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Recovery of Sulfur, Copper, and Gold from Old Flotation Tailings at Bor, Serbia

Abstract: Pyritic mining and processing tailings often contain significant amounts of valuable metals, representing valuable sources of secondary raw materials. This is especially the case in earlystage mine cycle operations, in which the head grades were higher than nowadays, and the tailings still contain a considerable number of valuable metals due to inadequate separation processes compared to present technologies. Serbia has copper deposits that have been exploited since ancient times, and these operations have generated substantial amounts of mineral processing tailings. They have since been causing acid and metalliferous drainage, which is a substantial risk to the environment. The main objective of this study is to show how harmful and valuable elements such as sulfur, copper, and gold can be recovered from chemically and mineralogically challenging tailings. Flotation experiments were performed to evaluate the effects of particle size, various collectors, pH, and pulp potential on copper and sulfur recovery, and how copper-pyrite selective flotation can be applied. This evaluation included detailed chemical and mineralogical characterization of the tailings on laboratory scale. During the experimental study, copper and gold were recovered effectively into pyrite concentrate. Based on the simulation results with the flowsheet that was developed, a pyrite concentrate containing 48.7% S can be produced at 87% recovery. Moreover, the pyrite concentrate contains 1.59% Cu and nine ppm Au with a recovery of 80% and 49%, respectively. Although the pyrite concentrate produced does not meet with the flash smelter specifications for copper grade, the concentrate can be utilized in flash smelting as a source of energy and precious metals. Moreover, the SO₂ generated can be used in sulfuric acid production, thus reducing the environmental impact caused by acid mine drainage.

Keywords: copper, gold, mineralogy, tailings, flotation, value recovery, modeling

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1. Introduction

Efficient reprocessing of tailings, i.e., mine waste, is crucial to the elimination of environmental risks connected with tailing ponds, such as tailings dam failure and the generation of acid mine drainage. In the meantime, the high quantity of mineral processing tailings provides an economical source of metals that can be considered secondary deposits (Xie et al., 2005). Tailings also represent a valuable source of secondary raw materials. Therefore, tailings are nowadays considered globally as potentially exploitable and costsaving raw material sources. This potential is because they consist of already mined and comminuted minerals (Lutandula and Maloba, 2013; Falagán et al., 2017; Babel et al., 2018; Drobe et al., 2021; Shengo, 2021).

The current research is part of the European Institute of Innovation and Technology (EIT) funded RIS-CuRE (Regional Innovation Scheme-Zero waste recovery of copper tailings in the ESEE region) project, which aims at extraction of valuable metals from old tailings and utilization of cleaned tailings.

Serbia has copper deposits that have been exploited since ancient times, especially in Bor (a district in the eastern part of Serbia), dating back to 4500 BC. The Bor copper deposits mainly consist of 600 metric tons of porphyry and 200 metric tons of high sulfidation zone (Armstrong et al., 2005; Antonijević and Mijatović, 2014). These operations have generated large amounts of mineral processing tailings. The Bor copper mine started exploiting the high-grade (17% Cu) copper ores in 1903 (EJATLAS, 2021). Open-pit mining

Article

began in 1912 and continued until 1986 with decreasing copper grades (Urošević et al., 2018).

Approximately 700 million metric tons of waste rock and tailings were disposed of in the Bor Valley (Antonijević et al., 2008). The dumped tailings are highly acidic, with pH values ranging between 2 and 4.25 (Markovic et al., 2010), due to the decomposition of an abundance of pyrite, which has led to acid generation (Jambor, 1994; Đorđievski et al., 2018).

Overall, a century of mining has left its mark on the landscape of the Bor area, one of Serbia's most polluted places (Urošević et al., 2018). The soil surrounding the old flotation tailings pond is contaminated with metallic elements containing exceptionally high concentrations of iron, copper, and arsenic (Antonijević et al., 2012; Filimon et al., 2021). The acidic conditions facilitate the mobility and bioavailability of toxic heavy metals. Therefore, the outcomes of this study are of significant importance to save the Bor Valley from the ongoing pollution.

In the case of a tailings dam failure, there is the potential for a significant portion of the toxic material to run directly into the Borska River and onward into the Danube River, resulting in enormous environmental consequences for the entire region (Markovic et al., 2010; Stanković et al., 2018).

Reprocessing the tailings can reduce acid generation and the subsequent release of metals to waterways and soils by removing pyrite and simultaneous recovery of valuable metals. Several studies have been conducted on laboratory scale to recover copper and gold from the Bor tailings by acid leaching (Antonijević et al., 2008; Conić et al., 2020). Acid leaching followed by flotation of leaching residue (Stanojlović et al., 2014) led to copper and sulfur recoveries of 70% and 77%, respectively. Conić et al. (2020) concluded that the most efficient technology for copper recovery from the old flotation tailings of the Bor copper mine was leaching with biogenic lixiviant. Falagán et al. (2017) achieved copper recoveries of over 90% by bioleaching.

Bulatovic (2007) found that bulk flotation is typically used in cases where the ore is acidic and the pyrite content does not exceed 15% by weight. Markovic et al. (2010) and Han et al. (2014) used bulk sulfide flotation to recover valuable metals from the Bor tailings. Markovic et al. (2010) achieved a copper recovery of over 97% and a pyrite recovery of over 87% with bulk flotation in medium alkaline pulp conditions. The copper and sulfur grades in the bulk concentrate were 1.34% and 42.74%, respectively. Moreover, recoveries of gold and silver were above 60%. Similarly, Leppinen et al. (1997) used bulk sulfide flotation to remove sulfur and recover valuable pyrite-rich tailings to control acid mine drainage.

Quartz and clay minerals are common gangues of sulfide-type ore bodies and their tailings (Sun et al. 2016). Rao et al. (2011) found that sodium silicate is one of the most valuable depressants for silica gangues and that it can increase the selectivity of pyrite flotation.

2. Materials and Methods

2.1 Materials

The old Bor flotation tailings were disposed of in the Bor River valley from 1933 to 1987 in two tailings ponds covering an area of approximately 1.6 km². The ponds contain 27 million metric tons of tailings with an average range of 0.2-0.3% copper and 0.3-0.6ppm of gold (Serafimvoski et al., 2021).

As part of the RIS-CuRE project, a sample of approximately 230 kg of tailings was taken from the pond by drilling and delivered for characterization and beneficiation experiments to Metso Outotec Research Center, Pori, Finland. The sample was homogenized by shoveling and divided into subsamples by a rotating sample divider.

2.2 Methods

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The methods used consisted of four main steps, including a chemical and mineralogical study, grinding test, flotation experiments, and flowsheet simulation, as shown in Figure 1.



Figure 1. Overall diagram of the methodology.

2.3. Chemical composition of tailings

After total dissolution, the chemical analysis of metals in solids was conducted using Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES, Thermo Scientific iCAP 6000), The total carbon and sulfur concentrations were determined by the combustion method (Eltra CS-2000). The gold content was determined using Inductively Coupled Plasma Mass Spectrometry (ICP-MS, Thermo Scientific).

A four-step selective dissolution method was performed on the sized and bulk samples to quantify different copper minerals, according to the procedure described by Young (1974) and further developed by Metso Outotec. The selective dissolution method was carried out by analyzing the filtrated sample in dissolution by ICP-OES, with the residual continuing onto the next dissolution. The steps were as follows: P1) H₂O dissolution, P2) H₂SO4 dissolution, P3) KCN dissolution, and P4) HNO₃ + Br₂ dissolution. Based on this method, copper sulfates, copper oxides and carbonates, secondary copper sulfides, and primary chalcopyrite can be distinguished, and their amounts calculated using the element-to-mineral conversion (EMC) method.

2.4. Mineral composition of tailings

The mineralogical sample was screened using sieves from 600 μm to 20 μm in size. The P80 was calculated as 116 μm and the P50 as 29 μm . Seven polished resin mounts were prepared from the size fraction samples and from one bulk sample for the mineralogical studies. Due to the small amount of material in the fractions of -600 μm to +212 μm they were combined into one size fraction sample.

The polished samples were first studied in reflected light using a Zeiss Axio Imager M2m optical microscope and were later coated with a layer of approximately 10 nm carbon for electron microscopy studies.

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Commented [V8]: Please add city name and abbreviation of state for this equipment. The electron microscopy measurements were performed using a field emission scanning electron microscope (FEG-SEM, JEOL JSM 7000F) equipped with an Oxford Instruments energy-dispersive X-ray spectrometer (EDS, X-Max 80). [The imaging and the EDS analyses were performed under routine conditions using 20 kV acceleration voltage and a beam current of 1 nA. The mineral amount was calculated based on the EDS analyses of the minerals and the analyzed chemical composition of the size fractions using the EMC method (Lamberg et al., 1997; Lund et al., 2013); this is included in the HSC Chemistry[®] 10 software (Metso Outotec, 2021).

Additionally, the main minerals were identified from two powder samples: original unwashed and -20 μ m size fraction samples, using a PANalytical Aeris X-ray diffractometer (XRD) with Co K α radiation at 40 kV and 15 mA, using step size of 0.0110 °2 θ and a step time of 48.1950 s.

2.5. Mineral liberation analysis

Liberation measurements were performed using FEG-SEM-EDS coupled with the Aztec Mineral liberation analysis software developed by Oxford Instruments. Mineral liberation measurements were carried out, focusing on sulfide minerals.

2.6. Grinding tests

Wet grinding tests were performed using a mild steel laboratory ball mill (215 mm height x 205 mm diameter) using 27 mm (3.3kg) and 19 mm (8.7 kg) balls with 65% w/w (weight by weight) solids ratio. Grinding times of 5, 10, 20, 40, 80, and 105 minutes were selected to create ore grindability data for the flotation experiments.

2.7. Flotation experiments

The Bor flotation experiments were carried out using a laboratory-type Outotec GTK Lab Cell flotation machine. The GTK flotation lab cell is equipped with an adjustable external water pump to maintain a constant pulp level. Froth is recovered using an automatic froth scraper system. The flotation and grindability experiments were conducted with locally sourced tap water. The pH of the slurry was measured as 3.5.

The Bor bulk flotation experiments were carried out at a 35% w/w solids ratio using 1 kg and 2 kg feed in 2- and 4-liter flotation cells, respectively. Moreover, 2 kg feeds were used for the cleaner flotation experiments to maintain a sufficient amount of material. Makeup water was added during the flotation experiments to provide a constant pulp level in the flotation cell. The airflow was set to a steady 3 L*min^-1 throughout the flotation experiment.

The air was fed in with a 15-20 second delay to build froth. Once the froth reached the discharge lip, the flotation time and froth scraping were recorded. Impeller rotational speeds of 1300 and 1500 rpm were used in the 2- and 4-liter flotation cells. Rougher flotation concentrates were collected at 2, 6, 14, 22, and 30 minutes to determine the flotation kinetics. The error of the flotation experiments was determined as 2.3% especially for the bulk stage. Flowsheets from the flotation experiments are shown in Figures 2, 3, and 4.

Sulfidation was applied in the bulk flotation experiments by adding sodium hydrosulfide (NaHS) stagewise. Additionally, the pulp potential (Eh) was monitored with a commercial Pt-Ag(AgCl) electrode during the flotation experiment. Five minutes of conditioning time was targeted for the sulfidation. A standard hydrogen electrode (SHE) was used for the pH measurement.

Collectors were added stagewise during the flotation experiments. Moreover, threeand ten-minute conditioning times were applied for the collectors and depressants, respectively. The specifications of the reagents used in the study are shown in Table 1.

RF- Rougher Flotation **RC**- Rougher Concentrate

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Figure 2. Flowsheet for kinetic rougher flotation RC1-5 refers to rougher concentrates 1 to 5.





Figure 3. Flowsheet for kinetic bulk cleaner flotation experiments.





Figure 4. Flowsheet for copper pyrite selective flotation experiments.

Table 1. Reagents used for the flotation experiments

Reagents	Formula	Туре	Commercial name
Sodium isobutyl xanthate	(CH3)2CHCH2OCSSNa(K)	Collector	SIBX
Sodium dialkyl dithiocarbamate	(C2H5)2NCS2Na	Collector	XD 5002
Sodium diisobutyl dithiophos-			
phinate	C8H18NaO2PS2	Collector	Solvay 3418A
Polypropylene glycol methyl ether	C3H6O	Frother	Dow froth 250
Sodium hydrosulfate	NaHS	Modifier	NaHS
Sodium silicate	Na2SiO5	Depressant	Zeopol 30
Sodium metabisulphite	Na ₂ S ₂ O ₅	Depressant	Merck
Dextrin	C18H32O16	Depressant	Beardow Adams
Polyacrylamide	(C3H5NO)n	Depressant	Solvay S7260

3. Results

3.1. Chemical and mineral composition

Based on the assays, the old Bor tailings sample contained 57.3% silica, 11.9% alumina, 7.7% iron, 7% sulfur, 0.24% copper, 0.41 ppm gold, and 2.2 ppm silver. Water and acid-soluble copper minerals carried almost 60% of the total copper within the sample. P1 refers to H2O dissolution, P2 to H2SO4 dissolution, P3 to KCN dissolution, and P4 to HNO3 + Br_2 dissolution. The detailed chemical analysis is shown in Table 2.

Table 2

Fe

С

s

SO4 2-

Magnetic

%

%

%

%

%

ICP OES

Combustion

Combustion

Ion chrom.

Satmagan

7.71

0.09

8.17

3.25

0.7

Element/			
oxide	••	Method	Analyzed bulk
SiO ₂	%	Uv-Vis.	57.3
Na2O	%	ICP OES	0.39
MgO	%	ICP OES	0.25
Al ₂ O ₃	%	ICP OES	11.88
P2O5	%	ICP OES	0.11
K2O	%	ICP OES	0.69
CaO	%	ICP OES	1.23
TiO ₂	%	ICP OES	0.42
Ba	%	ICP OFS	0.22
As	%	ICP OFS	0.014
Sr	%	ICP OFS	0.09
51 A.a.	/0	ICD MS	0.09
Ag	ррш	ICP MS	2.2
Au Cu Pl	ppm o/	D1	0.41
Cu Pi	% 0/	PI	0.111
Cu P2	%	P2	0.022
Cu P3	%	P3	0.053
Cu P4	%	P4	0.036
		Sum	0.222
Cu	%	ICP OES	0.241
Fe P1	%	P1	< 0.05
Fe P2	%	P2	0.702
Fe P3	%	P3	0.36
Fe P4	%	P4	6.64
		Sum	7.70

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The main minerals of the sample were quartz (40.7 wt%), dickite (17.85 wt%), and pyrite (13.3 wt%). The detailed mineralogy of the sample is presented in Table 3. The mineralogy of the old Bor tailings is complex due to the subsequent alteration of the primary sulfides and the occurrence of subsequent water- and acid-soluble copper minerals, like chalcanthite and different types of copper oxides.

Copper in the old Bor tailings was carried by secondary chalcanthite-type minerals (57.5%) and by relicts of primary chalcopyrite (16.6%), enargite (16.3%), and covellite (9.6%). In addition, over 95% of the total sulfur content was carried in the pyrite. The sulfide fraction consisted of pyrite (95%), and the rest, 5 %, included various copper sulfides and sulfates. The copper content of the sulfide fraction was 1.58%.

Based on the size-by-size liberation measurements, the copper minerals formed locked particles with gangue silicates and pyrite. In the coarser size fractions, copper minerals occurred as fine-grained inclusions in gangue silicates, and the appropriate liberation degree was reached in the finest -20µm size fraction.

The bulk liberation degree of the copper minerals was 57%, i.e., 57% of the copper minerals were 90% to 100% liberated. Overall, 85.5% of the pyrite occurred as liberated grains, and the rest formed locked particles with silicates (Figure 6).

Mineral (wt%)	Analyzed bulk		
Quartz	40.68		
Dickite	17.85		
Pyrite	13.31		
Muscovite	5.62		
Plagioclase	4.23		
Alunite	2.11		
Gypsum	2.02		
Slag (fayalite + glass)	1.26		
Amphibole	0.87		
Magnetite	0.70		
Svanbergite	0.68		
Rutile	0.40		
Barite	0.34		
Apatite	0.02		
Chalcopyrite	0.10		
Enargite	0.07		
Covellite (+bornite)	0.03		
Copper sulfates, oxides, carbonates	0.47		
Others	9.23		

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3.2. Flotation experiments

Altogether 24 flotation experiments were carried out at a medium alkaline pH of 9.75-9.85 to prevent copper dissolution into the water phase by adding Ca(OH)2 Moreover, the dissolved copper was precipitated by adjusting the pH to >8. In the base case, sodium isobutyl xanthate (SIBX), which has earlier been reported to be the most effective collector under medium alkaline conditions (Markovic et al., 2010), was the primary collector, and polypropylene glycol methyl ether (Dow froth 250) was the main frother. Based on previous studies, the reagent dosages were 140 g/t SIBX and 80 g/t Dow froth 250.

3.2.1. Effect of particle size on bulk flotation

To study the effect of different particle sizes on flotation experiments, the sample was ground for different grinding times to produce a particle size distribution (P₈₀) between 50 μ m and 13.4 μ m. The highest copper grade, 0.49% Cu, was achieved when the P₈₀ was 50 μ m. Although the highest copper recovery of 93% was obtained with a finer particle size having a P₈₀ of 26 μ m, the concentrate grade, 0.46% Cu, remained slightly lower. The lowest copper recovery (78%) and grade (0.25% Cu) for the bulk concentrate were obtained without grinding at 132 μ m (P₈₀). Decreasing the particle size of the feed improved both recovery and grade as the degree of liberation of sulfides increased (Figure 7).



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Figure 7. Effect of particle size distribution on flotation performance a) copper grade versus copper recovery and b) bulk copper recovery and grade by P₈₀.

3.2.2. Effect of different collectors on bulk flotation

Thiocarbamate and dithiophosphates are some of the most important collectors that enable robust flotation performance, especially for oxidized copper minerals like those found in the Bor tailings. Therefore, Solvay's 3418A (Sodium di-isobutyl-dithiophosphinate) and XD 5002 (alkyl thiocarbamate) were tested to achieve better flotation performance (Bulatovic, 2007).

According to the copper recoveries and grades, the overall flotation kinetics are almost identical for both collectors. The different types of collectors did not improve the flotation performance and selectivity (Figure 8).



Figure 8. Effect of different promoters on bulk flotation performance a) copper grade versus copper recovery and b) copper recovery against mass pull on the concentrate.

3.2.3. Effect of NaHS dosage

In the presence of oxidized minerals, sulfurizing agents such as sodium sulfide (Na₂S) and sodium hydrosulfide (Na_HS) are among the most common modifiers that increase the recovery of oxidized minerals (Yin et al., 2019). The mineral surface of heavily oxidized sulfide minerals can be transformed into a hydrophobic surface by flotation modifiers such as Na_HS. Therefore, Na_HS makes such oxidized minerals more floatable in the presence of xanthate or thiol collectors (Leja, 2007).

The copper phase of the Bor tailings consists of oxide-type minerals. The pulp potential needs to be precisely adjusted to avoid excessive dosages using the sulfidation method.

The flotation kinetic rate was increased by reducing the pulp potential using NaHS, but it also had a negative effect on the copper grades and recoveries. Additionally, when decreasing the pulp potential to below -150 mV (Pt-Ag/AgCl) with NaHS, the flotation selectivity for gangue minerals decreased (Figure 6). Therefore, test number 3 with the reagent regimes 140 g/t SIBX and 80 g/t was determined as the baseline for the bulk stage.





Figure 9. Effect of NaHS dosage on bulk flotation performance a) copper grade versus copper recovery and b) copper recovery against mass pull.

3.2.4. Effect of Na2SiO3 on cleaner stages

Test number 3 bulk flotation experiment conditions were used to produce material for the cleaner kinetic flotation experiments. All reagents were added stagewise in the cleaner flotation experiments to avoid excessive dosages and obtain proper kinetic trends. According to the results, an addition of 250 g/t Na₂SiO₃ (Zeopol 30) did not improve the copper grade in the first cleaner concentrate. Based on the results, adding Na₂SiO₃ to the first cleaner stage had no major effect; therefore, in test number 9, cleaner flotation experiment conditions were selected (Figure 10).



 $\label{eq:Figure 10. Na2SiO3 on cleaner flotation performance a) copper recovery versus copper grade and b) copper recovery against mass pull.$

3.2.5. Copper-pyrite selective flotation

After determining the rougher and scavenger reagent regime and the first cleaner flotation stage, bulk concentrate was produced at 1% copper grade with 85% recovery. In the copper-pyrite selective flotation experiments, the aim was to depress pyrite in the presence of dextrin, polyacrylamide, and sodium metabisulfite. The depression mechanism of dextrin and polyacrylamide can be induced through interaction with surface ferric hydroxide (López-Valdivieso et al., 2018; Yang et al., 2021). On the other hand, in the presence of Na₂S₂O₅, sulfide ions can be easily transformed to SO_{5-} and these

Commented [V23]: Do you mean on peroxomonosulfate ion, SO₇²⁻? complexes can convert the copper sulfides on pyrite surfaces to copper oxides. Due to the oxidation of pyrite surfaces, xanthate collectors cannot attach to the pyrite surfaces, so pyrite is depressed (Anand et al., 2012; Chen et al., 2013) and copper concentrate can be separated from the pyrite successfully.

Copper-pyrite selective flotation experiments on the second bulk cleaner concentrate were carried out by increasing the pH to 12. Moreover, 1250 g/t polyacrylamide (Solvay 7261A), 1250 g/t potato-based Dextrin, and 500 g/t Na₂S₂O₃ were used to depress the pyrite and float the copper minerals (Figure 9).

The copper-pyrite selective flotation experiment results showed that different depressants did not depress pyrite in the copper circuit. However, 1250 g/t of dextrin decreased the sulfur recovery to below 40% in copper concentrate. A decent quality (Cu>15%) of copper concentrate was not achieved even though dextrin depressed pyrite in the copper-pyrite selective circuit (Figure 11). Therefore, the studied flowsheet focused on bulk flotation with the two-cleaner stage.



Figure 11. Effect of depressants on selective copper-pyrite flotation a) sulfur grade versus sulfur recovery and b) sulfur recovery versus copper recovery.

3.2.6. Flowsheet simulation

A simulation was carried out using HSC Chemistry ® 10 software on the preliminary flowsheet that was developed, including rougher, scavenger, and two bulk cleaner stages to determine the final copper and sulfur recoveries in a continuous process (Lamberg, 1997).

The HSC Sim Model Fit® tool was used for mass balance and middling stream calculation. In addition, the Klimpel flotation model for batch flotation was used to calculate kinetic parameters such as the infinitive recovery and the flotation rate constant. Mass balances were calculated, including grades and recoveries for each mineral in each stream. In the Klimpel model (Equation 1), R_{max} shows the highest possible recovery for each mineral, and the k of rectangular distribution reveals how fast each mineral is floated (Metso Outotec, 2021):

$$R = R_{max} \left\{ 1 - \frac{1}{kt} [1 - e^{-kt}] \right\}$$
(1)

where t is the cumulative residence time, R $_{max}$ is the infinitive recovery, K $_{max}$ is the flotation rate constant, and R $_{max} \leq 1.$

HSC Sim Model Fit was used to draw kinetic curves for each mineral and define k_{max} and R_{max} . These values were then used for flotation modeling. A tentative 100 t/h plant capacity was selected and simulated using the rectangular distribution method for batch flotation time with R_{max} and k_{max} constants.

The simulation model calculated the residence time based on the volumetric pulp feed to the flotation bank with a fixed gas hold-up of 10%. The simulation model consisted of a 2x100 m³ Metso Outotec[™] Tank Cell rougher and a 2x100 m³ Metso Outotec[™] Tank Cell scavenger stage. Additionally, the first cleaner and second cleaner cell configurations were determined as a 2x100 m³ Metso Outotec[™] Tank Cell circuit and a 1x10 m³ Tank Cell, respectively. The solids content of the rougher and cleaner stages was adjusted to 35% w/w and 25% w/w, respectively.

Identifying the appropriate scale-up factor of geometallurgically challenging tailings is always problematic due to the slower flotation kinetics and low-grade upgrade. Moreover, determination of scale-up factor is always a complex process which cannot be completely solved and usually results in the overscaling of laboratory data values (Mesa et al., 2019). Therefore, the scale-up factor was selected as 2.5 for this specific sample, taken from the Metso Outotec database system.

According to the simulation results, a pyrite concentrate containing 48.73% S can be produced at 87% recovery. The pyrite concentrate contains 1.59% Cu at 82% recovery (Figure 12).



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Figure 12. Simulated flowsheet with copper and sulfur grades and recoveries.

4. Conclusion

This study produced an efficient recovery method for pyrite and copper minerals by using bulk flotation. According to bench-scale experiments and continuous simulation results, the following conclusions can be made:

Decreasing the particle size of the tailings was the only factor that had a significant positive effect on flotation results, due to the liberated and cleaned sulfides and their surfaces. The inactivity of various collectors can be explained by the oxidation of copper and subsequent activation of pyrite and gangue mineral surfaces by Cu^{2+,} which destroyed the selectivity of the sulfide bulk flotation stage (Anand et al., 2012; Chen et al., 2013; Ejtemaei and Nguyen, 2017).

The particle size (P_{80}) of 26 μ m was selected for optimal pyrite concentrate flotation with high recoveries for copper and sulfur. Due to the inefficacy of different reagents, non-selective sulfide flotation, and the results of previous experiments (Markovic et al., 2010, Han et al., 2014), further optimization experiments were considered unnecessary.

Decreasing the pulp potential to below -150 mV by NaHS negatively affected the copper grades and recoveries in the bulk stage; therefore, sulfidation is not recommended for the bulk flotation stage.

The addition of sodium silicate in the first cleaner stage experiments showed that it had no depressing effect on the gangue minerals.

Sequential flotation of copper and pyrite minerals was investigated in the study; however, due to the intense activation of the pyrite, no selectivity was achieved, and for that reason, medium alkaline bulk conditions were selected for the remaining flotation experiments.

The water-soluble copper minerals were successfully recovered to the bulk concentrate by adjusting the pulp conditions to a medium alkaline pH.

Based on the experiments conducted, copper and gold can be recovered effectively into a pyrite concentrate. Ultimately, 85% of the sulfur and 80% of the copper were recovered into the concentrate using two bulk cleaner stages. The recoveries for gold and silver were 49% and 84%, respectively.

Reprocessing old mine tailings can be both an economical and environmentally friendly solution. The flowsheet, reagent, and flotation conditions created by this study can be a useful solution, especially for problematic sulfide mining tailings that cause environmental pollution. The pyritic concentrate produced in this study can be used in flash smelting as a source of energy and valuable metals (Bryk et al., 1958; Kojo and Storch, 2006), and the generated SO₂ can be used to produce sulfuric acid (Alexander et al., 2021). In the long term, removing pyrite from the tailings and increasing the slurry pH up to 9.80 can reduce the environmental impact caused by acid mine drainage.

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Subject: RE: Request for Review: AMMS-23-RA-898 From: Aspects in Mining & Mineral Science <mining@crimsonpublication.org> Date: 12/21/2023, 1:41 PM To: Vesna Marjanovic <vesna.marjanovic@irmbor.co.rs>

Dear Dr. Marjanovic Vesna,

Thank you for your reply.

We are thankful for your valuable review comments. Soon, we will reach you with another new manuscript.

Hope you understand us.

Have a nice day!

Sincerely

Alissa Lively

From: Vesna Marjanovic <vesna.marjanovic@irmbor.co.rs>
Sent: Thursday, December 21, 2023 12:54 PM
To: Aspects in Mining & Mineral Science <mining@crimsonpublication.org>
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Dear Alissa,

the revised manuscript and review for are in the attachment. We will be in touch if anything else is required.

Sincerely

dr Vesna Marjanovic