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Optimization of differential and cumulative flotation strategies for lead and silver recovery from mixed oxide-sulfide ores

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ABSTRACT

Differential and cumulative flotation techniques were used to concentrate two representative samples, A and B, from the Khanjar lead-silver mine (Semnan Province, Iran). These samples had lead grades of 0.93% and 0.95% and silver contents of 16.8 and 16 g/t, respectively. According to the analysis of the lead-containing minerals in the samples, only a small fraction of the total lead occurred as galena, with 0.73% as non-sulfide forms. According to the flotation test results, type A and type B ore behave differently during the flotation process, with type B ore exhibiting superior flotation performance. The lead recovery from ores type A and B was 68.41% and 84.61%, respectively, in the combined rougher and scavenger stages by adding 50 g/t of collector Z11 in the flotation stage of sulfide minerals and 500 g/t of sodium sulfide as a surface activator in the flotation stage of non-sulfide minerals along with 50 g/t of frother MIBC at pH 8.5. The grade and recovery of lead and silver increased in the experiments to improve the flotation efficiency of lead and silver from ores of type A by adding secondary collectors 3418A and M46, but the sodium silicate dispersant had no discernible effect on the recovery. The appropriate flotation technique for mixed oxidized-sulfide lead and silver ores can be determined using the research's findings..

Keywords: *Flotation, Lead-Silver, Mixed oxide-Sulfide, Differential, Cumulative.*

1. Introduction

With their distinct physical characteristics and wide range of uses, global consumption of lead and zinc in 2020-2021 ranks fifth after iron, aluminum, copper, and manganese. Due to their expanding use in a variety of industries, such as lead-acid batteries, pigments, radiation shielding, and ammunition, the demand for lead-containing ores has grown recently [1]. Galena (PbS), cerussite (PbCO3), and anglesite (PbSO4) are the three most significant lead minerals [2], with galena accounting for the majority of lead production [3]. An alternate source for the mineral galena is cerussite, one of the most significant lead oxide minerals [1, 4]. Three types of lead ores can be found in nature: mixed sulfide-oxide ores, non-sulfide ores, and galena sulfide ore. Significant deposits of mixed sulfide-oxide ores can be found all over the world, but processing them is extremely difficult because of the complexity of mineralogy [5].

The flotation process typically works well for sulfide minerals. However, because of their greater solubility and greater surface hydrophilicity, non-sulfide minerals need to be activated in order to float during the flotation process [1, 4, 6]. The sulfidation process enhances the mineral surface hydrophobicity, electronegativity, and collector adsorption. Surface sulfidation, roasting sulfidation, mechanochemical sulfidation, and hydrothermal sulfidation are among the sulfidation techniques [1]. Sulfidizing the surface of non-sulfide minerals with sodium sulfide is one of the most popular techniques for activating these minerals [1, 4, 7]. The solubility product of cerussite is about 7.4×10^{-14}, indicating that cerussite is a semi-soluble salt mineral. For this reason, some lead ions always enter the solution. Xanthate ions prefer to react with the lead ions in the solution. Therefore, sulfidation of the cerussite surface and the formation of a thin film of PbS on the cerussite surface are necessary [8]. Actually, equation 1 states that when reagents containing sulfide ions dissolve, they react with lead ions to form PbS, which then deposits on the surface of cerussite (equation 2) [9]. After sulfidation, the cerussite surface is converted to PbS, and the core is $PbCO_3$ [10].

$$Na_2S.9H_2O = 2Na^+ + S^{2-} + 9H_2O \tag{1}$$

$$PbCO_3 + S^{2-} = PbS + CO_3^{2-} \tag{2}$$

Using the sulfidation-flotation method, Hajati and Soltani (2023) found that the maximum grade and recovery of lead in the rougher-scavenger stages were 8.61% and 54.8%, respectively, based on their study of the mixed oxidized-sulfide lead ore of Zarigan (Yazd Province, Iran) containing 6.12% lead. They used 7 kg/t of sodium sulfide after optimizing the amount and number of steps for adding sodium sulfide. Their low flotation efficiency was caused by the variety of non-sulfide lead minerals, which included notable amounts of plumbogummite and plumbojarosite lead minerals [5]. Altınışık et al. (2022) also conducted research on lead-zinc complex ores and concluded that mineralogical characteristics had a significant effect on the flotation behavior of lead and zinc minerals [11]. By using sulfidation-flotation on an oxidized ore that contained 2.96% lead and 177 g/t of silver, Zhang and Chen (2021) were able to recover 65.16% and 87.81% of the lead and silver, respectively, by using 10 kg/t of sodium sulfide. Additionally, they concluded that the collector di-isobutyl dithiophosphinate (Aerophine 3418A) has the best selectivity for silver minerals [12]. By studying an oxidized ore that contained 17.09% zinc and 4.55% lead, Lan et al. (2013) optimized the amount and stage of sodium sulfide addition, ultimately

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using 6 kg/t of sodium sulfide [13]. By a combination of 450 g/t of potassium amyl xanthate (PAX), 150 g/t of sodium diethyl dithiocarbamate, and 7 kg/t of sodium sulfide, Kamran Haghighi et al. (2024) improved the grade and recovery of lead and silver. They yielded a maximum recovery of 52.6% for silver [14]. The results of micro flotation tests by Feng et al. (2016) on a lead and zinc ore showed that if the amount of sodium sulfide added to the pulp is either less or more than the optimal level, it has a negative effect on flotation performance. That is, with an increase in the amount of sodium sulfide, its performance changes and it acts as a depressant; also, with a low amount of it, it has no effect on flotation performance [4].

In this study, the flotation conditions of the mixed oxide-sulfide lead-silver samples from Khanjar mine (Semnan Province, Iran) were investigated in terms of the proper point for adding sodium sulfide and the secondary collector to increase lead and silver recovery, taking into account the effect of sodium sulfide in the flotation of sulfide and non-sulfide lead and silver minerals. The innovation of the present research is to compare the behavior of two different types of samples from a mixed oxide-sulfide ore and to investigate the performance of differential and cumulative flotation methods for these two types of samples. Finally, experiments have been conducted to investigate the feasibility of improving the flotation efficiency of the sample with poorer flotation performance.

2. Materials and methods

2.1. Sample preparation

The A and B samples used in this study were initially 10–15 cm in size and came from the Khanjar lead and silver mine in Semnan Province, Iran. Using jaw and roller crushers in a closed circuit with a 3.34 mm control screen, the samples were first crushed. To find the ideal grinding time, the crushed samples were separated into 2-kilogram packages using a Jones riffle. Representative samples were prepared for use in mineralogical studies, such as optical microscopy and XRD.

Based on the physical characteristics, color, and surface properties of samples, they were divided into two types, A and B (Figure 1). The purpose of this first separation was to examine how flotation behavior is affected by mineralogical changes. 2 kg samples of A and B ore types were ground in a rod mill for 10, 20, and 27 minutes in order to find the ideal grinding time. The d_{80} of sample A after 10, 20, and 27 minutes of grinding was 78, 48, and 43 µm, respectively. For sample B, the d_{80} was 115, 63, and 50 µm under similar conditions. Given that the flotation plant located within the mine operates with a d_{80} of about 50 µm (with a mixture of types A and B), the 27-minute grinding sample was used for both samples A and B in the flotation tests. Previous studies (at the time of the plant's construction) have shown that more than 80% of the galena and cerussite minerals are free in sizes finer than 63 µm.

2.2. Sample characterization

Thin section studies (transmitted light) and polished section studies (reflected light) were carried out using an optical microscope and XRD analysis to accurately investigate the mineralogical composition of the target and gangue minerals. The AF01 technique of the Zarazma company (Tehran, Iran) was employed for the chemical composition analysis of the samples, which identified the primary oxides found in the samples. Atomic absorption spectroscopy (AAS) was used to analyze the lead and silver elements found in the concentrates and tailings.

2.3. Flotation experiments

A series of laboratory-scale flotation experiments were planned and carried out in order to recover lead and silver from samples A and B. Flotation tests with 28.5% solids were carried out in order to optimize process conditions, such as pH (the sample's natural pH of 8.5 and approximately pH 10), the point of sodium sulfide addition (for differential or cumulative flotation of sulfide and oxide minerals), the effect of using sodium silicate dispersant, and the type of secondary collector. A 2.5-liter laboratory flotation cell with an agitation speed of about 1000 RPM was used for the experiments. In certain experiments, 25 g/t of laboratory-grade Aerophine 3418A (sodium diisobutyl dithiophosphinate from Cytec Industries) and M46 (a trade name for a certain Chinese collector) were used as secondary collectors in addition to 50 g/t of MIBC frother. The primary collector was a sodium isopropyl xanthate collector (marketed as Z11). 500 g/t of sodium sulfide ($Na_2S.9H_2O$, laboratory grade from Sigma) was used to sulfidize the surface of lead and silver bearing minerals. The grade and recovery of lead and silver were ascertained by weighing and analyzing the resultant concentrates and tailings.

Figure 1. An illustration of the locations of ore types A and B in Khanjar's mixed sulphide-oxide ore.

To investigate the repeatability of the results and also to determine the amount of error in the analysis of Pb and Ag and recovery calculations for both ore types A and B (statistical tests), the test was repeated three times under differential flotation conditions with 50 g/t Z11 collector and 50 g/t MIBC frother, and the grade and recovery errors of the flotation tests were calculated. It was assumed that this error was the same for all flotation tests.

3. Results and Discussion

3.1. Sample Characterization

3.1.1. Chemical composition of the samples

Table 1 displays the findings from the elemental analysis of samples A and B. While Type A has greater concentrations of CaO, Fe_2O_3, and LOI, Type B has more silica. This indicates that type A contains more carbonate and iron oxide minerals than type B, and the impact of this should be investigated in flotation tests.

Table 1 shows that samples A and B have respective concentrations of 16.8 and 16 g/t of silver and 0.93 and 0.95% lead. Non-sulfide minerals, primarily cerussite, contain 0.73% and 0.72% of the lead found in samples A and B, respectively. Consequently, surface sulfidization needs to be done during the flotation process.

3.2. Mineralogical composition of the samples

The results of the XRD analysis of the samples are shown in Table 2. Calcite is the primary phase found in sample A, while quartz and calcite are the primary phases found in sample B. Table 1 shows that samples A and B contain 17.97% and 24.2% of SiO_2, respectively. In both samples, cerussite and galena are significant lead minerals.

Microscopic studies of the ore type A

Type A ore is related to a carbonate breccia (Figure 2a), where the matrix of some sections consists of fine crystallized carbonate particles (microsparite). Large dolomite crystals can be found in some of the pieces (Figure 2c). Iron oxide veins frequently penetrate the rock, filling voids between fragmented sections (Figure 2d). Quartz can be found in vein-like formations, as aggregates of multiple minerals within cavities, or as coarse, dispersed crystals. Quartz crystals can occasionally measure

Table 1. Chemical composition of the A and B type samples.

Component	SiO_2(%)	CaO(%)	Fe_2O_3(%)	Al_2O_3(%)	Pb(%)	BaO(%)	K_2O(%)	MgO(%)	MnO(%)	LOI(%)	Ag (g/t)
Type A	17.97	42.16	3.04	1.08	0.93	0.27	0.32	0.34	0.14	33.28	16.8
Type B	24.20	39.72	1.24	1.15	0.95	0.21	0.35	0.42	0.14	31.09	16

Table 2. Results of the XRD analysis of A and B types of the ore.

Ore Type	A	B
Major phases	Calcite	Calcite, Quartz
Minor phases	Quartz, Cerussite, Galena	Cerussite, Galena
Trace phases	Barite	Barite

up to 2 mm in size. Large calcite crystals have developed in some areas (Figures 2b & c). In some cases, the surface of calcite particles displays sporadic iron oxide particles. On the surface of some siliceous rock sections, a collection of clay minerals and iron oxides is visible. The rock exhibits comparatively large cavities filled with clay minerals, measuring between 1 and 4 mm. This sample contains a small amount of metallic minerals. There are very tiny goethite, pyrite, and chalcopyrite particles scattered throughout the rock. There are two types of galena formation in the rock: vein-like and disseminated, with vein-like mineralization predominating. Galena has frequently changed from the margin to cerussite (Figure 2e). Cerussite and goethite are frequently observed together. Goethite contains traces of the primary mineral pyrite.

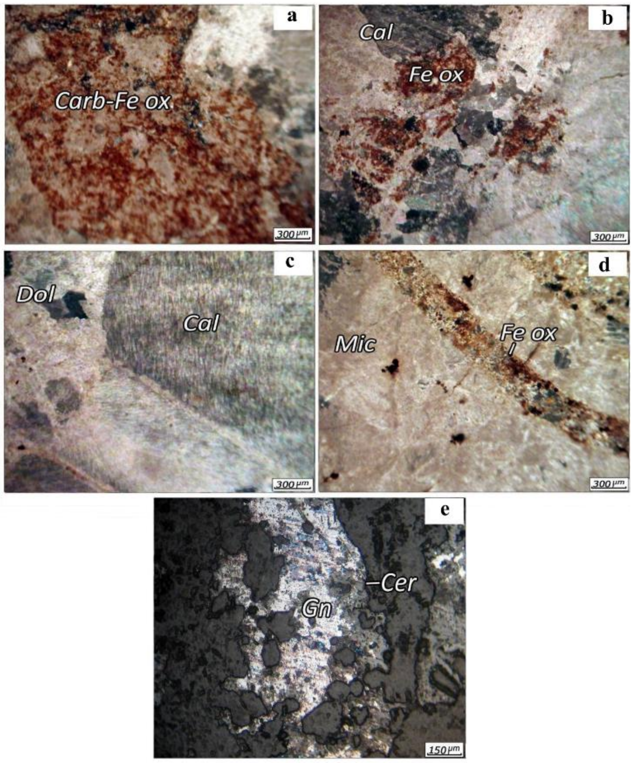

Figure 2. a) Accumulation of large amounts of iron oxides (Fe ox) in the rock, b) A carbonate vein cutting through the matrix composed of calcite (Cal) and iron oxide fragments, c) Two pieces of calcite and dolomite (Dol), d) A micritic mineral (Mic) with an iron oxide vein, and e) A galena (Gn) vein that has transformed from the margin to cerussite (Cer).

Microscopic studies of the ore type B

Quartz crystals make up the majority of the siliceous veins found in type B ore (Figure 3a). The rock contains vein-like, granular, and dispersed carbonate minerals. Occasionally, carbonate minerals show up as tiny inclusions in quartz. In certain sample sections, iron oxides were seen as disseminated crystals, most likely the result of pyrite oxidation. Certain portions of sample B also exhibit a predominance of microsparite and carbonate mud (micrite) (Figure 3b). Iron oxides and other opaque minerals are found in the rock as vein-like, dispersed grains (Figure 3b). The rock matrix is sliced through by carbonate veins, which can be as thick as 1-2 mm (Figure 3c & d). The primary metallic mineral that makes up the section is galena. The scattered galena crystals range in size from 50 µm to roughly 1 mm. In certain areas of the rock, galena has changed into cerussite (Figure 3e). Occasionally, the rock also contains fully cerussite pieces. Covellite and digenite, two secondary copper minerals with low abundance, were seen as disseminated rock grains (Figure 3e). Pyrite was found in inclusions with galena and in trace amounts. Additionally, goethite is found in trace amounts in type B ore.

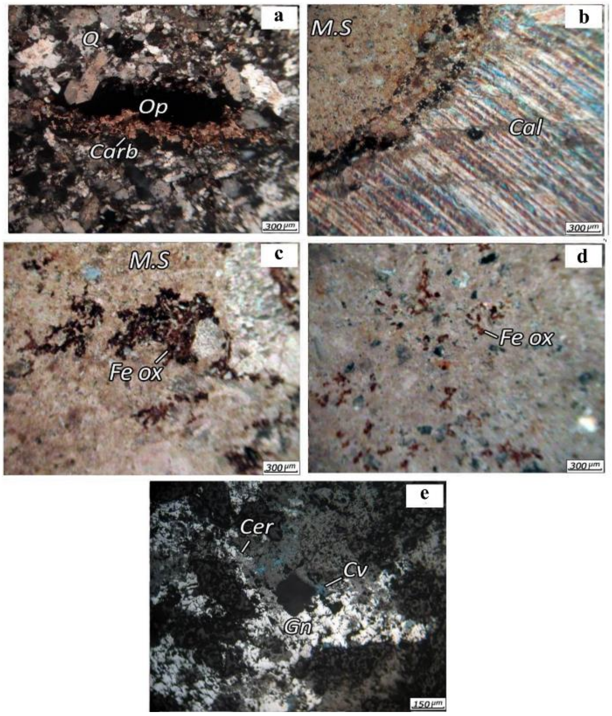

Figure 3. a) Fine quartz (Q) and carbonate mineral (Carb) particles in the rock; b) a piece of coarse calcite (Cal) and microsparite (M.S) within it, c and d) a buildup of scattered iron oxides (Fe ox) in the rock; and e) galena (Gn) that is partially altered to cerussite with blue-colored covellite (Cv) inclusions within cerussite (Cer). Op stands for opaque minerals that are black-colored in transmitted light sections.

3.3. Flotation process optimization

3.3.1. Comparison of differential and cumulative flotation methods

Flotation of type A ore

The performance of two techniques, differential and cumulative flotation, was compared in the flotation of a mixture of lead and silver oxide/sulfide minerals, taking into account the characteristics of the lead minerals found in ore types A and B. Figure 4a displays the chart for the differential flotation method, while Figure 4b displays the chart for the cumulative flotation method. The thiol collector Z11 is used in the first stage of the differential flotation method to float the lead and silver sulfide minerals. In the second stage, non-sulfide minerals are surface sulfurized by using sodium sulfide ($Na_2S.9H_2O$) and then floated using collector Z11. The sodium sulfide and collector Z11 were added in the first stage of the cumulative flotation method to simultaneously float both sulfide and non-sulfide minerals. The differential method, which floated sulfide minerals in the first stage and non-sulfide minerals in the second, performed better, according to the results, which are shown in Table 3. Lead grade and recovery were 5.11% and 68.41%, respectively, in the differential method and 16.24% and 58.43%, respectively, in the cumulative method. Similar results were obtained in the case of silver flotation, such that with staged flotation of sulfide and non-sulfide minerals, the grade and recovery of silver in the rougher concentrate were 86 g/t and 63.81%, respectively. The lead grade (16.24%) in the cumulative flotation method of sulfide and non-sulfide minerals showed a significant difference compared to the differential flotation method (5.11%), but its recovery was significantly lower, meaning that 17.08% more lead was lost to the tailings. Consequently, it is advised to employ the differential flotation method for the sulfide and non-sulfide phases of ore type A.

The pH has been below 10 during the flotation stage of sulfide minerals, as shown in the chart in Figure 4a. According to equation 3, the galena mineral is depressed because of the absence of collector adsorption and loses its surface wettability in alkaline pH as a result of the formation of metal hydroxides such as $Pb(OH)_2$ [15]. The ideal pH range for galena collectorless flotation, according to Kang and Chen (2011), is 2-11 [16]. The addition of sodium sulfide, which raises pH and eventually causes galena to be depressed, may be one factor contributing to the decrease in flotation recovery in the cumulative flotation method of sulfide and oxide minerals. Figure 4b shows that adding 500 g/t of sodium sulfide increased the pH of ore type A in the oxide-sulfide mineral cumulative flotation method from 9.15 to 10.22.

$$2PbS + 7H_2O \rightarrow 2Pb(OH)_2 + S_2O_3^{2-} + 10H^+ + 8e^- \tag{3}$$

The sulfide ion (S^{2-}), which sulfidates the surface of the cerussite mineral, may contribute to the depression of the galena mineral. Perxanthate ($ROCSSO^-$) can be formed when the sulfide ion decomposes the xanthate in the solution. Equation (4) states that lead dixanthate and sulfite react to decompose the surface of galena and produce the hydrophilic lead sulfite [15]. Because of this, the flotation of galena in the first stage (without an activator that contains sulfide ions) and cerussite in the presence of the activator has produced better flotation results.

$$PbX_2 + SO_3^{2-} \rightarrow PbSO_3 + 2X^- \tag{4}$$

Another effect of adding sodium sulfide is the reduction of pulp potential. Taheri et al. (2014) investigated the effect of adding sodium sulfide to the chalcopyrite flotation pulp and concluded that the depression effect of sodium sulfide on chalcopyrite depends on the potential of the pulp, and if the pulp potential is low, even large amounts of sodium sulfide have little depression effect on chalcopyrite [17]. Rodriguez et al. (2024), by examining the flotation conditions of galena in the presence and absence of a PAX collector and sodium sulfide activator, concluded that controlled oxidation conditions improved the flotation of galena mineral and that excessive increase or decrease in potential reduced the flotation recovery of this mineral [18]. Therefore, in addition to the pH factor, the pulp potential should also be controlled by adding different amounts of sodium sulfide.

Flotation of type B ore

Comparing the performance of the differential and cumulative flotation methods for ore type B (Table 4) revealed similar outcomes to those of ore type A. In contrast to the cumulative method, which had a 2.83% higher grade but a 14.07% lower recovery, the differential method yielded a lead grade and recovery of 19.60% and 84.61%, respectively. In terms of silver, the recovery was 90.46% and the grade was 397 g/t. The grade and recovery of lead and silver in the resultant concentrate have been higher, and ore type B has performed better in flotation than ore type A. Given that the initial pH of sample B was 8.85 (as opposed to the initial pH of 9.15 for sample type A), Figure 5b shows that the final pH rose to 9.67 after 500 g/t of sodium sulfide was added. The flotation pH was 8.88 in the scenario where the galena mineral floated first (in the absence of sodium sulfide activator) (Figure 5a). Therefore, in comparing the flotation efficiency of the differential and cumulative methods, two factors seem to be responsible for the better performance of the differential method over the cumulative method: 1) The addition of sodium sulfide increases the pH, forms lead hydroxides, and depresses galena. This change has a greater effect for sample A because its final pH (10.25-10.28) is higher. The increase in pH is due to the mineralogical characteristics of sample A, which has more carbonate minerals and finer particles than sample B, given that both samples had the same amount of sodium sulfide. 2) Lead sulfide is an insoluble salt, and the addition of sodium sulfide leads to the formation of PbS instead of $Pb(EX)_{2(S)}$ and the depression of galena [19]. By comparing the flotation efficiency of lead and silver in samples of type A and B (Tables 3 and 4), the reason for the better flotation conditions of sample B compared to sample A can be stated as follows: The amount of CaO in sample A is 2.44% higher than that of sample B (Table 1), which is also well presented in mineralogical and XRD studies (Table 2). Furthermore, the amount of iron oxides in sample A is 1.8% higher than that of sample B. This is also well known in the grinding stages of the samples, so that at the same grinding time of 27 minutes, d80 of samples A and B was 43 and 50 µm, respectively. The increase in the amount of carbonate minerals and iron oxides in sample A led to a rise in fine particles in the feed, which in turn decreased the flotation efficiency of type A compared to type B.

3.2.2. Examining the possibility of increasing the flotation efficiency of ore type A

Other factors like the use of secondary collectors (M46 and 3418A), the use of sodium silicate dispersant, and the addition of pine oil to improve frothing properties in the differential flotation method were investigated in light of the ore type A's lower flotation test performance. As shown in Figure 6a, by adding the 3418A secondary collector, the lead grade in the rougher concentrate increased from 5.11 to 7.80% and the silver grade from 86 to 127.7 g/t. The addition of the M46 secondary collector also significantly increased the lead and silver grades to 9.17% and 161.5 g/t, respectively. This means that the effect of M46 on increasing the grade was better than that of 3418A. In terms of silver recovery, both M46 and 3418A collectors increased it from 63.81% to 66.57% and 67.19%, respectively. However, in terms of increasing lead recovery, the 3418A collector performed better, as the use of this collector increased the lead recovery by 5.71%. Studies on the adsorption mechanism of the 3418A collector on galena mineral have shown that the adsorption of this collector is of the chemisorption type. In fact, this collector (as a chelating agent) has a strong affinity for lead ions on the surface of lead minerals while being highly selective towards iron-containing species. This property is also the reason for the significant increase in lead and silver grades using this collector. It has been reported that the use of the 3418A in an operating unit in Mexico has increased the silver grade in the flotation concentrate from 10 to 30 kg/t [20]. Two advantages of the phosphine-based 3418A collector (Fig 7) are its selectivity for iron sulfides and its collector power for lead, copper, and zinc sulfides. This collector performs exceptionally well when it comes to gold and silver. Compared to the xanthate group collectors, the consumption of this collector is 30 to 50% lower [21].

Due to the presence of significant amounts of silicate minerals (quartz) and carbonate minerals (calcite and dolomite), the use of 500 g/t sodium silicate as a dispersant as well as a silicate mineral depressant was investigated. According to the results presented in Figure 6b, using sodium silicate, the lead grade increased from 5.11 to 8.84% and the silver grade from 86 to 152.7 g/t. However, the use of sodium silicate significantly reduced the recovery of lead and silver. The use of a combination of pine oil and MIBC also had a similar effect compared to sodium silicate, as shown in Figure 6c, in that the lead and silver grades in the concentrate increased and the recovery of these elements decreased.

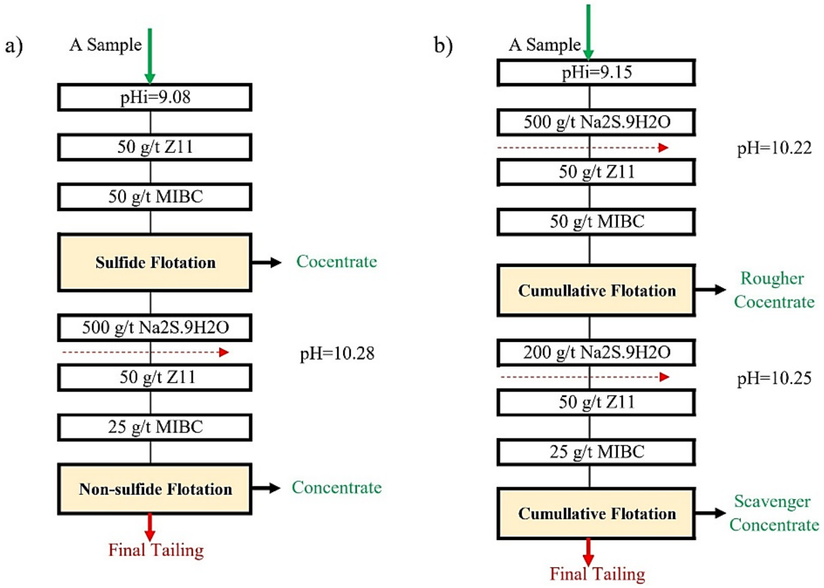

Figure 4. a) Differential and b) cumulative flotation of oxide-sulfide minerals of the ore sample type A.

Table 3. Results of the differential and cumulative flotation of oxide-sulfide minerals of the ore sample type A.

Flotation method	Product	Wt.%	Pb grade (%)	Pb recovery (%)	Ag grade (g/t)	Ag recovery (%)
differential	Feed	100.00	0.93	100	16.8	100
	Concentrate	12.45	5.11	68.41	86	63.81
	Tailing	87.55	0.34	31.59	6.9	36.19
Cumulative	Feed	100.00	0.93	100	16.8	100
	Concentrate	3.35	16.24	58.43	208.5	41.54
	Tailing	96.65	0.4	41.57	10.2	58.46

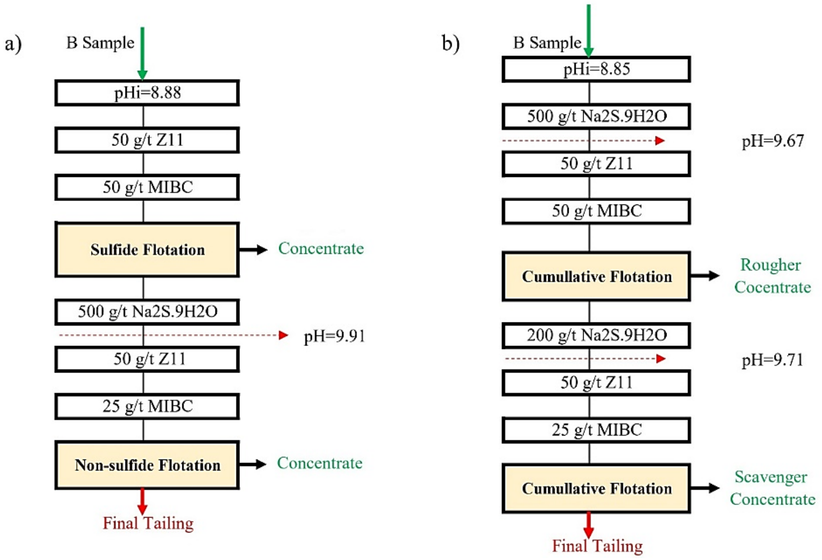

Fig. 5. a) Differential and b) cumulative flotation of oxide-sulfide minerals of the ore sample type B.

Table 4. Results of the differential and cumulative flotation of oxide-sulfide minerals of the ore sample type B.

Flotation method	Product	Wt.%	Pb grade (%)	Pb recovery (%)	Ag grade (g/t)	Ag recovery (%)
differential	Feed	100.00	0.95	100.00	16.00	100.00
	Concentrate	4.01	19.60	84.61	397.0	90.46
	Tailing	95.99	0.17	17.18	1.59	9.54
Cumulative	Feed	100	0.95	100.00	16.00	100.00
	Concentrate	3.03	22.43	70.54	304.5	57.65
	Tailing	96.97	0.28	29.46	7.00	42.35

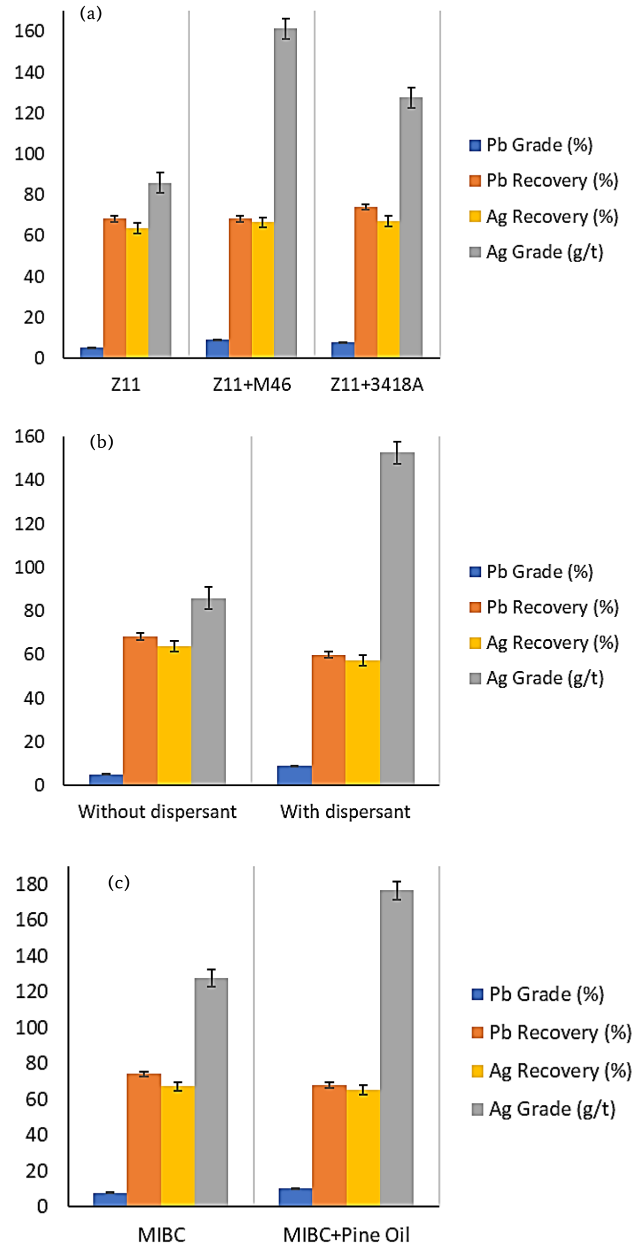

Fig. 6. a) Effect of collector type (50 g/t of Z11 and 25 g/t of 3418A/M46), b) dispersant (500 g/t of sodium silicate), c) frother type (50 g/t of MIBC and 50 g/t pine oil) on the flotation behavior of Pb and Ag minerals of the ore sample type A (differential flotation method at pH=8.5).

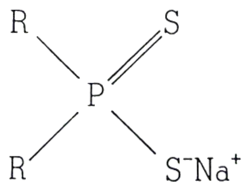

Fig. 7. Chemical structure of Aerophine 3418A collector [21].

Based on the results presented in Figure 8, 25 g/t 3418A, 50 g/t Z11, and 50 g/t MIBC were suggested for the first-stage flotation, and 500 g/t sodium sulfide (activator), 25 g/t 3418A, 50 g/t Z11, and 25 g/t MIBC were suggested for the second-stage flotation. Under these conditions, 74.12% lead and 67.19% silver were recovered, which is the highest recovery rate of total lead and silver among the conditions presented in Figure 6. It should be emphasized that in this article, only the behavior of samples A and B at fixed dosages of reagents has been compared, and optimizing the amounts of chemical consumption as well as examining the common effects of factors requires designing experiments and conducting more experiments.

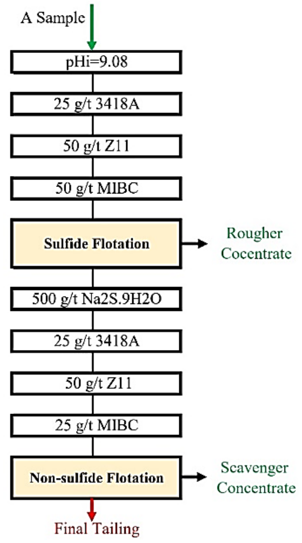

Figure 8. The proposed block diagram for differential flotation of type A ore includes the use of Z11 and 3418A collectors for the flotation of galena and silver sulfide and the use of sodium sulfide activator along with Z11 and 3418A collectors for the flotation of non-sulfide lead and silver minerals.

4. Conclusions

The purpose of this study was to use differential and cumulative methods to examine and enhance the flotation process efficiency of two ore types (A and B) from the Khanjar mixed oxide-sulfide lead and silver ore. According to the results, the differential flotation method, which floats sulfide minerals first, followed by non-sulfide minerals, performed noticeably better than the cumulative flotation of oxide and sulfide minerals. This approach produced greater recoveries of lead (84.61%) and silver (90.46%), particularly in ore type B. Furthermore, the use of the secondary collector 3418A in conjunction with Z11 demonstrated a notable improvement in the grade and recovery of silver as well as an increase in the grade and recovery of lead in the rougher concentrate during the analysis of the important parameters for the flotation of ore type A. On the other hand, despite increasing the grade, the addition of pine oil frother and sodium silicate dispersant also decreased recovery. It is advised to use a combination of 50 g/t of Z11, 25 g/t of 3418A, and 50 g/t of MIBC frother at a pH of 8.5, taking into account the importance of recovery in the rougher stage. Overall, the results of this study demonstrated that improving the performance of the flotation process of complex lead and silver oxide-sulfide mixed ores requires careful consideration of the flotation strategy (differential and cumulative) and the reagent scheme used..

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