with trough idler angle of 36º on carry side and two return rolls with trough idler angle of 10º on return side.

Construction of belt conveyor includes a loading station, standard sections and discharge-drive-take up station.

Loading station consists of a bearing construction, charging chute, tail pulley, carrying amortizing rolls and return rolls.

There are two types of standard sections which differ in length, so that longer sections (type „A“) are predicted at straight segments of the belt conveyor route, while shorter sections (type „B“) are predicted at bends.

Discharge-drive-take up station consists of a bearing construction („arrow“), discharge pulley, drive pulleys, snap pulleys and take up pulley, conveyor drive unit, take up device and carrying and return rolls.

Conveyor driving is obtained by means of two pulleys, and it consists of two identical driving units, each consisting of an electric motor, a coupling between the motor and gear unit, the gear unit, coupling between the gear unit and drive pulley and a drive pulley itself.

Belt tension is obtained taking up pulley placed within the discharge-drive-take up station, tightened by counter weight using a rope and system of pulley blocks.

Discharge is done by discharge pulley-unloading pulley, placed at the upper end of belt conveyor.

To prevent the reversal motion of belt conveyor, the backstopping device is predicted, placed on a discharge pulley.

A scraping rubber, mounted on a discharge pulley, is predicted for cleaning the material layer off the belt.

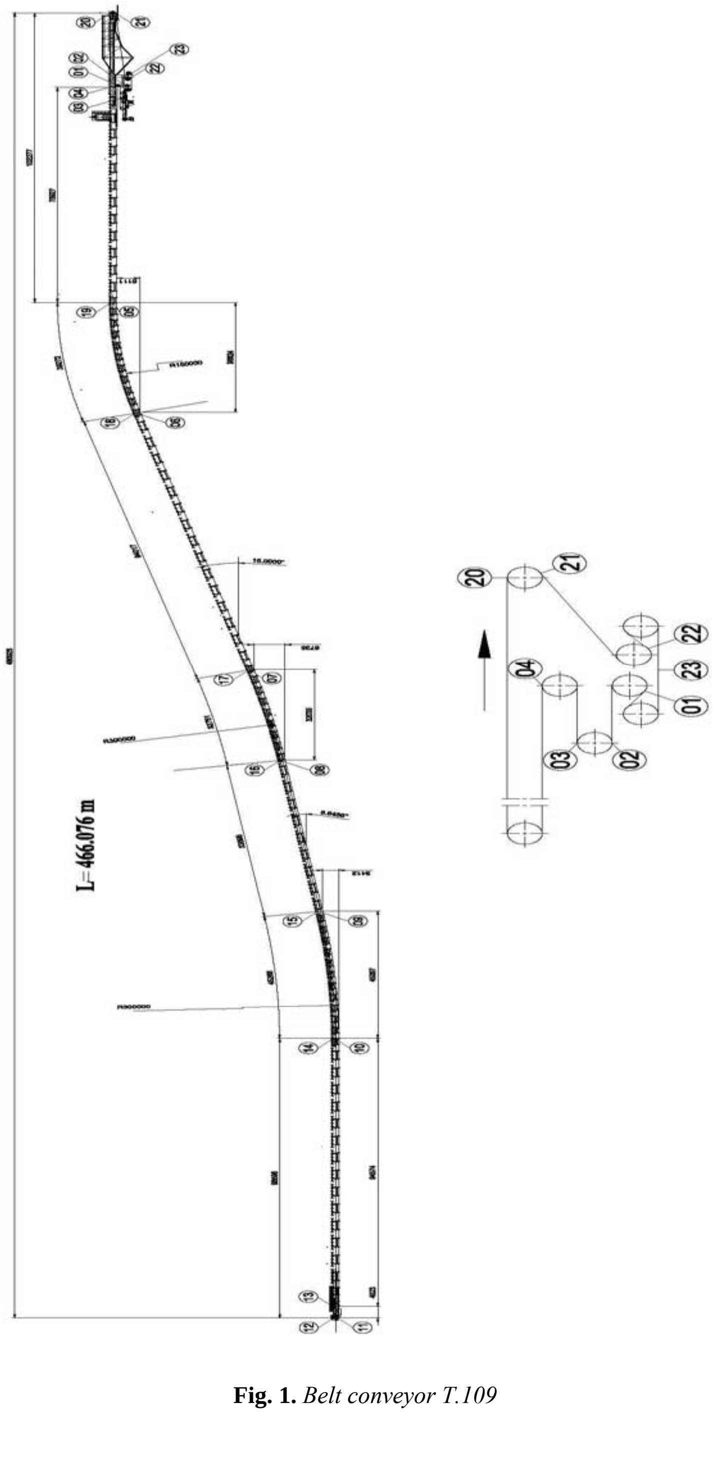

Fig. 1. *Belt conveyor T.109*

3 .CALCULATION OF REVERSAL MOTION OF THE BELT CONVEYOR T.109

1. Calculation was aimed to determine the brake torque required for belt conveyor backstopping device, mounted on the shaft of discharge pulley serving for material unloading at the end of belt conveyor.

Calculation is done using the contour bypass method towards [1] the reversal motion of belt conveyor. Characteristic points, in which the tension forces in belt are calculated, are given in Figure 1. Calculation starts at point 3 at which the belt, during the reversal motion, ascends take up pulley and the tension force is determined by the weight of counter weight. The calculation is done from point 3 to point 21 in the direction corresponding to the reversal motion of the belt, whilst the calculation from point 4 to point 20 is done in opposite direction. The difference between calculated tension forces at points 20 and 21, representing ascending and descending points of the belt at discharge pulley on whose shaft the backstopping device is mounted, determines the value of the brake torque.

2. Loads per meter of the belt length

2.1. From transported material

$$q_{ter} = \frac{Q_m \cdot g}{3.6 \cdot v}\left[\frac{N}{m}\right] = 1434.211\left[\frac{N}{m}\right]$$

2.2. From the weight of rotary parts of carry side rolls

$$q_{r.rg.A} = \frac{m_{r.rg.A} \cdot g}{l_{r.rg.A}}\left[\frac{N}{m}\right] = 228.9\left[\frac{N}{m}\right]$$

$$q_{r.rg.B} = \frac{m_{r.rg.B} \cdot g}{l_{r.rg.B}}\left[\frac{N}{m}\right] = 34335\left[\frac{N}{m}\right]$$

2.3. From the weight of rotary parts of return side rolls

$$q_{r.pg.A} = \frac{m_{r.pg.A} \cdot g}{l_{r.pg.A}}\left[\frac{N}{m}\right] = 127.857\left[\frac{N}{m}\right]$$

$$q_{r.pg.B} = \frac{m_{r.pg.B} \cdot g}{l_{r.pg.B}}\left[\frac{N}{m}\right] = 191.785\left[\frac{N}{m}\right]$$

2.4. From the belt weight

$$q_{tr} = m_{tr} \cdot g\left[\frac{N}{m}\right] = 241.326\left[\frac{N}{m}\right]$$

where:

$m_{r.rg} = 28[kg]$ –mass of rotary parts of carrying rolls

$l_{r.rgA} = 1.2[m]$ –distance between carrying rolls type „A“

$m_{r.pg} = 39.1[kg]$ -mass of rotary parts of return rolls

$l_{r.pgA} = 3[m]$ -distance between return rolls type „A“

$l_{r.pgB} = 2[m]$ -distance between return rolls type „B“

$l_{r.rgB} = 0.8[m]$ -distance between carrying rolls type „B“

$m_t = 24.6[kg]$ –belt mass per meter

3. Calculation of tension forces at characteristic points of belt conveyor:

$$F_3 = \frac{F_z \cdot \eta_z}{2} = 62152[N]$$

where i:

$F_z = 130847[N]$ –tension force

$\eta_z = 0.95[-]$ –efficiency of take up device

$$F_2 = F_3 \cdot k_p = 65259.6[N]$$

$$F_1 = F_2 \cdot k_p = 68718.3[N]$$

$$F_{23} = F_1 \cdot k_p = 72360.4[N]$$

$$F_{22} = F_{23} \cdot k_p = 76195.5[N]$$

$$F_{21} = F_{22} \cdot k_p = 80233.9[N]$$

$$F_4 = \frac{F_3}{k_p} = 59.192.4[N]$$

$$F_5 = F_4 - W_{4-5} = 58071.2[N]$$

$$W_{4-5} = q_{tr} \cdot L_{4-5} \cdot w + q_{r.pg.A} \cdot L_{4-5} \cdot w = \\ = 1121.2[N]$$

$$F_6 = F_5 - W_{5-6} = 56161.7[N]$$

$$W_{5-6} = q_{tr} \cdot (L_{H5-6} \cdot w + L_{V5-6}) + \\ + q_{r.pg.B} \cdot L_{5-6} \cdot w = 1909.5[N]$$

$$F_7 = F_6 - W_{6-7} = 48932[N]$$

$$W_{6-7} = q_{tr} \cdot L_{6-7}(\cos\beta \cdot w + \sin\beta) + \\ + q_{r.pg.A} \cdot L_{6-7} \cdot w = 7229.9[N]$$

$$F_8 = F_7 - W_{7-8} = 46746.2[N]$$

$$W_{7-8} = q_{tr} \cdot (L_{H7-8} \cdot w + L_{V7-8}) + \\ + q_{r.pg.B} \cdot L_{7-8} \cdot w = 2185.8[N]$$

$$F_9 = F_8 - W_{8-9} = 44001.1[N]$$

$$W_{8-9} = q_{tr} \cdot L_{8-9}(\cos\beta \cdot w + \sin\beta) + \\ + q_{r.pg.A} \cdot L_{8-9} \cdot w = 2745.1[N]$$

$$F_{10} = F_9 - W_{9-10} = 42395.1[N]$$

$$W_{9-10} = q_{tr} \cdot (L_{H9-8} \cdot w + L_{V9-10}) + \\ + q_{r.pg.B} \cdot L_{9-10} \cdot w = 1606[N]$$

$$F_{11} = F_{10} - W_{10-11} = 40939.1[N]$$

$$W_{10-11} = q_{tr} \cdot L_{10-11} \cdot w + q_{r.pg.A} \cdot \\ \cdot L_{10-11} \cdot w = 1456[N]$$

$$F_{12} = \frac{F_{11}}{k_p} = 38989.6\,[N]$$

$$F_{13} = F_{12} + W_{ut} = 38989.6\,[N]$$

$$F_{14} = F_{13} - W_{13-14} = 37210.8\,[N]$$

$$W_{13-14} = q_{tr} \cdot L_{13-14} \cdot w + q_{r.pg.A} \cdot L_{13-14} \cdot \\ \cdot w = 1778.8\,[N]$$

$$F_{15} = F_{14} - W_{14-15} = 39283.5\,[N]$$

$$W_{14-15} = (q_{tr} + q_{ter}) \cdot \\ \cdot (L_{H14-15} \cdot w - L_{V14-15}) + \\ + q_{r.rg.B} \cdot L_{14-15} \cdot w = -2072.7[N]$$

$$F_{16} = F_{15} - W_{15-16} = 48793{,}5\,[N]$$

$$W_{15-16} = (q_{tr} + q_{ter}) \cdot \\ \cdot L_{15-16} \cdot (\cos\beta \cdot w - \sin\beta) + \\ + q_{r.rg.A} \cdot L_{15-16} \cdot w = -9510[N]$$

$$F_{17} = F_{16} - W_{16-17} = 57481.4\,[N]$$

$$W_{16-17} = (q_{tr} + q_{ter}) \cdot \\ \cdot (L_{H16-17} \cdot w - L_{V16-17}) + \\ + q_{r.rg.B} \cdot L_{16-17} \cdot w = -8687.9[N]$$

$$F_{18} = F_{17} - W_{17-18} = 91305.6\,[N]$$

$$W_{17-18} = (q_{tr} + q_{ter}) \cdot \\ \cdot L_{17-18} \cdot (\cos\beta \cdot w - \sin\beta) + \\ + q_{r.rg.A} \cdot L_{17-18} \cdot w = -33824.2[N]$$

$$F_{19} = F_{18} - W_{18-19} = 96727.9\,[N]$$

$$W_{18-19} = (q_{tr} + q_{ter}) \cdot \\ \cdot (L_{H18-19} \cdot w - L_{V18-19}) + \\ + q_{r.rg.B} \cdot L_{18-19} \cdot w = -5422.3[N]$$

$$F_{20} = F_{19} - W_{19-20} + W_{str} = 88348.7\,[N]$$

$$W_{19-20} = (q_{tr} + q_{ter})\cdot$$
$$\cdot L_{19-20} \cdot w + q_{r.rg.A} \cdot L_{19-20} \cdot w =$$
$$= -7791.2[N]$$

where:

$k_p\,[-]$ - coefficient of tension force increase due to belt winding around pulley

$w = 0{,}04[-]$ – resistance coefficient of belt conveyor for trough belt profile and outside operation

$L_{a-b}\,[m]$ – length of specific section

$\beta\left[^{o}\right]$ – inclination angle of specific section

$L_{Ha-b}\,[m]$ – length of specific section in horizontal projection

$L_{Va-b}\,[m]$ – length of specific section in vertical projection

$W_{ut} = 0[N]$ – resistance on the loading part of belt conveyor

$W_{str} = 490 \cdot B = 490 \cdot 12[N]$ – resistance of scraping rubber of the cleaner

The required brake force, at discharge pulley, is expressed by equation:

$$F_k = F_{20} - F_{21} = 8114.8[N]$$

Brake torque on low speed shaft of discharge pulley is given by equation:

$$M_k = \beta_k \cdot \frac{F_k \cdot D}{2} = 5629.6[Nm]$$

where:

$\beta_k = 1.25$ – braking efficiency

$D = 1.11[m]$ – outer diameter of discharge pulley

4. DISCUSSION OF CALCULATION

The obtained positive value of brake force by calculation shows that the reversal motion of belt conveyor is possible at its stoppage.

To prevent reversal motion of belt conveyor, a suitable device is predicted (Figure 2) which consists of a stationary outer race, fixed to the outer construction with a lever, and a rotating inner cam with ramps, between which rollers turn placed in a suitable cage attached via springs to the rotating inner cam. During the belt motion in working direction, the rollers turn freely over the outer race while they are wedged between adequately profiled surface of cam and outer race at reversal motion. The brake torque is transferred via the outer race and lever to the outer construction (Figure 3).

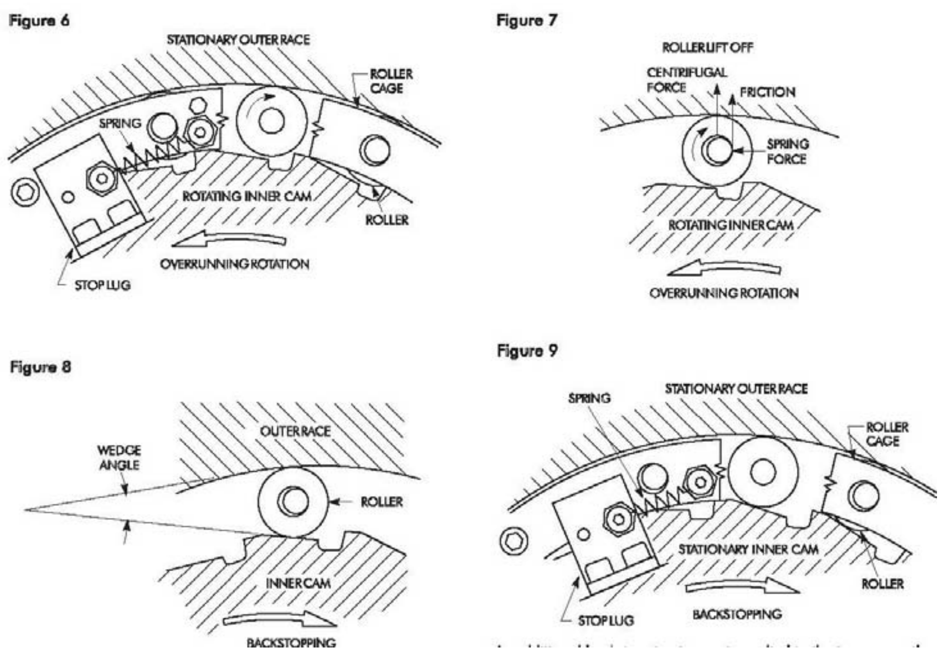

Fig. 2. *The operation principle of the belt conveyor backstopping device*

Fig. 3. *The assembled backstopping device of belt conveyor on discharge pulley shaft*

5. CONCLUSION

According to the manufacturer Falk catalogue data [3], the backstopping device with label 1105NRT has maximum allowable brake torque of 60975 Nm, which is higher than the needed one, meaning that the existing backstopping device satisfies.

Although the low speed backstop device, explained in this paperwork, represents only one way of solving this problem, it has the undoubted advantages along with backstop devices with pawl and high speed backstops, first of all in simple assembling, quite and reliable operation and long operating life.

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Bojan Drobnjaković, Dragan Milanović*, Vesna Drobnjaković*

IZBOR POGONSKE GRUPE TRAKASTOG DODAVAČA DROBILIČNOG POSTROJENJA ZA RUDU/JALOVINU NA POVRŠINSKOM KOPU RUDNIKA VELIKI KRIVELJ**

Izvod

Izgradnjom primarnog drobiličnog postrojenja br.2 u okviru površinskog kopa rudnika Veliki Krivelj postiže se veći kapacitet prerade rude bakra u cilju povećanja kapaciteta rudnika bakra Veliki Krivelj sa 8,0 na 10,5x10^6 t/god rude. [1,2] Veoma važna i skupa tehnološka pozicija u okviru postrojenja je trakasti dodavač ispod primarne drobilice tipa kružno-konusna Allis Chalmers 48"x74" kojim se doprema ruda dalje na transportni sistem za rudu ili na transportni sistem za jalovinu, ukoliko se vrši drobljenje jalovine. Sa povećanjem kapaciteta prerade rude/jalovine rudnika bakra Veliki Krivelj povećava se i količina raskrivke/rude istoimenog površinskog kopa a posledično i količina produkovane topioničke šljake iz procesa topljenja koncentrata bakra dobijenog iz flotacija Veliki Krivelj, Bor i Majdanpek.

Ključne reči: *transportni sistem, ruda/jalovina, trakastidodavač, pogonska grupa*

UVOD

Prema usvojenim strateškim planovima razvoja RTB - Bor potrebno je bilo izvršiti izgradnju drobiličnog postrojenja br. 2 u okviru transportnog sistema za jalovinu od površinskog kopa rudnika Veliki Krivelj do odkopanog prostora površinskog kopa Bor. Dodatnim sagledavanjima došlo se od ideje da se ovo drobilično postrojenje iskoristi i za drobljenje rude a, ne samo za drobljenje jalovine kao što je slučaj sa postojećim drobiličnim postrojenjem br. 1. Sastavni delovi drobiličnog postrojenja su: prijemni bunker za rudu/jalovinu, člankasti dodavač, prihvatni bunker drobilice, kružno-konusna primarna drobilica, izlazni levak drobilice, trakasti dodavač ispod drobilice, pretovarna sipka, reverzibilni dodavač, izlazna sipka za rudu i izlazna sipka za jalovinu.

Sastavni deo drobiličnog postrojenja je trakasti transporter - dodavač, koji se nalazi

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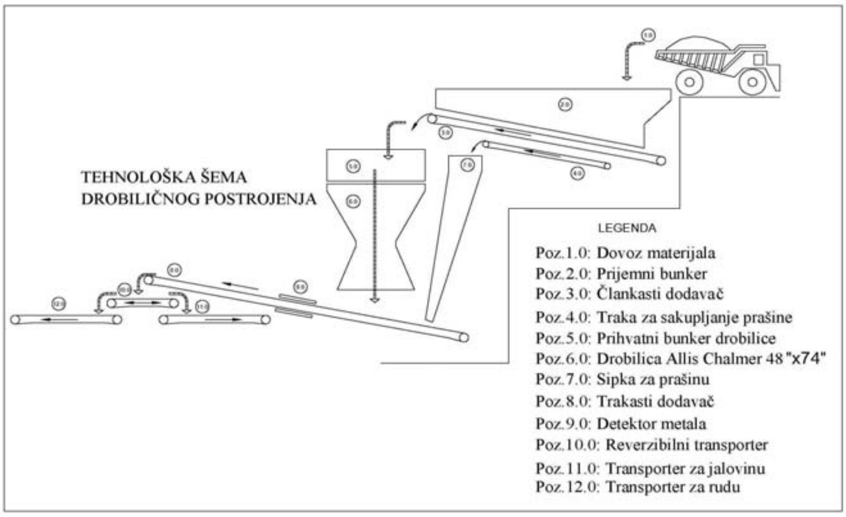
ispod drobilice i doprema material na reverzibilni transportet i dalje na transporter za rudu, odnosno transporter za jalovinu. Postavljen je pod uglom od 5,50.

Investitor je doneo odluku da za ovu namenu iskoristi postojeći trakasti dodavač iz rudnika Majdanpek, sa tzv. faze IV koji više godina nije u radu. Na slici br.1 data je tehnološka šema sa pozicijama sastavnih delova instalisanog drobiličnog postrojenja br. 2, dok je na slici br. 2 prikazano izgrađeno drobilično postrojenje br. 2, a na slikama br. 3 i br. 4 su prikazane faze ugradnje pogonske grupe trakastog dodavača.



Sl. 1. *Tehnološka šema instalisanog drobiličnog postrojenja br.2.*

Sl. 2. *Vizuelni prikaz instalisanog drobiličnog postrojenja br.2.*

Sl. 3 *Instalacija pogonske grupe trakastog dodavača poz. T-103.*

Sl. 4. *Prikaz pogonske grupe. Elektromotor reduktor spojnica.*

PRORAČUN POGONSKE GRUPE TRANSPORTERA T-103

Potrebno je sprovesti analizu instalisanog kapaciteta transportera na poz. T-103, samim tim i proveru pogonske snage elektromotora tog transportera s obzirom da se visina dizanja transportovanog materijala povećava za 1,5 m, što nije zanemarljivo. [3,4]

S obzirom na to, očekivanja su da pogonska snaga elektromotora bude na samoj granici zadovoljavanja pogonskih uslova. Količina transportovanog materijala je ista, kako na poziciji u rudniku Majdanpek tako i na ovoj poziciji u rudniku Veliki Krivelj.

Proračun ovog trakastog transportera može da se vrši na osnovu standarda SRPS M.D2.050, s obzirom da je transporter male dužine.

Proračun kapaciteta stacionarnog transportera podrazumeva verifikaciju kapaciteta trakastog transportera – dodavača ispod primarne drobilice, tehnološke pozicije T-103, u dužini od 25 m i visine dizanja 4,15 m pri brzini gumene transportne trake od 2,43 m/s.

Izbor širine trake

Širina trake je određena ne kapacitetom već izlaznim otvorom drobilice. Širina gumene trake je 2200 mm, i ona zadovoljova u svakom pogledu.

Provera preseka nasipa trake prema količini transportovanog materijala

Presek nasipa trake se izračunava prema obrascu:

$$A = \frac{1}{k_1 \cdot k_2} \cdot \frac{Q}{3600 \cdot \rho \cdot v}$$

gde je:

A - površina preseka nasipa, u m^2

Q – količina (masa) transportovanog materijala, u t/h,

ρ - zapreminska masa (gustina) transportovanog materijala u kg/m^3,

v – brzina trake, m/s,

k_1–koeficijent smanjenja teorijskog kapaciteta, zbog neravnomernog i nepotpunog nasipanja,

k_2–koeficijent smanjenja teorijskog kapaciteta, zbog nagiba transportera.

Tabela. 1

Parametar	T 103 B
Q	2350 t/h
ρ	1600 kg/m^3
k_1	0,95
k_2	0,95
v	2,1 m/s,
A	0,215 m^2

- Aktivna širina trake:

$$b_1 = \sqrt{\frac{A \cdot 3600}{f}} = 1{,}2 \text{ m}$$

gde je:

b_1 – aktivna širina trake , u m

f – factor oblika preseka nasipa. Za koritasti oblik, sa bočnim uglovima od 35°, faktor ima vrednost, f = 550.

Aktivna širina trake je 1,5m. tj. širina izla-znog otvora usipnog levka ispod drobilice je 1,5 m.

-Potrebna obodna sila na obodu pogonskog bubnja izračunava se iz izraza:

$$F_{bo} = C \cdot t \cdot L \cdot g \cdot (G_t + \frac{Q_h}{3{,}6 \cdot V}) + \\ + \frac{Q_h \cdot H}{3{,}6 \cdot V} daN$$

C – koeficijent koji uzima u obzir povećanje vučne sile zbog sporednih otpora duž transportera. Zavisi od dužine transportera.

Za ovaj slučaj uzeće se vrednost 1,05.

L(m) – dužina transportera.

t - koeficijent trenja u ležajevima (bubnjeva i valjaka). Za uslove povećane zaprašenosti uzima se vrednost 0,015 za rolne sa kugličnim ležajevima.

g (m/s^2) – ubrzanje zemljine teže.

G_t (kg/m) – masa pokretnih delova transportera po 1 m dužine transportera.

Q_h(t/h) – masa transportovanog mateijala u t/h.

v (m/s) – brzina gumene transportne trake.

H(m) – visina dizanja ili spuštanja tereta.

Tabela 2. *Vrednosti parametara*

Parametar	
C (/)	1,05
L (m)	25
T (/)	0,015
G_t(kg/m)	137,303
Q_h (t/h)	2350
H (m)	4,15
v (m/s)	2,47
b (m)	2200
F_{bo}	3661daN

- Snaga na vratilu pogonskog bubnja se računa iz izraza:

$$P_{bo} = \frac{F_{bo} \cdot v}{102},$$

$$P_{bo} = 75 \text{ kW}$$

- Dodatna potrebna snaga usled čistača trake se računa iz izraza:

$$P_d = 1.6 \cdot b \cdot v \cdot n$$

gde je:

n - broj skidača

$$P_d = 1,6 \cdot 2,2 \cdot 2,47 \cdot 2 = 17\,kW$$

-Dodatna potrebna snaga usled bočnih vođica:

$P_v = 0,08 \cdot 10\,m = 0,8\ kW$, gde je:

- Efektivna snaga na vratilu pogonskog bubnja se računa iz izraza:

$$P_e = P_{bo} + P_d + P_v,$$

$$P_e = 75 + 17 + 0,8 = 92,8 \text{ kW}$$

- Snaga pogonskog elektromotora se računa iz izraza:

$$P_{el} = \frac{P_e}{\eta} = \frac{P_e}{0,89 \cdot 0,95}$$

gde je:

η-stepen stepen korisnosti dejstva

$$P_{el.} = 109,75 \text{ kW}$$

Usvajaja se električna snaga elektromotora od 110 kW.

ZAKLJUČAK

Ovom analizom se dokazuje da trakasti dodavač sa postojećim elektromotorom snage 110 kW, kao i postojećom gumenom trakom može da ispuni zahtevani kapacitet od 2350 t/h. Interesantno u ovoj analizi je to, što sračunata snaga odgovara usvojenoj standardnoj snazi elektromotora, što je bitno za projektante u donošenju odluke oko izbora snage sletromotora. Prema ovoj analizi investotor je preuzeo sve potrebne aktivnosti da izvrši preseljenje postojećeg transportera sa lokacije rudnika Majdanpek i da uz dodatne modifikacije i repaparacije izvrši njegovu ugradnju na novoj lokaciji drobiličnog postrojenja br. 2 na površinskom kopu rudnika Veliki Krivelj. Nakon godine dana rada izgrađenog postrojenja, pokazalo se da je ova računska potvrda bila ispravna.

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SELECTION OF A BELT FEEDER DRIVE GROUP OF CRUSHING PLANT FOR ORE/WASTE AT THE OPEN PIT OF VELIKI KRIVELJ MINE **

Abstract

Construction of the primary crushing plant No. 2 within the open pit of the Veliki Krivelj mine has resulted in the increase copper ore processing capacity in order to increase the capacity of copper mine Veliki Krivelj from 8.0 to 10.5 x 10^6 t/year of ore [1,2]. An important and expensive technological position, within the facility, is the belt feeder below the primary crusher, type circular-conical Allis Chalmers 48" x 74", which delivers ore further on a transport system for ore or transport system for waste, if the waste is crushed. With the capacity increase of ore/waste from the copper mine Veliki Krivelj, the quantity of overburden/ore is also increased of the same named open pit, and consequently the quantity of produced smelter slag from the process of copper concentrate smelting, obtained from the flotation plants Veliki Krivelj, Bor and Majdanpek.

***Key words**: transport system, ore/waste, belt feeder, drive group*

1. INTRODUCTION

According to the adopted strategic plans for the development of RTB Bor, it is necessary to carry out the construction of crushing plant Plant No. 2 within the transport system for waste from the open pit of Veliki Krivelj mine to the excavated area of the open pit Bor. Futher considerations have resulted into an idea to use also this crushing plant for ore crushing, not only for waste crushing as it is the case with the existing crushing plant No. 1. The components of crushing plant are: receiving bin for ore/waste, products, sectional feeder, reception bin of crusher, circular conical primary crusher, output hopper of crusher, belt feeder under crusher, reloading chute, reversable feeder, output chute for ore anad output chute for waste. An integral part of crushing plant is a belt conveyor - feeder, which is located below crusher, and delivers the material on a reversible transporter for ore or transporter for waste. It is installed at angle of 5.5^o.

* *Mining and Metallurgy Institute Bor*

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The investor has decided to use for this purpose the existing belt feeder from the Majdanpek mine, from so-called phase IV, which is out of operation for several years. Figure 1 gives a technological layout with component positions of installed crushing plant No. 2, while Figure 2 presents the constructed crushing plant No. 2, and Figures 3 and 4 present the the stages of installation the belt feeder drive group.

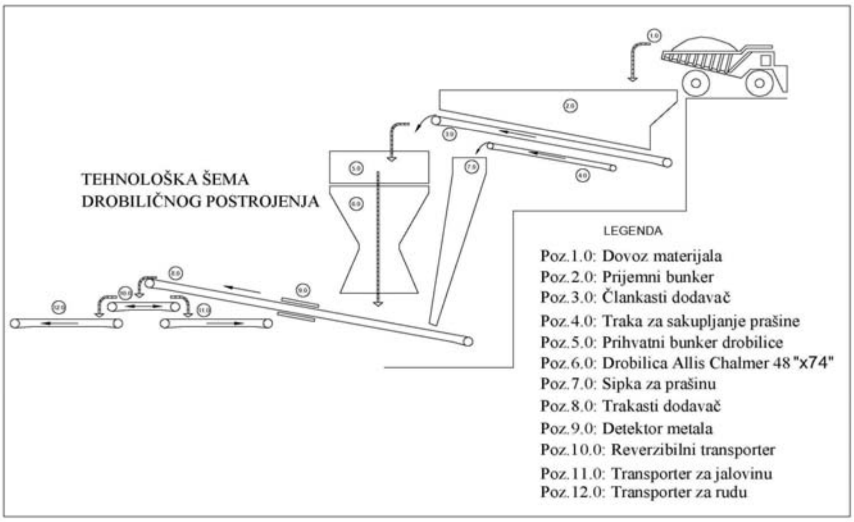

Fig. .1. *Technological layout of installed crushing plant No. 2*

Fig. 2. *Visual review of installed crushing plant No. 2*

Fig. 3. *Installation of belt feeder drive group position T-103.*

Fig. 4. *Review of drive group - electric motor reducer of coupling*

CALCULATION OF A CONVEYOR DRIVE GROUP T-103

It is necessary to carry out an analysis of installed capacity of conveyorat position T-103, thus checking the driving force of electric motor considering that a lifting height of transported material increases by 1.5 m, which is not negligible [3,4].

Whereas, it is expected that drive power of electric motor is in a limit of meeting the operational requirements. The amount of transported material is the same both at the position in the Majdanpek mine and at thisposition in the Veliki Krivelj mine.

Calculation of this belt conveyor can be made on the basis of the standard SRPS M.D2.050, due to a short length of conveyor.

Capacity calculation of the stationary belt conveyor means the verification of belt conveyor –feeder capacity under the primary crusher, technological positions of T-103, in a length of 25 m and lifting height of 4.15 m at a speed of rubber conveyor belt of 2.43 m/s.

Selection of belt width

Belt width is not determined by the capacity but by the crusher outlet. Width of rubber belt is 2200mm, and it satisfies in every way.

Checking a section of belt filling according to the quantity of transported material

A section of belt filling is calculated by the formula:

$$A = \frac{1}{k_1 \cdot k_2} \cdot \frac{Q}{3600 \cdot \rho \cdot v}$$

where:

A – section area of filling, in m^2,

Q – quantity (mass) of transported material, in t/h,

ρ - zapreminska masa (gustina) transportovanog materijala u kg/m^3,

v – belt speed, m/s,

k_1–coefficient of theoretical capacity reduction due to uneven and incomplete filling,

k_2– coefficient of theoretical capacity reduction due to conveyor slope,

Table 1.

Parameter	T 103 B
Q	2350 t/h
ρ	1600 kg/m^3
k_1	0.95
k_2	0.95
v	2.1 m/s.
A	0.215 m^2

-Active belt width:

$$b_1 = \sqrt{\frac{A \cdot 3600}{f}} = 1{,}2 \text{ m}$$

where:

b_1 – active belt width, in m

f – factor of filling section shape. For troughed shape, with lateral angles of 35°, the factor has a value, f = 550

Active belt width is 1.5m, i.e. the width of the chute outlet under the crusher is 1.5 m.

-The required circumferential force on the edge of drive drum is calculated from the expression:

$$F_{bo} = C \cdot t \cdot L \cdot g \cdot (G_t + \frac{Q_h}{3{,}6 \cdot V}) + \\ + \frac{Q_h \cdot H}{3{,}6 \cdot V} daN$$

C - coefficient that takes into account the increase of traction due to side resistances along conveyor.

It depends on the length of conveyor. For this case, the value of 1.05 will be taken

L (m) – conveyor length

t - coefficient of friction in bearings (drums and rolls). For conditions of increased dust, the value of 0.015 will be taken for rolls with ball bearings takes the ball-bearing rollers.

g (m/sec^2) – gravity acceleration

G_t (kg/m) – mass of moving parts of conveyor per 1 m length of conveyor

Q_h (t/h) – weight of transported material in t/h

v (m/s) – rubber conveyor belt speed

H (m) – height of lifting or lowering loads

Table 2. *Parameter values*

Parameter	
C (/)	1.05
L (m)	25
T (/)	0.015
G_t(kg/m)	137.303
Q_h (t/h)	2350
H (m)	4.15
v (m/s)	2.47
b (m)	2200
F_{bo}	3661daN

- Power on a shaft of drive drum is calculated from the expression:

$$P_{bo} = \frac{F_{bo} \cdot v}{102},$$

P_{bo} = 75 kW

- Additional required power due to the belt cleanera is calculated from the expression:

$$P_d = 1.6 \cdot b \cdot v \cdot n$$

where:

n – number of scrapers

$$P_d = 1{,}6 \cdot 2{,}2 \cdot 2{,}47 \cdot 2 = 17\,kW$$

- Additional required power due to the lateral guides:

$$P_v = 0{,}08 \cdot 10\,m = 0{,}8\ kW$$

where:

- Effective power on a shaft of drive drum is calculated from the expression:

$$P_e = P_{bo} + P_d + P_v,$$

P_e=75+17+0,8=92,8 kW

- Power of drive electric motor is calculated from the expression:

$$P_{el} = \frac{P_e}{\eta} = \frac{P_e}{0{,}89 \cdot 0{,}95}$$

where:

η- degree of efficiency

$P_{el.}$ = 109.75kW

Electric motor power of 110 kW is adopted.

CONCLUSION

This analysis has confirmed that the belt feeder with the existing electric motor, power of 110 kW, and the existing rubber belt, can not meet the required capacity of 2350 t/h. Interestingly, it is in this analysis that the calculated power corresponds to the adopted standard power of electric motor, what is important for designers in making decisions about selection of electric motor power. According to this analysis, the investor has taken all necessary activities to carry out the relocation of the existing conveyor from the Majdanpek mine location, and with additional modifications and reparations to install it at the new location of crushing plant No. 2, at the open pit of Veliki Krivelj mine. After a year of constructed facility, it was shown that this computational confirmation was correct.

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EKONOMSKE PROJEKCIJE VALORIZACIJE BAKRA IZ OŠTRELJSKIH PLANIRA I STAROG FLOTACIJSKOG JALOVISTA**

Izvod

Ekonomske projekcije se zasnivaju na stručnom sagledavanju godišnje valorizacije bakra iz Oštreljskih planira i starog flotacijskog jalovišta za narednih deset godina u cilju okvirnog sagledavanja isplativosti proizvodnje ukupnog kapaciteta oko 3040 t katodnog bakra na osnovu složenih tehničko tehnoloških istraživanja. Preliminarna ekonomska sagledavanja pokazala su visoko pozitivne rezultate uspešnosti i isplativosti kroz bilans uspeha, ekonomski novčani tok, cenu koštanja itd.

Ključne reči: *valorizacija bakra, rezultat, uspešnost, isplativost*

1. UVOD

1. Na osnovu definisanih tehničko tehnoloških rešenja valorizacije bakra iz Oštreljskih planira i starog flotacijskog jalovišta, procenjena ukupna invesiciona ulaganja u osnovna sredstva iznose: 13.495.000 USD.

- Iznos ukupnog kredita 12.500.000 USD
- Rok vraćanja kredita je 8 godina, kamatna stopa je 10% godišnje, jednaki godišnji anuitet

Tabela 1. *Ukupna investiciona ulaganja i konstrukcija finansiranja u 000$*

I. UKUPNO OSNOVNA S.	13495	13495
OBRTNA SREDSTVA	3200	3200
UKUPNE INVESTIC.	16695	16695
II. IZVORI		
1.Sopstvena sred.	4195	4195
2.Krediti	12500	12500
UKUPNI IZVORI	16695	16695

* *Institut za rudarstvo i metalurgiju Bor*

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Tabela 1.1. *Plan otplate kredita* u 000$

NAZIV KREDITA: KREDIT 1		USD:	500.00	
USLOVI: ROK: 5 KAM.: 10.000%				
1	132	50	82	500
2	132	42	90	418
3	132	33	99	328
4	132	23	109	229
5	132	12	120	120
Tot:	660	160	500	
Pros:	132	32	100	
NAZIV KREDITA: KREDIT 2		USD:	12000.00	
USLOVI: ROK: 8 KAM.: 10.000%				
1	2249	1200	1049	12000
2	2249	1095	1154	10951
3	2249	980	1270	9796
4	2249	853	1397	8527
5	2249	713	1536	7130
6	2249	559	1690	5594
7	2249	390	1859	3904
8	2249	204	2045	2045
Tot:	17992	5994	12000	
Pros:	2249	749	1500	
UKUPNO	18652	6154	12500	

2. OBRAČUN PRIHODA

Prihod je obračunat na bazi planiranih količina katodnog bakra i prodajnih cena .

Tabela 2. *Obračun prihoda* u 000$

PROIZ-VOD	KOLICINA	PRODAJNA CENA	UKUPNI PRIHOD	PRIHOD DALJE PRERADE	CIST PRIHOD
* GODINA:	1-10				
1 Cu	1500.0000	4.2000	6,300.00	0.00	6,300.00
1 Cu	1540.0000	4.2000	6,468.00	0.00	6,468.00
** GODISNJI PRIHOD			12,768.00	0.00	12,768.00
* UKUPNI PRIHOD			127,680.00	0.00	127,680.00

3. CENA KOŠTANJA

1) Ukupna cena koštanja 3.040 t katodnog bakra godišnje.

Tabela 3. *Ukupna cena koštanja* u 000 $

GODINE	1	2	3	4	5	6
1. Sirov.i mater.	2620	2620	2620	2620	2620	2620
2. Energija	182	182	182	182	182	182
- Elektrika	182	182	182	182	182	182
3. Odrzavanje	1271	1271	1271	1271	1271	1270
4. Amortizacija	1501	1501	1501	1501	1501	1496
5. Ostali mat.tr.	1000	1000	1000	1000	1000	1000
6. Nemater.trosk.	800	800	800	800	800	800
7. Licni dohoci	420	420	420	420	420	420
8. Kamate	1250	1137	1012	876	725	559
9. Osiguranje	270	270	270	270	270	269
I. TROS.POSLOVANJA	9313	9200	9076	8939	8788	8616
10. Zakonske obav.	345	357	369	383	398	415

-NASTAVAK	7	8	9	10	UKUPNO	PROSEK
1. Sirov.i mater.	2620	2620	2620	2620	26200	2620
2. Energija	182	182	182	182	1815	182
- Elektrika	182	182	182	182	1815	182
3. Odrzavanje	1270	1270	375	375	10914	1091
4. Amortizacija	1496	1496	751	751	13495	1350
5. Ostali mat.tr.	1000	1000	1000	1000	10000	1000
6. Nemater.trosk.	800	800	800	800	8000	800
7. Licni dohoci	420	420	420	420	4200	420
8. Kamate	390	204			6154	615
9. Osiguranje	269	269	150	150	2458	246
I. TROS.POSLOVANJA	8447	8261	6298	6298	83236	8324
10. Zakonske obav.	432	451	647	647	4444	444
II.PUNA CENA KOST.	8879	8712	6945	6945	87681	8768

Prosečna cena koštanja iznosi: 2884,2 USD/T KATODE

4. BILANS USPEHA

Tabela 4. *Bilans uspeha* u 000 $

GODINE	1	2	3	4	5	6
A. UKUPNI PRIHOD	12768	12768	12768	12768	12768	12768
B. UKUPNI RASHODI	9313	9200	9076	8939	8788	8616
1.POSLOV.RASHODI	8063	8063	8063	8063	8063	8057
-Mater.troskovi	2620	2620	2620	2620	2620	2620
-Energija	182	182	182	182	182	182
-Odrzavanje	1271	1271	1271	1271	1271	1270
-Osiguranje	270	270	270	270	270	269
-Amortizacija	1501	1501	1501	1501	1501	1496
-Usl.i ost.mat.	1000	1000	1000	1000	1000	1000
-Nemater.trosk.	800	800	800	800	800	800
-Licni dohoci	420	420	420	420	420	420
2.RASHODI FINANS.	1250	1137	1012	876	725	559
-Kamata	1250	1137	1012	876	725	559
C. BRUTO DOBIT	3455	3568	3692	3829	3980	4152
-Porezi i dop.	345	357	369	383	398	415
E. NETO DOBIT	3109	3211	3323	3446	3582	3737

-NASTAVAK	7	8	9	10	UKUPNO	PROSEK
A. UKUPNI PRIHOD	12768	12768	12768	12768	127680	12768
B. UKUPNI RASHODI	8447	8261	6298	6298	83236	8324
1.POSLOV.RASHODI	8057	8057	6298	6298	77082	7708
-Mater.troskovi	2620	2620	2620	2620	26200	2620
-Energija	182	182	182	182	1815	182
-Odrzavanje	1270	1270	375	375	10914	1091
-Osiguranje	269	269	150	150	2458	246
-Amortizacija	1496	1496	751	751	13495	1350
-Usl.i ost.mat.	1000	1000	1000	1000	10000	1000
-Nemater.trosk.	800	800	800	800	8000	800
-Licni dohoci	420	420	420	420	4200	420
2.RASHODI FINANS.	390	204			6154	615
-Kamata	390	204			6154	615
C. BRUTO DOBIT	4321	4507	6470	6470	44444	4444
-Porezi i dop.	432	451	647	647	4444	444
E. NETO DOBIT	3889	4056	5823	5823	39999	4000

5. EKONOMSKI NOVČANI TOK

Tabela 5. *Ekonomski novčani tok* u 000 $

GODINE	1	2	3	4	5	6
I. NOVCANI PRILIVI						
1. Ukupni prihod	12768	12768	12768	12768	12768	12768
-Realizacija	12768	12768	12768	12768	12768	12768
-Ostali prihodi						
2. Ostatak vred.						
-Osnovna sreds.						
-Obrtna sreds.						
UKUPNI PRILIV	12768	12768	12768	12768	12768	12768
II. NOVCANI ODLIVI						
4. Investicije	16695					
6. Troskovi posl.	6142	6142	6142	6142	6142	6141
7. Licni dohoci	420	420	420	420	420	420
8. Zakonske obav.	345	357	369	383	398	415
UKUPNI ODLIV	23603	6919	6931	6945	6960	6976
NETO EKONOM. TOK	-10835	5849	5837	5823	5808	5792
Kumulativ	-10835	-4986	851	6674	12481	18274
DISKONTOVANA VRED:						
-Sa 8.00 %	-10835	5416	5004	4622	4269	3942
-Sa 10.00 %	-10835	5317	4824	4375	3967	3597
-Sa 12.00 %	-10835	5222	4653	4145	3691	3287
-Sa 53.17 % (ISR)	-10835	3819	2488	1620	1055	687

-NASTAVAK	7	8	9	10	UKUPNO	PROSEK
I. NOVCANI PRILIVI						
1. Ukupni prihod	12768	12768	12768	12768	127680	12768
-Realizacija	12768	12768	12768	12768	127680	12768
-Ostali prihodi						
2. Ostatak vred.						
-Osnovna sreds.						
-Obrtna sreds.				3200		
UKUPNI PRILIV	12768	12768	12768	15968	130880	13088
II. NOVCANI ODLIVI						
4. Investicije					16695	1670
6. Troskovi posl.	6141	6141	5127	5127	59387	5939
7. Licni dohoci	420	420	420	420	4200	420
8. Zakonske obav.	432	451	647	647	4444	444
UKUPNI ODLIV	6993	7011	6194	6194	84726	8473
NETO EKONOM. TOK	5775	5757	6574	9774	46154	4615
Kumulativ	24049	29806	36380	46154		
DISKONTOVANA VRED:						
-Sa 8.00 %	3639	3359	3552	4889	27858	2786
-Sa 10.00 %	3260	2954	3067	4145	24670	2467
-Sa 12.00 %	2926	2604	2655	3525	21872	2187
-Sa 53.17 % (ISR)	447	291	217	211	0	

6. FINANSIJSKI NOVČANI TOK

Tabela 6. *Finansijski novčani tok* u 000 $

GODINE	1	2	3	4	5	6
I. NOVCANI PRILIVI						
1. Ukupni prihod	12768	12768	12768	12768	12768	12768
-Realizacija	12768	12768	12768	12768	12768	12768
-Ostali prihodi						
2. Sopstvena sr.	4195					
3. Krediti	12500					
4. Ostatak vred.						
-Osnovna sreds.						
-Obrtna sreds.						
UKUPNI PRILIV	29463	12768	12768	12768	12768	12768
II. NOVCANI ODLIVI						
5. Investicije	16695					
7. Mater.rashodi	6142	6142	6142	6142	6142	6141
8. Licni dohoci	420	420	420	420	420	420
9. Zakonske obav.	345	357	369	383	398	415
10. Rashodi finan.	1250	1137	1012	876	725	559
11. Otplate	1131	1244	1369	1506	1656	1690
UKUPNI ODLIV	25984	9300	9313	9326	9341	9225
NETO FINANS. TOK	3479	3468	3455	3442	3427	3543
Kumulativ	3479	6947	10402	13844	17270	20813
DISKONTOVANA VRED:						
-Sa 8.00 %	3479	3211	2962	2732	2519	2411
-Sa 10.00 %	3479	3152	2856	2586	2340	2200
-Sa 12.00 %	3479	3096	2755	2450	2178	2010

-NASTAVAK	7	8	9	10	UKUPNO	PROSEK
I. NOVCANI PRILIVI						
1. Ukupni prihod	12768	12768	12768	12768	127680	12768
2. Sopstvena sr.					4195	420
3. Krediti					12500	1250
4. Ostatak vred.						
-Osnovna sreds.						
-Obrtna sreds.				3200		
UKUPNI PRILIV	12768	12768	12768	15968	147575	14758
II. NOVCANI ODLIVI						
5. Investicije					16695	1670
7. Mater.rashodi	6141	6141	5127	5127	59387	5939
8. Licni dohoci	420	420	420	420	4200	420
9. Zakonske obav.	432	451	647	647	4444	444
10. Rashodi finan.	390	204			6154	615
11. Otplate	1859	2045			12500	1250
UKUPNI ODLIV	9242	9261	6194	6194	103381	10338
NETO FINANS. TOK	3526	3507	6574	9774	44194	4419
Kumulativ	24339	27847	34421	44194		
DISKONTOVANA VRED:						
-Sa 8.00 %	2222	2047	3552	4889	30024	3002
-Sa 10.00 %	1990	1800	3067	4145	27615	2762
-Sa 12.00 %	1786	1587	2655	3525	25520	2552

Projekat je likvidan tokom celog perioda.

7. DINAMIČKA OCENA

Ova ocena je vrlo pozitivna po svim metodama:

- Intrena stopa prinosa iznosi 53,17%
- Period povratka ulaganja je 2 godine
- Neto sadašnja vrednost projekta sa 10% iznosi: 24.670.000 USD

8. STATIČKI POKAZATELJI U 5. GODINI VEKA PROJEKTA

1. **Koeficijent ekonomičnosti:**

Ukupan prihod/ukupni troškovi
= 12.768.000/8.788.000=**1,45**

2. **Stopa akumulativnosti:**

Dobit/ukupna ulaganja
= 3.980.000/16.695.000=**23,8%**

3. **Prosta stopa prinosa:**

Neto dobit/ukupna ulaganja
= 3.582.000/16.695.000=**21,46%**

4. **Reproduktivnost:**

Neto dobit + Amortizacija/ Ukupne investicije
= 5.083.000/16.695.000= **30,44%**

5. **Stopa dobiti:** dobit/ realizacija
= 3.980.000/12.768.000=**31%**

9. STATIČKA ANALIZA OSETLJIVOSTI

1) Određivanje prelomne tačke rentabiliteta tj. kritičnog kapaciteta:

Minimalno iskorišćen stepen kapaciteta određuje prelomnu tačku u korišćenju proizvodnih kapaciteta tj. određuje najniži nivo njihovog korišćenja na kojoj se projekat još uvek nalazi iznad zone gubitka.

$$BREK\ EVEN\ POINT\left(BEP\right)=\frac{Uk.\,fiksni\ troskovi}{Uk.\,prihod-Uk.\,varijabilni\ troskovi}\cdot 100=$$

$$=\frac{5.704.000}{12.768000-2.620.000}=56{,}2\%,\ \text{tj.}1708\ \text{t ukupnog katodnog Cu}$$

Na toj tački su prihodij jednaki troškovima i nema dobiti. Iznad te tačke je dobit, a ispod gubitak.

10. DINAMIČKA ANALIZA OSETLJIVOSTI

TABELA OSETLJIVOSTI

Tabela 10. *Tabela osetljivosti sa zadatim promenama*

%Promene	-60%	-50%	-40%	-30%	-20%	-10%	0%	+10%	+20%	+30%	+40%	+50%	+60%
PRIHOD:													
-E :	0	0	0	12	24	38	53	72	95	> 100			
-F :	0	0	0	5	16	27	40	55	73	96	0	0	0
-D :	0	0	0	12	26	41	59	81	0	0	0	0	0
TROSKOVI:													
-E :	0	0	99	85	73	63	53	45	37	30	23	17	11
-F :	0	88	76	65	56	48	40	33	27	21	15	10	4
-D :	0	0	0	98	83	70	59	49	40	32	24	17	11
INVESTICIJE:													
-E :				>100	78	63	53	46	40	35	31	24	20
-F :	0	0	90	69	56	47	40	35	31	27	24	21	19
-D :	0	0	0	0	87	70	59	50	44	39	34	31	28
PLATE:													
-E :	57	56	55	55	54	54	53	53	52	52	51	50	50
-F :	43	42	42	41	41	40	40	40	39	39	38	38	37
-D :	63	62	61	61	60	59	59	58	57	57	56	56	55
ZAKONSKE:													
-E :	56	56	55	55	54	54	53	53	52	52	51	50	50
-F :	43	42	42	41	41	40	40	40	39	39	38	38	37
-D :	59	59	59	59	59	59	59	59	59	59	59	59	59

Legenda: E = Ekonomska interna stopa rentabiliteta.
F = Finansijska interna stopa rentabiliteta.
D = Drustvena interna stopa rentabiliteta.

Iz prezentirane tabele sledi da je projekat najosetljiviji na promenu prihoda. Pad prihoda od 30 % dovodi do pada ISR na 12% što je jednako ceni pozajmljenih sredstava.

Analizom osetljivosti **povećanjem vrednosti ulaganja** sofverom „*REMIP*-računarsko ekonomski model investicionog planiranja "* dolazi se do sledećeg:

+60%

ISR-19,7%

NSV(12%) -6.633.000 USD

Period povratka ulaganja-5 godina

+50%

ISR-23,77%

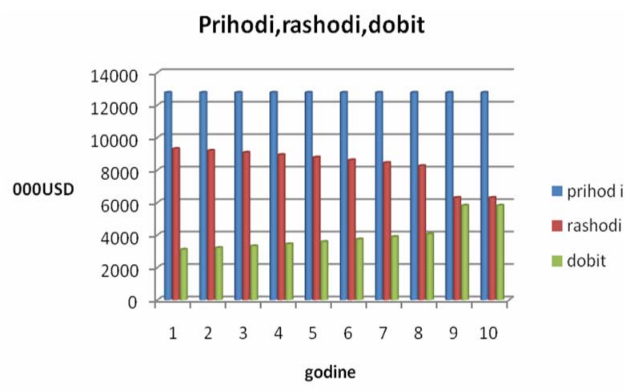
NSV(12%)-9.490.000 USD

Period povratka ulaganja-4 godine

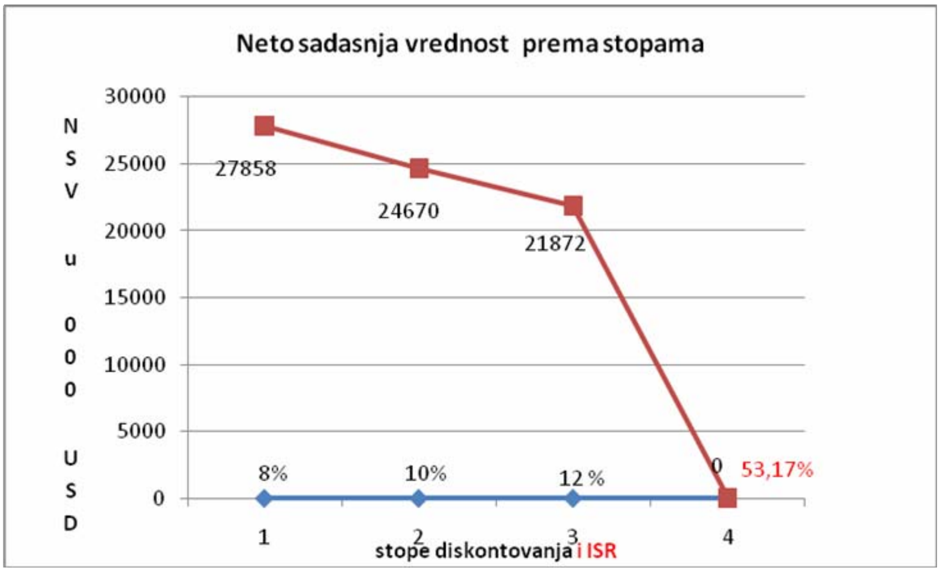
11. ZBIRNA OCENA

OPIS	vrednost	j.m.
1. VEK PROJEKTA	10	God.
2.UKUPNE INVESTICIJE : OSNOVNA SREDSTVA OBRTNA SREDSTVA	16.695.000 13.495.000 3.200.000	USD
3. PRIHOD		
- Ukupan prihod	127.680.000	USD
- Prosečni godišnji prihod	12.768.000	USD
4. RASHOD		
- Ukupni troškovi	83.236.000	USD
- Pros. godišnji troškovi	8.324.000	USD
5. DOBIT		
-Ukupna bruto dobit	44.444.000	USD
-Prosečna god. bruto dobit	4.444.000	USD
-Ukupna neto dobit	39.999.000	USD
-Prosečna god.neto dobit	4.000.000	USD
6. Prose~na Cena koštanja po t katode	2884,2	USD
7. Ukupna stopa dobiti	34,8	%
8.BEP	56,2	%
9. POKAZATELJI USPEŠNOSTI:		
ISR - Interna stopa rentabilnosti	53,7	%
PP - Period povraćaja sredstava.	2	GOD.
NSV - Neto sadašnja vrednost (10%)	24.670.000	USD

Na osnovu tehničko-tehnoloških rešenja ekonomska analiza je pokazala **visoko pozitivne rezultate**.



Grafik 1. *Grafik prihoda, rashoda i dobiti*



Grafik 2. *Grafik neto sadašnje vrednosti prema stopama diskontovanja*

ZAKLJUČAK

Ekonomska analiza i mogućnost povratka investicija zasnovana je na podacima tehnoloških mogućnosti proizvodnje godišnje količine 3040 t katodnog bakra. Zbirna ocena ekonomske analize pokazuje sledeće:

VEK PROJEKTA:10 godina, INVESTICIJE U OSNOVNA SREDSTVA 13.495.000 USD, PRIHOD: Ukupan prihod 127.680.000 USD, Prosečni godišnji prihod 12.768.000 USD, RASHOD: Ukupni troškovi 83.236.000 USD, Prosečni godišnji troškovi 8.324.000 USD; DOBIT:

Ukupna bruto dobit 44.444.000 USD, Prosečna godišnja bruto dobit 4.444.000 USD, Ukupna neto dobit 39.999.000 USD, Prosečna neto dobit 4.000.000 USD; Prosečna cena koštanja po t katode 2884,2 USD, Ukupna stopa dobiti 34,8%, BEP 56,2%; POKAZATELJI USPEŠNOSTI: Interna stopa rentabilnosti 53,7%, Period povraćaja sredstava 2 godine, Neto sadašnja vrednost (10%) 24.670.000 USD. Navedeni podaci pokazuju da, na osnovu datih tehničkih elemenata, polazna sagledavanja pokazuju visoko pozitivne rezultate.

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Gordana Slavković, Mile Bugarin*, Radojka Jonović**

ECONOMIC PROJECTIONS OF COPPER VALORIZATION FROM THE OSTRELJ WASTE DUMPS AND THE OLD FLOTATION TAILING DUMP**

Abstract

The economic projections are based on the expert insight of the annual copper valorization from the Ostrelj waste dumps and the Old Flotation tailing dump for the next ten years to the aim of approximate consideration the cost-effectiveness of production the total capacity of about 3040 t of cathode copper, based on a complex technical and technological investigations. Preliminary economic considerations have showed highly positive results of the success and profitability through the balance of success, economic cash flow, cost price and so on.

Key words: *copper valorization, result, success, cost effectiveness*

1. INTRODUCTION

Based on the defined technical-technological parametars for copper valorization from the Ostrelj waste dumps and the Old Flotation tailing dump, the estimated and defined total investments for the fixed assets are USD 13.495.000.

- Total loan is 12.500.000 USD
- Repayment period 8 years, the interest rate is 10%, equal to the annual instalments.

Table 1. *Total investments and financial construction*...................................In 000 $

I.Total fixed assets	13495	13495
Current assets	3200	3200
Total investments	16695	16695
II.SOURCES		
1.Self funds	4195	4195
2.Loans	12500	12500
Total sources	16695	16695

* *Mining and Metallurgy Institute Bor*

** *This paper is the result of the Project No. 37001 "The Impact of Mining Waste from RTB Bor on Pollution of Surrounding Watercourses 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*

Table 1.1. *Plan of loan repayment* **..**In 000 $

Name of loan : loan 1	USD:	500.00		
TERMS: 5 YEARS	RATE: 10.000%			
Year	repayment	rate	nstilment	loan
1	132	50	82	500
2	132	42	90	418
3	132	33	99	328
4	132	23	109	229
5	132	12	120	120
Total:	660	160	500	
Pros:	132	32	100	
Name of loan: Loan 2	USD:	12000.00		
Terms: 8 years	RATE.: 10.000%			
1	2249	1200	1049	12000
2	2249	1095	1154	10951
3	2249	980	1270	9796
4	2249	853	1397	8527
5	2249	713	1536	7130
6	2249	559	1690	5594
7	2249	390	1859	3904
8	2249	204	2045	2045
Total:	17992	5994	12000	
Pros:	2249	749	1500	
Total	18652	6154	12500	

2. CALCULATION OF REVENUE

Revenue was calculated on the basis of copper cathode quantity and selling prices.

Table 2. *Calculation of revenue* ....In 000 $

	PRODUCT	PRICE	REVENUE	REVENUE
* year :	1-10			
1 Cu	1500.0000	4.2000	6,300.00	6,300.00
1 Cu	1540.0000	4.2000	6,468.00	6,468.00
** Annual revenue			12,768.00	12,768.00
* Total revenue			127,680.00	127,680.00

3. PRODUCT PRICE

1) Total product price of 3040 t copper cathode per year

Table 3. *Total product price*...In 000 $

Years	1	2	3	4	5	6
1. Raw &material .	2620	2620	2620	2620	2620	2620
2. Energy	182	182	182	182	182	182
3. Maintenance	1271	1271	1271	1271	1271	1270
4. Amortization	1501	1501	1501	1501	1501	1496
5. Other mat.costs	1000	1000	1000	1000	1000	1000
6. Non-mater.costs	800	800	800	800	800	800
7. Salaries	420	420	420	420	420	420
8. Interests	1250	1137	1012	876	725	559
9. Insurance	270	270	270	270	270	269
I. TOTAL	9313	9200	9076	8939	8788	8616
10. Taxes	345	357	369	383	398	415

Years	7	8	9	10	TOTAL	AVERAGE
1. Raw &material	2620	2620	2620	2620	26200	2620
2. Energy	182	182	182	182	1815	182
3. Maintenance	1270	1270	375	375	10914	1091
4. Amortization	1496	1496	751	751	13495	1350
5. Other mat.costs	1000	1000	1000	1000	10000	1000
6. Non-mater.costs	800	800	800	800	8000	800
7. Salaries	420	420	420	420	4200	420
8. Interests	390	204			6154	615
9. Insurance	269	269	150	150	2458	246
I. TOTAL	8447	8261	6298	6298	83236	8324
10. Taxes	432	451	647	647	4444	444
II.Full product price	8879	8712	6945	6945	87681	8768

Average product price is 2884.2 USD/t cathode.

4 PROFIT & LOSS ACCOUNT

Table 4. *Profit & Loss Account*...In 000 $

Years	1	2	3	4	5	6
A. Revenue	12768	12768	12768	12768	12768	12768
B. Costs	9313	9200	9076	8939	8788	8616
1.Operating costs	8063	8063	8063	8063	8063	8057
-Raw &material	2620	2620	2620	2620	2620	2620
-Energy	182	182	182	182	182	182
-Maintenance	1271	1271	1271	1271	1271	1270
-Insurance	270	270	270	270	270	269
-Amortization	1501	1501	1501	1501	1501	1496
-Serv.other mat.	1000	1000	1000	1000	1000	1000
-Non-mater.costs	800	800	800	800	800	800
-Salaries	420	420	420	420	420	420
2.Financial costs	1250	1137	1012	876	725	559
-Interests	1250	1137	1012	876	725	559
C. Gross profit	3455	3568	3692	3829	3980	4152
-Taxes	345	357	369	383	398	415
E. Net profit	3109	3211	3323	3446	3582	3737

-continuation	7	8	9	10	total	average
A. Revenue	12768	12768	12768	12768	127680	12768
B. Costs	8447	8261	6298	6298	83236	8324
1. Operating costs	8057	8057	6298	6298	77082	7708
- Raw &material	2620	2620	2620	2620	26200	2620
- Energy	182	182	182	182	1815	182
- Maintenance	1270	1270	375	375	10914	1091
- Insurance	269	269	150	150	2458	246
- Amortization	1496	1496	751	751	13495	1350
- Serv.other mat.	1000	1000	1000	1000	10000	1000
- Non-mater.costs	800	800	800	800	8000	800
- Salaries	420	420	420	420	4200	420
2. .Financial costs	390	204			6154	615
- Interests	390	204			6154	615
C. Gross profit	4321	4507	6470	6470	44444	4444
- Taxes	432	451	647	647	4444	444
E. Net profit	3889	4056	5823	5823	39999	4000

5. ECONOMIC CASH FLOW

Table 5. *Economic Cash Flow* ....In 000 $

Years	1	2	3	4	5	6
I. CACH INFLOW						
1. CASH FROM SALES	12768	12768	12768	12768	12768	12768
2. Residual						
-Fixed assets						
-Current assets						
TOTAL	12768	12768	12768	12768	12768	12768
II. CASH OUTFLOWS						
4. Investments	16695					
6. Operational costs	6142	6142	6142	6142	6142	6141
7. Salaries	420	420	420	420	420	420
8. Taxes	345	357	369	383	398	415
TOTAL	23603	6919	6931	6945	6960	6976
Net cash flow	-10835	5849	5837	5823	5808	5792
Cumulative	-10835	-4986	851	6674	12481	18274
DISCOUNTED VALUE						
- 8.00 %	-10835	5416	5004	4622	4269	3942
-10.00 %	-10835	5317	4824	4375	3967	3597
-12.00 %	-10835	5222	4653	4145	3691	3287
-53.17 % (IRR)	-10835	3819	2488	1620	1055	687

-continuation	7	8	9	10	TOTAL	AVERAGE
I. CACH INFLOW						
1. CASH FROM SALES	12768	12768	12768	12768	127680	12768
2. Residual						
- Fixed assets						
- Current assets				3200		
TOTAL	12768	12768	12768	15968	130880	13088
II. CASH OUTFLOWS						
4. Investments					16695	1670
6. Operational costs	6141	6141	5127	5127	59387	5939
7. Salaries	420	420	420	420	4200	420
8. Taxes	432	451	647	647	4444	444
TOTAL	6993	7011	6194	6194	84726	8473
Net cash flow	5775	5757	6574	9774	46154	4615
Cumulative	24049	29806	36380	46154		
DISCOUNTED VALUE						
-8.00 %	3639	3359	3552	4889	27858	2786
-10.00 %	3260	2954	3067	4145	24670	2467
-12.00 %	2926	2604	2655	3525	21872	2187
-53.17 % (IRR)	447	291	217	211	0	

6. FINANCIAL CASH FLOW

Table 6. *Financial Cash Flow*...In 000 $

Years	1	2	3	4	5	6
I. CACH INFLOW						
1. CASH FROM SALES	12768	12768	12768	12768	12768	12768
2. Self funds	4195					
3. Loans	12500					
4. Residual						
- Fixed assets						
- Current assets				.		
TOTAL	29463	12768	12768	12768	12768	12768
II. CASH OUTFLOWS						
5. Investments	16695					
7. Operational costs	6142	6142	6142	6142	6142	6141
8. Salaries	420	420	420	420	420	420
9. Taxes	345	357	369	383	398	415
10.Interests	1250	1137	1012	876	725	559
11. Repayments	1131	1244	1369	1506	1656	1690
Total	25984	9300	9313	9326	9341	9225
Net cash flow	3479	3468	3455	3442	3427	3543
Cumulative	3479	6947	10402	13844	17270	20813
DISCOUNTED VALUE						
- 8.00 %	3479	3211	2962	2732	2519	2411
- 10.00 %	3479	3152	2856	2586	2340	2200
- 12.00 %	3479	3096	2755	2450	2178	2010

-continuation	7	8	9	10	total	average
I. CACH INFLOW						
1. CASH FROM SALES	12768	12768	12768	12768	127680	12768
2. Self funds					4195	420
3. Loans					12500	1250
4. Residual						
- Fixed assets						
- Current assets				3200		
TOTAL	12768	12768	12768	15968	147575	14758
II. CASH OUTFLOWS						
5. Investments					16695	1670
7. Operational costs	6141	6141	5127	5127	59387	5939
8. Salaries	420	420	420	420	4200	420
9. Taxes	432	451	647	647	4444	444
10. Interests	390	204			6154	615
11. Repayments	1859	2045			12500	1250
TOTAL	9242	9261	6194	6194	103381	10338
NETO FINANS. TOK	3526	3507	6574	9774	44194	4419
Cumulative	24339	27847	34421	44194		
DISCOUNTED VALUE						
- 8.00 %	2222	2047	3552	4889	30024	3002
-10.00 %	1990	1800	3067	4145	27615	2762
-12.00 %	1786	1587	2655	3525	25520	2552

The Project is liquid throughout the period.

7. DYNAMIC EVALUATION

Dynamic evaluation is very positive by all methods:

- Internal rate of return is 53.17%
- Payback period is 2 years
- Net present value (10%) is: 24.670.000 USD

8. STATIC EVALUATION IN THE 5. YEAR OF THE PROJECT

1. Coefficient in the economics:
Revenue/total costs
= 12.768.000/8.788.000=**1.45**

2. Rate of accumulation:
Gross profit/total investments
= 3.980.000/16.695.000=**23.8%**

3. Simple rate of return:

Net profit/total investments
= 3.582.000/16.695.000=**21.46%**

4. Reproduction:

Net profit+Amortization/ total investments
=5.083.000/16.695.000= 30.44%

5. Rate of profit:

Gross profit /revenue
=3.980.000/12.768.000=31%

9. STATIC SENSITIVITY ANALYSIS

$$\text{BREAK EVEN POINT(BEP)} = \frac{\textit{Total fixsed costs}}{\textit{Total revenue - Total variable costs}} \cdot 100 =$$

$$= \frac{5.704.000}{12.768.000 - 2.620.000} = 56.2\% 1708\,t \ \text{Total Cu cathode}$$

At that point, the revenues are equivalent to the costs. The profit is above that point, and the loss is below it.

10. DYNAMIC SENSITIVITY ANALYSIS

TABLE OF SENSITIVITY

Table 10. *Table of sensitivity with changes*

%changes	-60%	-50%	-40%	-30%	-20%	-10%	0%	+10%	+20%	+30%	+40%	+50%	+60%
REVENUE													
-E :	0	0	0	12	24	38	53	72	95	> 100			
-F :	0	0	0	5	16	27	40	55	73	96	0	0	0
-D :	0	0	0	12	26	41	59	81	0	0	0	0	0
COSTS													
-E :	0	0	99	85	73	63	53	45	37	30	23	17	11
-F :	0	88	76	65	56	48	40	33	27	21	15	10	4
-D :	0	0	0	98	83	70	59	49	40	32	24	17	11
INVESTMENTS													
-E :				>100	78	63	53	46	40	35	31	24	20
-F :	0	0	90	69	56	47	40	35	31	27	24	21	19
-D :	0	0	0	0	87	70	59	50	44	39	34	31	28
SALARIES													
-E :	57	56	55	55	54	54	53	53	52	52	51	50	50
-F :	43	42	42	41	41	40	40	40	39	39	38	38	37
-D :	63	62	61	61	60	59	59	58	57	57	56	56	55
TAXES													
-E :	56	56	55	55	54	54	53	53	52	52	51	50	50
-F :	43	42	42	41	41	40	40	40	39	39	38	38	37
-D :	59	59	59	59	59	59	59	59	59	59	59	59	59

Legend: E = Economic IRR
F = Financial IRR
S = Social IRR

From the present Table, it follows that the Project is the most sensitive on the revenue change. If the revenue falls 30%, IRR will be 12% and that is equivalent to the interest rate.

Sensitivity analysis **with an increase of investments,** using the software *"REMIP"*, the computer - economic model of investments planning results into the following:

+60% **+50%**

IRR – 19.7 % *IRR - 23/77 %*

NPV (12%) - 6.633.000 USD *NPV (12%) - 9.490.000 USD*

Payback period - 5 years *Payback period - 4 years*

11. TOTAL EVALUATION

DESCRIPTION	VALUE	U.M.
1. Period of the Project	**10**	**Year**
2.Total investments: **Fixed assets** **Current assets**	**16.695.000** **13.495.000** **3.200.000**	**USD**
3. REVENUE		
- Total revenue	**127.680.000**	**USD**
- Average revenue	**12.768.000**	**USD**
4. COSTS		
- Total costs	**83.236.000**	**USD**
- Average costs	**8.324.000**	**USD**
5. Profit		
-Total gross profit	**44.444.000**	**USD**
-Average gross profit	**4.444.000**	**USD**
-Total net profit	**39.999.000**	**USD**
-Average net profit	**4.000.000**	**USD**
6. Average product cathode price per tone	**2884.2**	**USD**
7. Total rate of profit	**34,8**	**%**
8.BEP	**56.2**	**%**
9.PAYABLE OF INVESTMENTS:		
IRR - Internal rate of return	**53.7**	**%**
PP - Payback period	**2**	**Year**
NPV – Net present value (10%)	**24.670.000**	**USD**

Based on the technical-technological parameters, the economic analysis has shown high positive results.

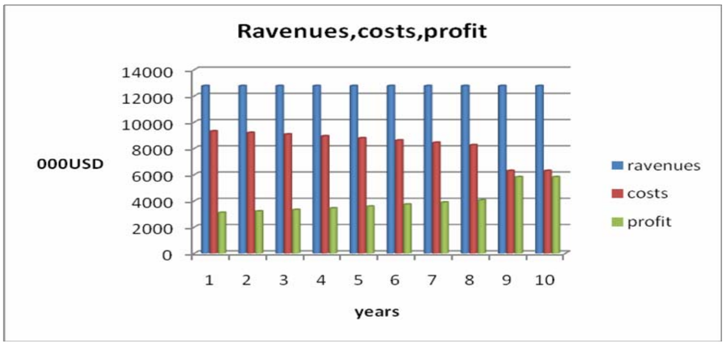

Figure 1. *Graph of the revenue, costs and profit*

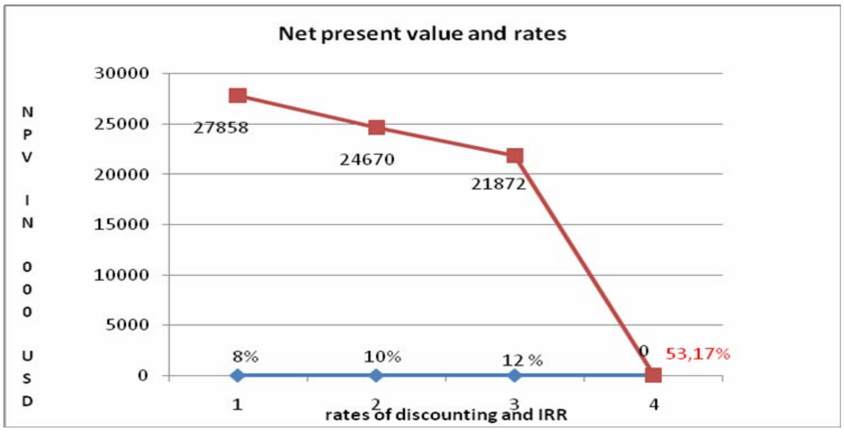

Figure 2. *Graph of the net present value according to the discount rates*

CONCLUSION

Economic analysis and possibility of investment return is based on data for the technical possibility of production the annual amount of 3040 t of cathode copper. Cumulative evaluation of economic analysis shows the following: PERIOD OF PROJECT: 10 years, INVESTMENTS IN FIXED ASSETS 13.495.000 USD, REVENUE: Total revenue 127.680.000 USD, Average revenue 12.768.000 USD, COSTS: Total costs 83.236.000 USD, Average costs 8.324.000 USD, PROFIT: Total gross profit 44.444.000 USD, Average gross profit 4.444.000 USD, Total net profit 39.999.000 USD, Average net profit 4.000.000 USD; Average product price 2884.2 USD per ton of copper cathode, Total rate of profit 34,8%,Break even point 56.2%; PAYABLE OF INVESTMENTS: Internal rate of return 53.7%, Payback period 2 years, Net present value (10%) 24.670.000 USD. The specified data have shown, according to the technical elements, that the initial considerations have shown high positive results.

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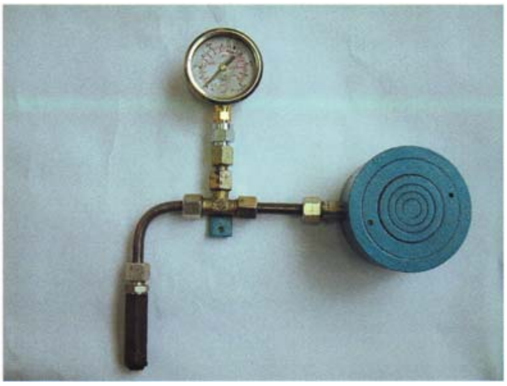

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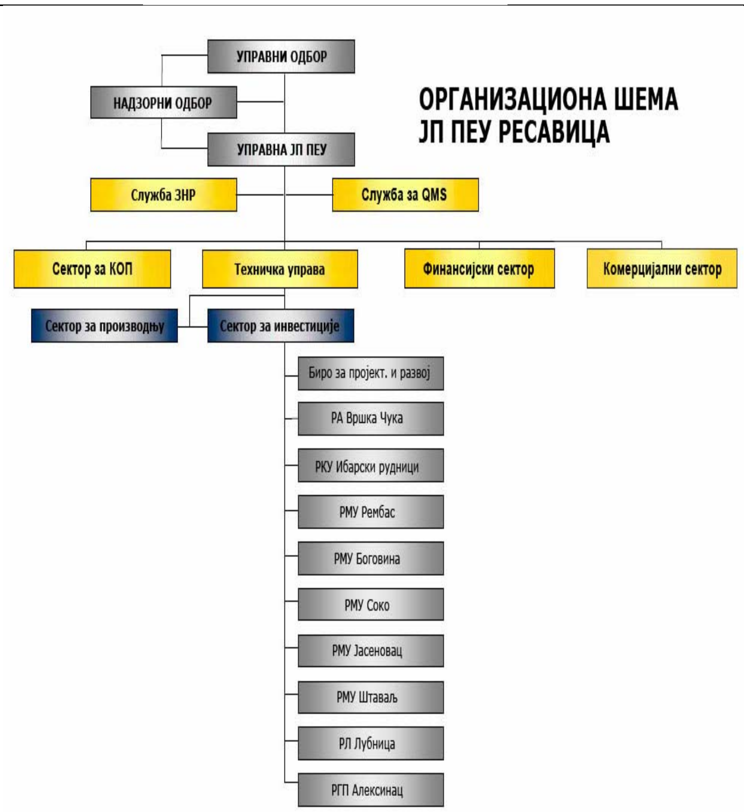

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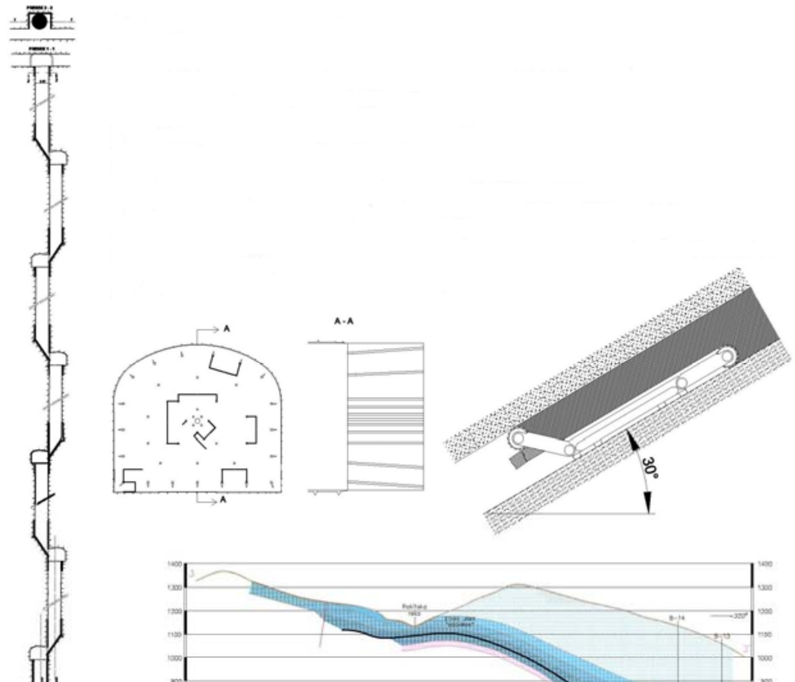

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