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INVESTIGATION OF THE FLOTATION POTENTIAL OF SULFIDE-OXIDE ORE FROM THE VELIKI MAJDAN DEPOSIT

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Abstract

This paper presents the results of an investigation into the applicability of single-stage flotation for processing complex lead sulfide–oxide ore from ore body RT255-37 at the Veliki Majdan deposit. The study aimed to optimize key technological parameters of the process, including grinding, pulp conditioning, flotation reagent dosage, and flotation regime. Granulometric analysis (particle size distribution) indicated that the ore is relatively hard. The optimal grinding conditions were defined as a grinding time of 9 minutes under a typical solid content in the pulp, which provided a suitable particle size distribution for flotation.

Special attention was given to the application of sulfidization, which creates favorable conditions for the flotation of lead oxide minerals. This step is essential for achieving high recovery. The required sodium sulfide dosage was relatively high, amounting to 3700 g/t. The reagent regime was further enhanced by increasing the dosage of the collector KEX and introducing a more efficient frother, Florrea Frother 520. Flotation kinetics were evaluated under conditions comparable to those in industrial operations.

Under laboratory conditions, recoveries of 93.37 % for lead sulfide minerals and 86.56% for oxide minerals were achieved, with a final concentrate grade of 68.67 % Pb. Locked cycle flotation test confirmed the stability and reliability of the optimized flotation regime, resulting in a concentrate with 69.55 % Pb and an overall lead recovery of 88.41 %. The results demonstrate the technical and technological feasibility of applying the optimized flotation process to the treatment of such complex lead ores.

Keywords: *flotation concentration, sulfidization, sulfide–oxide ore*

1. INTRODUCTION

Flotation concentration is one of the most efficient and widely used methods for the selective separation of mineral components based on their surface properties. The key property exploited in flotation is the wettability, or lack thereof, of specific minerals [1]. In addition to these differences in physical properties, flotation reagents play a crucial role in enhancing the separation process.

It is well known that the floatability of lead oxide minerals and oxidized sulfide minerals can be improved by surface sulfidization. [3] In ores that contain oxidized lead minerals, such as cerussite ($PbCO_3$), prior chemical activation is required, most commonly through sulfidization using sodium sulfide (Na_2S), to enable effective attachment to collectors such as xanthates.

The aim of this study is to investigate, through laboratory-scale testing, the efficiency of single-stage flotation applied to a complex sulfide–oxide ore from ore body RT255-37. The study focuses on determining the optimal parameters for grinding, pulp conditioning, reagent dosage, and flotation conditions. In addition, the quality of the resulting concentrate and the overall metal recovery under locked cycle flotation test conditions were evaluated.

2. EXPERIMENTAL

2.1. General Information about the Deposit and Mine

The Veliki Majdan lead–zinc deposit is classified as a hydrothermal–metasomatic type of mineralization, developed within a carbonate host environment during the Tertiary magmatic–volcanic activity of the Boranja Complex. The ore bodies are morphologically complex and spatially associated with the contacts between marbled limestones, quartz latite dikes, and terrigenous sediments. Among the oxidized lead minerals, cerussite ($PbCO_3$) and, to a lesser extent, hydrocerussite have been identified, while hemimorphite and likely smithsonite ($ZnCO_3$) are dominant among the zinc oxide minerals. [2]

2.2. Materials and Methods

For the purpose of laboratory testing, a composite sample with a total mass of 20 kg was provided by the geological service of the mine, originating from ore body RT255-37. The sample preparation included the following procedures: crushing and homogenization, density determination using the pycnometer method (3.11 g/cm^3), wet sieving to determine particle size distribution, chemical analysis of Pb and Zn content, and phase-specific distribution of these elements.

Chemical analysis indicated an elevated content of non-sulfide Pb and Zn minerals. Given the known technological challenges associated with the flotation of non-sulfide Zn minerals, they were excluded from experimental flotation testing.

Due to the pronounced presence of oxidized Pb phases, sulfidization using sodium sulfide (Na_2S) was applied in the flotation process to enhance mineral hydrophobicity and improve collector adsorption.

Following sample preparation, the subsequent procedures were conducted: determination of optimal grinding time, optimization of reagent dosages, flotation kinetics analysis, and a final locked-cycle flotation test. All flotation products were processed using standardized laboratory protocols and subjected to chemical analysis to determine metal content and recovery.

3. RESULTS AND DISCUSSION

This section presents the obtained results along with the corresponding comments and interpretations.

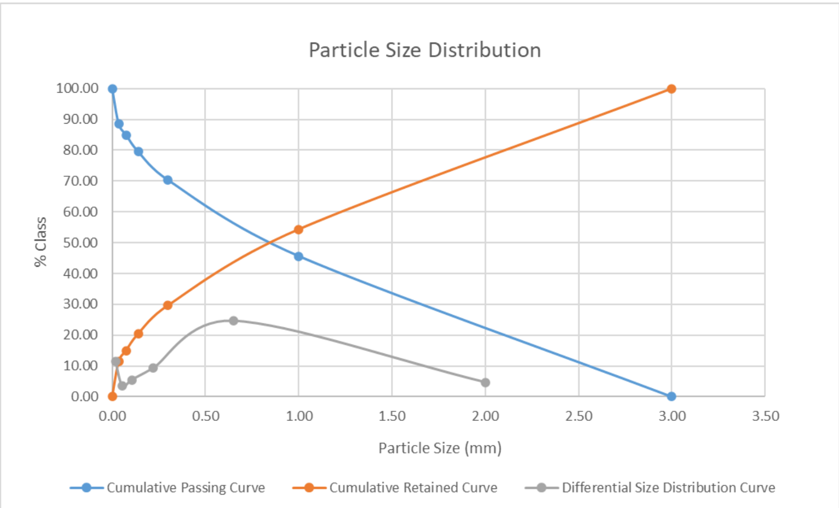

Figure 2. Particle size distribution of the final crushed sample

From the particle size distribution diagram, the upper grain size limit (d_{95}) was determined to be 2.55 mm. The most represented size fraction is -1 +0.3 mm, with a mass fraction of 45.66 %, while the least represented fraction is -0.074 +0.035 mm, with a mass fraction of 3.49 %. Based on these results, the ore can be considered relatively hard to grind. The results of the granulometric analysis are presented in Figure 1.

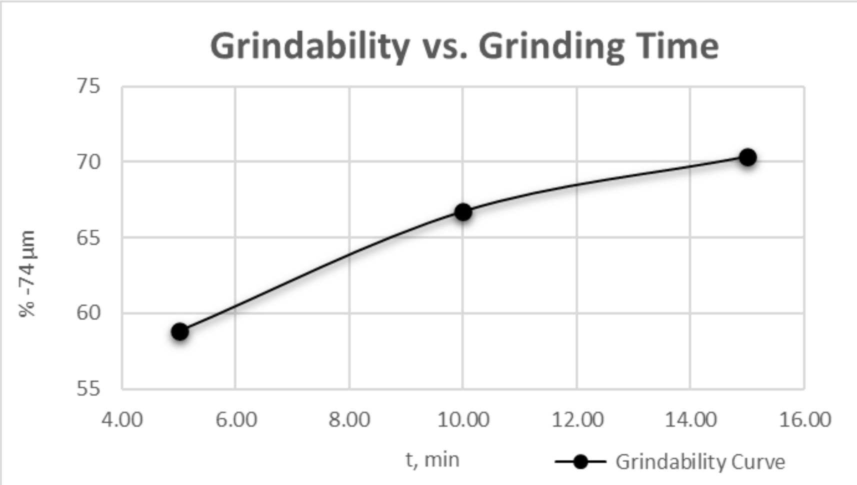

Figure 3. Diagram of grindability as a function of grinding time

Based on the analysis of Figure 2, the optimal grinding time of 9 minutes was selected to achieve approximately 65% of the -74 µm particle size fraction.

The reagent regime adopted for flotation included a sodium sulfide dosage of 3700 g/t to ensure complete sulfidization. The dosage of the potassium ethyl xanthate (KEX) collector was increased to 230 g/t. The consumption of depressants, specifically sodium cyanide and zinc sulfate, remained unchanged compared to the existing plant regime. The frother Florrea Frother 520 was used at a dosage of 120 g/t.

The durations of the individual flotation stages were as follows: rougher flotation – 20 minutes, scavenger flotation – 23 minutes, first cleaning – 10 minutes, and the second and third cleaning stages – 5 minutes each.

Table 5. Metal Balance for the Final Experiment

	M, g	M, %	Pb-tot, %	R Pb-tot, %		Pb-sul ,%	R Pb-sul, %		Pb-ox, %	R Pb-ox, %	
FEED	906.73	100.00	5.36	100.00	100.00	2.11	100.00	100.00	3.25	100.00	100.00
Final Conc	21.23	2.34	**68.67**	30.01		24.50	27.17		44.17	31.85	
C1 MID	16.37	1.81	29.02	9.78		9.87	8.44		19.15	10.65	
C2 MID	6.11	0.67	40.94	5.15	**89.25**	12.68	4.05	**93.37**	28.26	5.86	**86.56**
C3 MID	4.74	0.52	55.55	5.42		17.85	4.42		37.70	6.07	
Scav Conc	111.81	12.33	16.90	38.89		8.44	49.29		8.46	32.13	
Tailings	746.47	82.33	0.70	10.75	10.75	0.17	6.63	6.63	0.53	13.44	13.44

The analysis of the results indicates that a high lead recovery of 89.25 % was achieved in the primary concentrate, which is highly favorable considering the complexity of the sulfide-oxide ore. Recovery of galena was also successful despite sodium sulfide typically acting as a depressant for this mineral. Additionally, the recovery of oxidized lead minerals reached 86.56 %. The quality of the final concentrate is satisfactory, with a lead grade of 68.67 % Pb after the third cleaning stage. Throughout the process, a high pH range of 10.9–11.3 was maintained, which did not hinder the efficient flotation of galena. The summarized flotation test results are presented in Table 1. Following the laboratory tests, locked cycle flotation test was performed to better simulate industrial conditions, with the results discussed in the subsequent section of this paper.

Table 6. Metal Balance of the Locked Cycle Flotation Test

	M, g	M, %	Pb-tot, %	R Pb-tot, %		Pb-sul, %	R Pb-sul, %		Pb-ox, %	R Pb-ox, %	
FEED	2706.08	100.00%	5.52	100.00%	100.00%	2.82	100.00%	100.00%	2.70	100.00%	100.00%
Final Conc	107.02	3.95%	**69.55**	49.82%		42.56	59.64%		26.99	39.55%	
C1 MID	205.57	7.60%	19.97	27.48%		10.66	28.69%		9.31	26.20%	
C2 MID	28.09	1.04%	13.2	2.48%	**88.41%**	4.65	1.71%	**94.69%**	8.55	3.29%	**81.85%**
C3 MID	31.43	1.16%	28.08	5.91%		7.19	2.96%		20.89	8.99%	
Scav Conc	8.47	0.31%	48.17	2.73%		15.24	1.69%		32.93	3.82%	
Tailings	2325.50	85.94%	0.74	11.59%	11.59%	0.17	5.31%	5.31%	0.57	18.15%	18.15%

The results of the locked cycle flotation test confirmed the previous findings and demonstrated excellent process performance. The concentrate quality was further improved, reaching a lead grade of 69.55 % Pb, while lead recovery in the primary concentrate slightly decreased to 88.41 %. A high recovery of the sulfide lead mineral galena was also confirmed, amounting to 94.69 %, despite the presence of sodium sulfide acting as a depressant. The recovery of non-sulfide lead minerals remained relatively high at 81.58 %, further confirming the effectiveness of the applied sulfidization process. A detailed overview of the locked-cycle flotation results is presented in Table 2**.**

4. CONCLUISON

Based on the conducted research, it was determined that for efficient grinding of ores containing hard particles, a grinding time of 9 minutes under typical pulp density conditions enables achieving approximately 65% of the -74 µm size fraction in the ground material. For complete sulfidization of the oxide minerals, a sodium sulfide (Na_2S) dosage of 3700 g/t is required, while the collector potassium ethyl xanthate (KEX) was optimized to 230 g/t. The quantities of depressants (sodium cyanide and zinc sulfate) remained unchanged compared to the existing operational regime. Replacing the D-250 frother with Florrea Frother 520 at a dosage of 120 g/t contributed to an improvement in concentrate quality. The flotation kinetics were established with the following stage durations: rougher flotation for 20 minutes, scavenger flotation for 23 minutes, first cleaning flotation for 10 minutes, and second and third cleaning flotations for 5 minutes each.

Under laboratory conditions, a galena recovery of 93.37 % and lead oxide mineral recovery of 86.56 % were achieved, with a lead content in the concentrate of 68.67 % Pb after the third cleaning stage. It is important to emphasize that the use of sodium sulfide (Na_2S), although known as a depressant for galena, did not negatively affect the recovery of this mineral. The locked cycle flotation test confirmed these results, achieving an even higher lead content in the concentrate of 69.55 % Pb and a lead recovery of 88.41 %, including 94.69 % recovery of sulfide minerals and 81.58 % recovery of nonsulfide minerals. These results confirm the efficiency of the optimized technological regime and its applicability under industrial conditions.

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