RGNZ/MGPB

Rudarsko-geološko-naftni zbornik
(The Mining-Geology-Petroleum Engineering Bulletin)
UDC: 622:699
DOI: 10.17794/rgn.2024.1.2

Preliminary communication

Implementation of flotation to recover lead and barite from Komsheche Mine jig tailings

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Abstract
With the increase in the basic metal prices, including lead, in the global markets, the processing of this precious metal, particularly for low-grade deposits, has a high economic justification. Investigating the extraction of lead minerals, including galena which is one of the minerals associated with barite, is considered one of the products of barite by-products. The purpose of this research was to study on the potential of producing high-grade barite, and recovering lead concentrate as a by-product from the jig tailings of the Komsheche barite mine. The prepared sample was subjected to mineralogical studies, and the obtained results indicated that the sample contains barite, iron hydroxides and galena as the dominant minerals and the degree of liberation was 90-95% in the size of 100 microns. The work index of the sample was obtained by the standard bond ball mill which was 8.72 kWh/t, indicated that the sample was not hard. The lead and barite grades in sample were 0.34% and 64.83%, respectively. Considering the high specific gravity of lead and barite, the processing tests were carried out in 2 stages of primary pre-processing using Mozley multi-gravity separator and then enrichment by flotation approach. After the end of the Mozley tests, the grade of lead reached to 0.6%, some fines were removed, and the classified sample was prepared for the next stage, flotation. At the end, the grade of lead increased from 0.6% to 53% with a final recovery of 73.65% and separation efficiency was 73.48%. Also, the specific gravity and grade of barite reached from 3.9 g/cm^3 and 67% to 4.4 g/cm^3 and 95%, respectively. These values are remarkable and meet the needs of the industry.

Keywords:
lead; barite; concentration; gravity separation; flotation

1. Introduction

In the last century, the mining industry has developed significantly. However, during this period, metal deposits were extracted only in the richest parts with very high grades and processed using classic processes (**Larachi et al., 2019**). In addition, many polymetallic sulfide ores (Pb/Zn/Cu, Pb/Ba, etc.) have been exploited to produce a concentrate, while other useful materials and oxidized minerals are considered as by-products (**Abkhoshk et al., 2014**).

Barite ($BaSO_4$), barium sulfate (Ba) is the main industrial source of barium. Barium metal itself is rarely used, but its sulfate (i.e. barite) is widely considered as an industrial mineral. One of the main reasons for the commercial durability of barite is its specific gravity above 4.5 g/cm^3, and it is the heaviest non-metallic mineral (**Brobst, 1970; King, 2022**). Gravity separation techniques are used but alone are not effective as ore grades continue to decline. Currently, froth flotation is the most widely used (**Penaloza et al. 2023**). Lead sulfide (PbS) is the world's primary source of lead used in energy storage systems associated with electricity generation. This mineral contains an equal amount of Pb^{2+} and S^{2-} ions in a cubic structure (**Nowosielska et al., 2023**). Lead metal (Pb) has wide applications in industries and daily life (**Naveed et al., 2022; Yin et al., 2021**). Lead occurs mainly in the form of sulfide ores. Currently, it is mainly obtained from lead-zinc sulfide ore (**Zhang et al., 2021**). Flotation, which is based on the difference in hydrophobicity of minerals, is an effective and common method to separate galena (PbS) from lead ore (**Huang et al., 2021; Hurtig et al., 2018**).

Due to the increasing demand, the reduction of mineral resources and adoption of strict environmental laws around the world, recycling is an important issue for societies (**Valero et al., 2010**). The tailings from the mineral processing plants are normally to be considered as low-grade deposits, the recycling and reuse of these tailings would be an effective approach and, in many cases economically attractive to reduce the environmental im-

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pact, especially to minimize the environment pollution around the mine, in addition to having economic benefits (**Geise et al., 2011; Edraki et al., 2014; Broadhurst et al., 2015; Asghari et al., 2018**). One of the possible ways to achieve cost-effective production is to recover barite from waste materials/tailings. This of course, saves extraction costs which in some cases include more than 50% of the total production expenses. However secondary raw materials are often complex, leading to the need for new processing techniques (**Heinrich, 1989**). The recovery of minerals from tailings would be achieved by various processing methods, such as flotation (**Navidi Kashani and Rashchi, 2008; Lutandula and Maloba, 2013; Yang et al., 2015; Qiu et al., 2016; Yin et al., 2018**). The technological progress of this mining process allows it to be the most widely used method for the environmental desulfurization of waste minerals, concentrating of low grade ores, soil decontamination, and wastewater treatment (**Ikotun et al., 2017; Broadhurst et al., 2015; Deliyanni et al., 2017**).

In this study, it is attempted to use the flotation approach to recover lead from the concentrate part of Multi-Gravity Separator (MGS). The feed is jig's tailing with a low grade of barite (3.4 cm^3) from the Komsheche barite mine. To do that, after preparing the required representative sample, initial identification is carried out by X-ray diffraction analysis and mineralogical studies, and after determining the type of associated minerals, and their involvement with lead, flotation experiments are accordingly conducted with the aim of producing individual lead and barite concentrates.

2. Materials and methods

2.1. Materials

The required sample for this research was prepared from the Komsheche barite mine, located in Isfahan City, Iran. Analysis and preliminary studies were employed to fully identify the sample including mineralogy, XRD, XRF, microscopic studies, and polished sections. In **Table 1**, the results of element analysis are reported. Also, to identify the mineral composition of the sample, it was analyzed using X-ray diffraction (XRD). In **Figure 1**, the graph related to the peaks of XRD analysis is given. The abundance of minerals in the tested sample was barite ($BaSO_4$), iron- dolomite ($Ca(Mg,Fe)(CO_3)_2$), calcite ($CaCO_3$), quartz (SiO_2) and celestine ($SrSO_4$).

Microscopic studies and polished sections in +75-106 μm size fraction indicated that metallic and semi-metallic minerals were included in a total of about 1-1.5 percent in volume of all parts. Galena in the range of 0.1-0.25% in volume, goethite, hematite and rarely magnetite in the range of 0.6-0.8% by volume, pyrite in the

Table 1: The results of element analysis.

BaSO4	Mn	Zn	S	Mg	Fe	Cu	Ca	Al	Pb
64.83	0/05	0.03	0.12	1.5	1.14	0.10	6.95	0.43	0.34

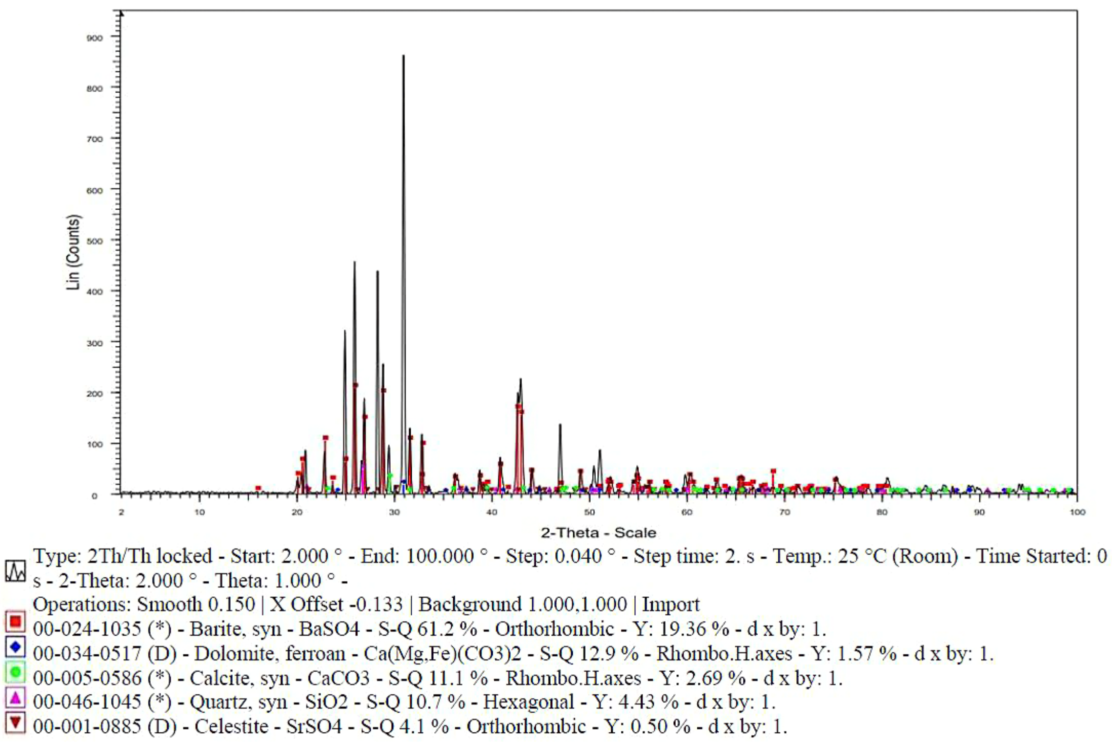

Figure 1: XRD pattern of the studied sample.

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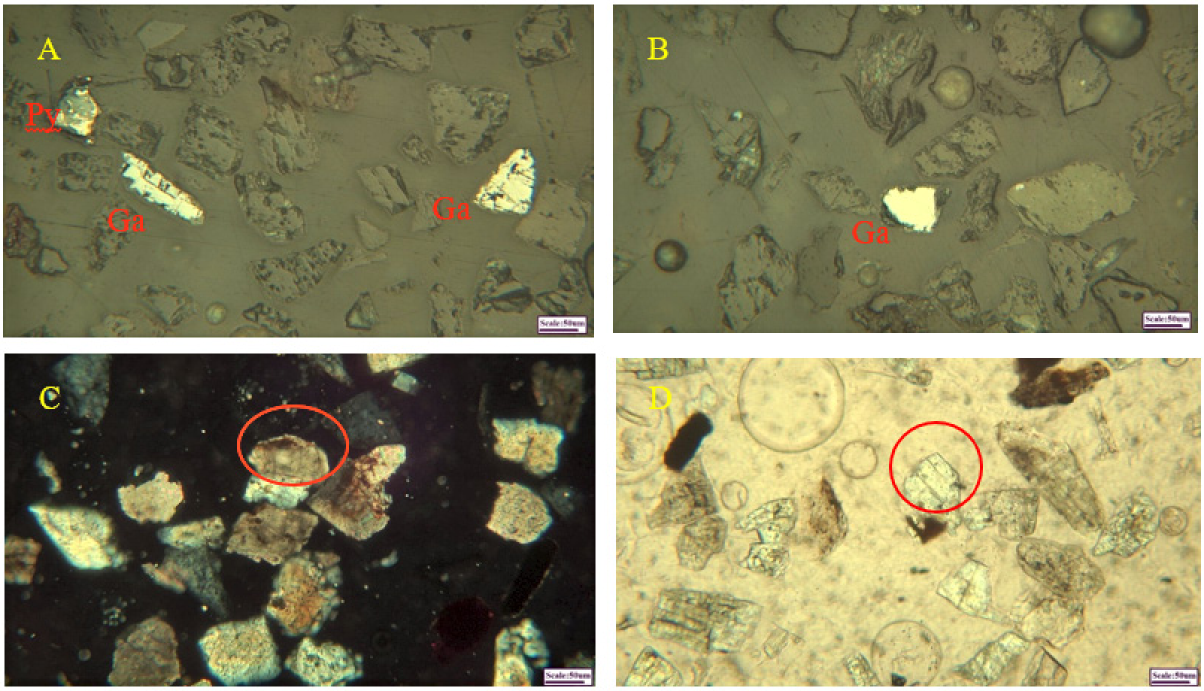

Figure 2: A: free particles of galena and oxidized pyrite, B: free particle of galena, C: barite associated with calcite, D: barite associated with goethite.

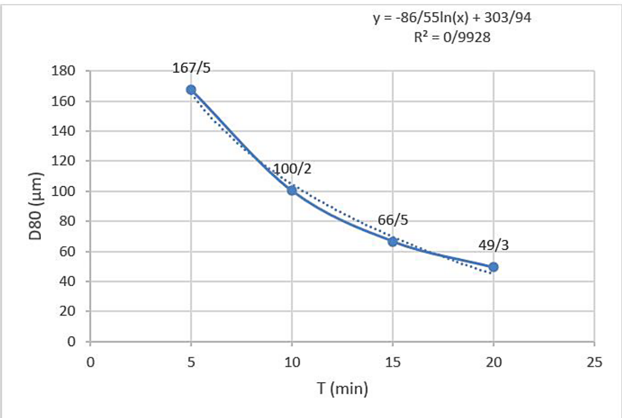

Figure 3: The trend of d80 values versus of relevant grinding times.

range of 0.1-0.2% in volume, cerusite and malachite in the range of 0.1%, rarely chalcopyrite, magnetite and smithzonite can be mentioned. Most of the particles of hematite, galena, goethite and pyrite are in free or unlocked form, rarely are the particles involved. The galena degree of liberation in the current size fraction was determined to be about 90-95%, for malachite about 85-90%, and for cerusite about 80-85%. The degree of liberation of hematite, goethite, magnetite and pyrite was at least 95%. In the non-metallic part, barite was the main transparent mineral, followed by calcite. Dolomite, quartz and in small amounts siderite and ankrite were secondary transparent minerals. The barite degree of liberation was at least 95%. The few non-free or locked fragments of barite were mainly associated with calcite, dolomite and goethite (see **Figure 2**).

The optimal grinding time by a laboratory rod mill, and also the work index were measured by the bond standard ball mill. For this purpose, the raw materials were crushed to finer than 3 mm and after performing the Bond Standard Ball Mill test, the work index of the sample was determined equal to 8.72 kWh/t which indicates the sample was not hard.

The d80 size diagram of the material was obtained according to the different grinding times by a rod mill (see **Figure 3**). According to the microscopic studies, the appropriate degree of liberation of 100 µm was chosen. Also **Figure 3** shows that after 10 minutes of rod milling, the d80 of the product reaches to 100 microns.

2.2. Methodology

Considering the high specific gravity of lead and the need for initial pre-processing, gravity separation methods were the best and most efficient approaches prior to flotation, which are the cheapest approaches to be used as pre-processing/pre-concentration of these types of material. Therefore, the experiments were planned and carried out in 2 series, each one containing several tests to check the influencing parameters and their optimal conditions. These 2 series include the following:

i) 1st series: Mozley multi gravity separation pre-processing tests.
ii) 2nd series: flotation tests.

2.2.1. Mozley Multi Gravity Separation

Mozley multi gravity separation tests were performed in 2 steps designed with conditions of: variable slope (2

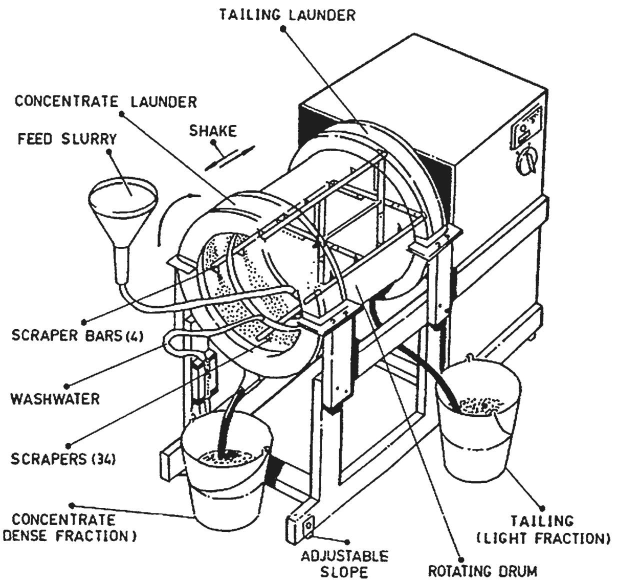

Figure 4: Mozley Multi Gravity separation device (**T. Ççek, 2008**).

and 4 degrees), rotation speed of Mozley drum 280 rpm, feed water flow rate 2 l/min., washing water flow rate 4 l/min., the total feeding time was 2 minutes, and the total test time was 5 minutes. A view of the Mozley Multi Gravity separation device is shown in **Figure 4** (**T. Ççek, 2008**).

2.2.2. Flotation tests

Flotation experiments were performed using laboratory Denver D12 flotation cell in 2 stages. First, lead was separated, and then barite flotation tests were performed on the final tailings of lead tests.

2.2.2.1. Lead flotation tests

In lead flotation, 1-kilogram samples recovered from Mozley gravity separator tests, to optimize different parameters such as dispersant dosage, collector dosage, type and dosage of frother, and also sodium silicate as a dispersant, ethyl Zanthate (Z4) as a collector, and MIBC, RC250, A65 as a frother were used. The flotation experiments were performed using different doses of Na_2SiO_3 from 500 to 15000 g/t and the optimal dose of collector used was determined by adding 100, 150 and 200 g/t to the pulp, also the value of pH=7.6, adjusted rotor speed of 1250 rpm, 29% pulp density, preparation time of disperser, collector, and frother were 5, 3, and 1 minutes, respectively, and froth collecting time of 5 minutes, which was constant.

After determining the optimal parameters for the initial rougher test (the maximum recovery in each stage), a closed and continuous circuit test was designed, which used about 2800 g of the sample.

2.2.2.2. Barite flotation tests

To perform barite flotation experiments, all tailings from 5 continuous circuit tests of the previous step were combined and homogenized. In the first step, to calculate the initial barite grade, the specific weight of the sample was calculated prior to any testing, according to the following conditions and methods.

In order to calculate the specific weight of the barite sample using a pycnometer, first the weight of the empty pycnometer (P_0) was measured. Then the weight of the pycnometer filled with oil was measured (P_w). Next, the weight of the pycnometer was measured with the solid, which should be about 30 to 70 g (P_s). At the end, the weight of the pycnometer was measured with solid and oil (P_p), and the specific gravity was calculated with the following **Equation 3**.

$$\sigma = \frac{\mathrm{Ps} - \mathrm{P0}}{(\mathrm{Pw} - \mathrm{P0}) - (\mathrm{Pp} - \mathrm{Ps})} \cdot (\sigma \ liquid) \qquad (3)$$

To obtain a product with a high grade and to be acceptable for industry, the floated part of the previous steps was conducted to the cleaner cells. In cleaner 1, no chemicals were added, but in cleaner 2, 1 ml of frother (pine oil) was added. A 2-liter cell was used as cleaner 1, and cleaner 2 was 1 liter, and the rotor speed was set to 1000 rpm, which was identical for both tests.

3. Results and Discussion

3.1. Mozley Multi Gravity Separation Tests

Table 2 presents the results which indicate the grades in the test (10 minutes grinding time), were about 0.42%, 0.43%. However, the recoveries in this test are better values.

The continuous test is designed in a closed circuit which was run in this work. In this way, the sample was initially passed through the Mozley drum with 2 degrees of slope (to obtain better recovery) and the concentrate was then passed through with a 4 degrees Mozley slope to achieve the maximum grade. The results are shown in **Table 3**.

The gravity tests increased the acceptable grade from 0.34% Pb to 0.6% Pb in Mozley's initial tests, as well as removing the fine and separating the materials.

Table 2: The first test (Mozley gravity separation - primary feed)-10 minutes of grinding.

Test	Feed weight (g)	Concentrate weight (g)	Tail weight (g)	Concentrate Pb (%)	Tail Pb (%)	Recovery	Slope
1	1000	693	307	0.43	0.14	87.6	4
2	1000	745.9	254.1	0.42	0.11	92.14	2

Table 3: Mozley gravity separation tests-primary feed.

Test	Feed weight (g)	Concentrate weight (g)	Tail weight (g)	Concentrate Pb (%)	Tail Pb (%)	Recovery	Slope
1	2000	1287.8	712.2	-	-	72.3	2
2	1287.8	819.4	468.4	0.6	0.13		4

3.2. Flotation tests

3.2.1 Determining the optimal parameters of lead flotation test

3.2.1.1. Sodium silicate

Sodium silicate is normally added to the pulp as a dispersant agent to reduce fine particle disturbance and maintaining particle dispersion in flotation (**Atrafi et al. 2012 Larachi et al. 2019**). It was observed that the dose of 500 g/t leads to an improvement in the recovery of lead. However, with a dose of 1500 g/t, the lead recovery decreased. Therefore, results show that the dose of 500 g/t was optimal (see **Figure 5a**). In this part, it would be postulated that sodium silicate played a dual role within the flotation. It means that at lower doses, it only acts as a dispersant and increased Pb recovery, while at higher doses it decreased Pb recovery through the precipitation of Pb^{2+} ions in the solution.

3.2.1.2. Ethyl Xanthate Z4

The results showed that lead would be recovered with 51.90% when the collector dose was 100 g/t. The recovery of Pb increased significantly for ethyl xanthate dose of 150 g/t, and reached to 62.09%. However, with a dose of 200 g/t, the recovery of galena was improved and reached to 67.82%. According to **Figure 5b**, the recovery increased with an increase in the amount of ethyl xanthate.

3.2.1.3. Type and dosage of frother

The effects of three types of frothers: i) MIBC, ii) Polypropylene glycol (A65), and iii) RC250, with differ-

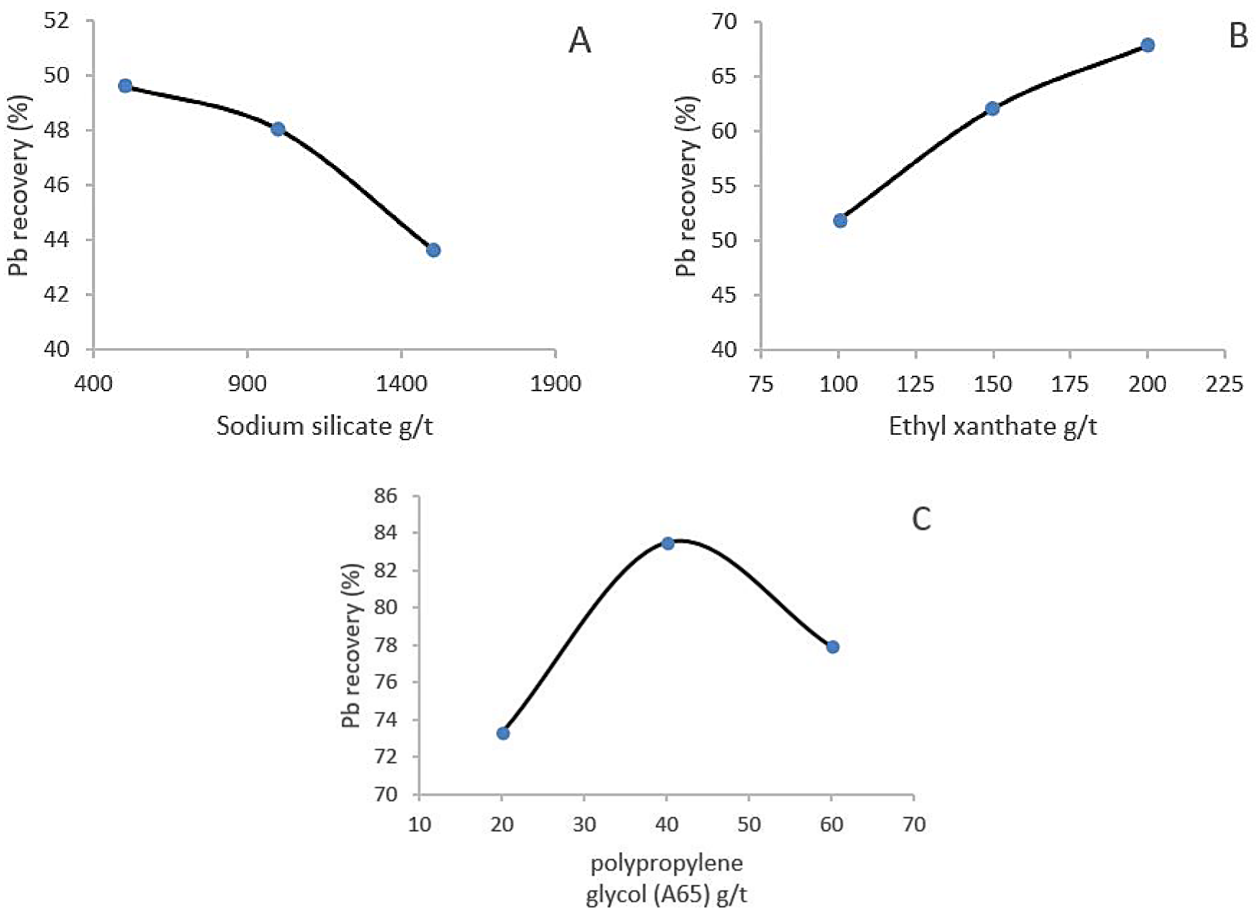

Figure 5: Effect of parameters A: sodium silicate, B: ethyl xanthate and C: polypropylene glycol on lead recovery.

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Table 4: Optimum conditions of the flotation test of the closed circuit stage.

	Rougher	Scavenger	Cleaner 1	Cleaner 2
pH	7.6	7.6	7.6	7.6
Dispersant	sodium silicate - 500 g/t	-	-	-
Collector	Ethyl Zanthate (Z4)- 200 g/t	Ethyl Zanthate (Z4)- 200 g/t	-	-
Frother	A65–40 g/t	A65–40 g/t	A65–40 g/t	-
Rotor speed	1400 rpm	1400 rpm	1100 rpm	1100 rpm
Pulp density	25	25	25	25
Preparation time of disperser	5 min	-	-	-
Preparation time of collector	3 min	1 min	-	-
Preparation time of frother	1 min	1 min	30 sec	-
Froth collecting time	4 min	8 min	2 min	1 min

ent properties and solubility on lead recovery were studied. In identical conditions and a dosage of 20 g/t, the tests showed that lead recovery with MIBC, A65 and RC250 were 67.82%, 73.31% and 72.38%, respectively, which means according to the results, the highest recovery was for the A65 type frother.

In order to check the frother dosage, different amounts of 20, 40 and 60 g/t were added to the pulp. The optimal dose was determined to be 40 g/t. With optimal conditions, the amount of collector and disperser and the amount of 40 g/t, the recovery of Pb reached to 83.54%. While for 20 g/t, the Pb recovery was 73.31%, and by increasing the amount of frother, a recovery of 77.91% was achieved, as presented in **Figure 5c**.

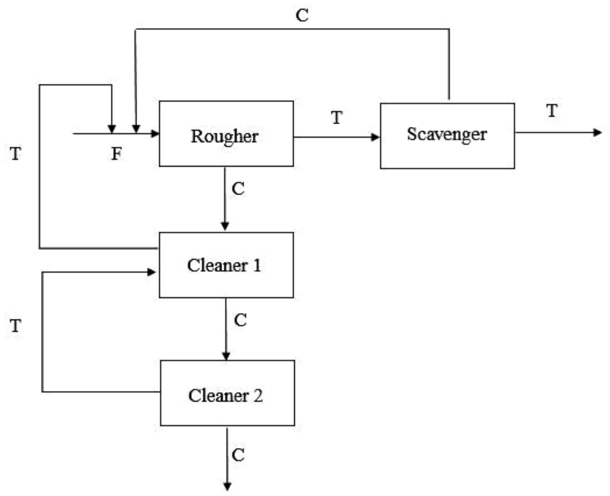

Figure 6: Lead closed circuit flowsheet.

3.2.1.2. Continuous and closed-circuit of lead flotation test

At this stage, according to the determination of the optimal flotation conditions, the closed circuit test was performed under the following conditions, and at the end, the closed cycle flowsheet was accordingly drawn (see **Figure 6**). The optimal conditions are listed in **Table 4**.

These tests were done in 5 closed cycles, and in each step, 2800 g of sample was used as fresh feed.

At the end of each closed cycle, from test 1 to 4, sampling was carried out from the tailings and the final concentrate. For the last series of tests which was test 5, tailings and final concentrate, the representative samples were also prepared and analyzed from the tailings and concentrates returned in the circuit. The results are listed **Table 5**.

The overall circuit recovery was calculated using **Equation 1**, as following:

$$R = \frac{c(f-t)}{f(c-t)} \cdot 100 = \frac{52.88(0.34-0.09)}{0.34(52.88-0.09)} \cdot 100 = 73.65\% \quad (1)$$

And the separation efficiency, Shultz equation (**Wills, 2006**) was calculated based on **Equation 2**, as follows:

Table 5: The results of the optimal flotation test of the closed circuit stage.

Sample no.	Concentrate weight (g)	Tail weight (g)	Lead grade at concentrate (%)	Lead grade at tail (%)
1	9.2	2693	39.15	0.05
2	8.8	2747.8	53.85	0.10
3	18.9	2719.1	53.67	0.11
4	18.2	2775.1	52.29	0.09
5	17.1	2763.6	53.47	0.09
Tailing of cleaner 1, test 5	-	54.5	-	2.41
Tailing of cleaner 2, test 5	-	7.4	-	37.47
Return concentrate scavenger, test 5	29.8	-	5.83	-

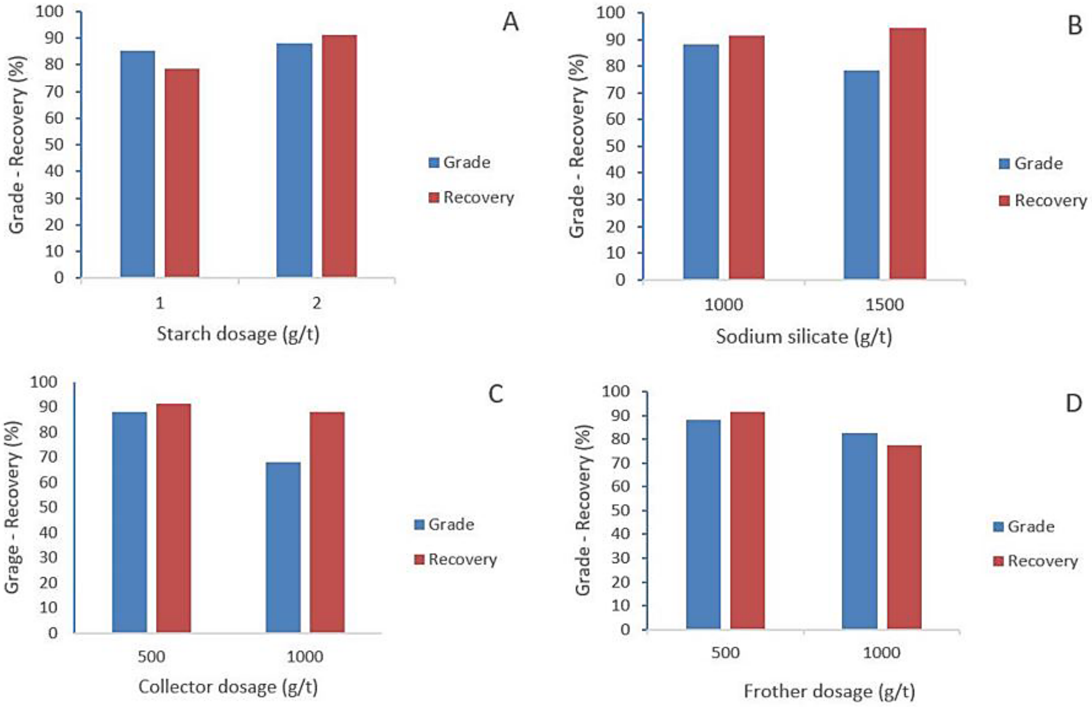

Figure 7: A: Effect of starch dosage, B: The effect of sodium silicate dosage, C: The effect of collector dosage and D: The effect of frother dosage on barite grade-recovery.

$$S.E = \frac{c(f-t)(c-f)(100-t)}{f(c-t)^2(100-f)} \cdot 100 =$$

$$= \frac{52.88(0.34-0.09)(52.88-0.34)(100-0.09)}{0.34(52.88-0.09)^2(100-0.34)} \cdot 100 =$$

$$= 73.48\% \tag{2}$$

3.2.2. Barite flotation tests

According to **Equation 3**, the specific weight of the sample was calculated. The sample was obtained with a specific gravity of 3.9 and a grade of 67%. The desired grade of barite required in relevant industry is a specific weight above 4.2 and grade of 83.5% or higher. Then, optimum tests were designed and performed to study on the effect of various factors such as depressant, collector, and frother.

According to the proposed collectors for barite flotation which include AERO 825 and Oleic acid, no favourable results were obtained from the oleic acid tests, and practically no special grade increase occurred. Therefore, AERO 825 was selected as the best collector. In addition, regarding the pH, the initial tests in the pH of the environment (around 6.7) did not have any special results, and the optimal tests were performed at pH 9.5 to 10.5. The pH was adjusted by sodium carbonate. The rotor speed and pulp density were 1250 rpm and 31%, respectively, and the preparation time of depressant, collector, frother, and froth collecting time were 5, 3, 1, and 8 minutes, respectively.

Due to the presence of about 10% silicate and 12% carbonate in the original sample, sodium silicate and starch were used to depressant silicate and carbonate gangue, respectively. Silicates and carbonates reduced the quality of the output product due to their low specific gravity.

Barite flotation was studied by changing the dosage of AERO 825 as a collector from 500 to 1000 g/t. The flotation test performed using a dose of 1000 g/t which re-

Table 6: Results of primary and secondary washing tests - barite

Test	Barite specific gravity at concentrate (g/cm^3)	Barite specific gravity at tail (g/cm^3)	Barite grade at concentrate (%)	Barite grade at tail (%)
1	-	4.05	-	75.5
2	-	4.32	-	90.5
3	4.39	4.13	94	19.87
4	4.40	4.17	95	20.63

Test 1: The waste of the cleaner 1; Test 2: The waste of the cleaner 2; Test 3: final concentrate (first minute froth collecting); Test 4: final concentrate (second minute froth collecting)

sulted barite product with a grade of 68% and a recovery of 88.13%. While with 500 g/t of collector, the recovery was 91.35% and the grade was 88%. Usually, collectors are used in a lower concentration to recover barite from waste (**Heinrich 1989; Bulatovic 2015**). According to the results of experiments, increasing the amount of starch as a depressant of carbonate waste increased the grade and recovery of barite. With the increase of starch from 1 to 2 g/t, the recovery increased by about 13%. However, adding sodium silicate as a depressant of silicate gangue in the presence of starch did not present much effect on the grade and recovery. Moreover, with an increase in the dose of frother, both grade and recovery values decreased. **Figure 7** shows the effect of starch dosage, the effect of sodium silicate dosage, the effect of collector dosage, and the effect of frother dosage on barite grade-recovery.

3.2.2.1. Primary and secondary cleaner tests

According to the results of cleaner test in 2 steps and with no need to re-add chemicals, a 95% of concentrate grade with a specific gravity of 4.4 g/cm^3 was obtained, which would remarkably meets the needs of the industry (4.2 g/cm^3). The test results are listed in **Table 6**.

4. Conclusions

In this research, the jig tailings sample from Komsheche barite mine was used, and based on the results the following conclusions were made:

- The results of mineralogical studies indicated the presence of barite, iron hydroxides and galena as the dominant minerals, and the degree of liberation was 90-95% in the size of 100 microns. The work index of the sample was obtained by the Bond standard of ball mill, equal to 8.72 kWh/t, which indicated that the sample was not hard.
- After gravity separation tests with a Mozley Multi-Gravity Separator at different times and slopes, the resulting values were very close to each other, such that, in slope 4, the grade of lead was 0.43, and the grade of lead in slope 2 was 0.42. The grade of lead reached from 0.34% to 0.6%, and the total recovery was about 72%. Appropriate pre-processing and fine removal was performed with a more uniform sample with a relatively higher grade to be available for the next steps.
- The results of the different grinding times indicated that grinding for more than 10 minutes did not make much difference in grade and recovery. In addition, for the most optimal state of grade and recovery, and considering economic and cost issues, it can be considered that a time as short as 10 minutes is convenient.
- The grade of lead increased slightly with Mozley's pre-processing tests which was not acceptable, and then the flotation approach was employed to increase it.
- After performing relevant tests to optimize all the different parameters of lead flotation, including pH, dispersant, collector and frother, the best result for flotation in the final circuit was produced. The grade of lead reached from about 0.6% to 53% and the final recovery was 73.65% and separation efficiency was 73.48%. Results based on the conditions: pH: about 6.7, dispersant: sodium silicate 5% - 500 gr/t, collector: ethyl xanthate (Z4) 1%- 200 gr/t, frother: A65 1% - 40 gr/t, agitation speed: 1400 rpm, pulp solid percentage: 25%, dispersant preparation time: 5 minutes, collector preparation time: 3 minutes, frother preparation time: 1 minute, froth collecting time: 4 and 5 minutes were obtained.
- Lead flotation tailings were then used for barite flotation. After optimizing all the different parameters of barite flotation, including pH, depressant, collector and frother, the best result for flotation in the final circuit was achieved. The specific weight and grade of barite increased from 3.9 g/cm^3 to 4.4 g/cm^3 and 67% to 95%, respectively. Due to the presence of 10% silicate and 12% carbonate in the original sample, sodium silicate and starch were used as depressant silicate and carbonate gangues, respectively. Results based on the conditions: sodium carbonate (adjust pH=10), sodium silicate 5% (depressant of silicate gangue - 1000 g/t), starch (depressant of carbonate gangue - 2 g), collector: AERO 825 1% - (500 g/t), frother: pine oil (1 cc), agitation speed: 1250 rpm, pulp solid percentage: 31%, depressant preparation time: 5 minutes, collector preparation time: 3 minutes, frother preparation time: 1 minute, froth collecting time: 8 minutes were obtained.

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SAŽETAK

Primjena flotacije za dobivanje olova i barita iz jalovine plakalice rudnika Komsheche

Porastom cijena osnovnih metala i olova na svjetskim tržištima prerada ovoga važnog metala dobiva sve veću ekonomsku opravdanost posebno za ležišta niskoga sadržaja rude. Galenit je jedan od najvažnijih minerala olova u rudama, a njegova pojava može biti povezana i s baritom. Kada je to slučaj, u produktima oplemenjivanja barita javlja se i olovo, odnosno galenit. Svrha ovoga istraživanja bila je istražiti potencijal za dobivanje/proizvodnju visokokvalitetnoga barita uz izdvajanje koncentrata olova kao nusprodukta jalovine iz plakalice rudnika barita Komsheche. Mineralna analiza uzorka jalovine pokazala je da uzorak pretežno sadržava barit, hidrokside željeza i galenit te da je pri veličini zrna od 100 mikrona stupanj oslobođanja/raščina tih minerala u intervalu od 90 do 95 %. Bondov radni indeks od 8,72 kWh/t pokazao je da se ne radi o tvrdome materijalu. Sadržaj olova odnosno barita u uzorku iznosio je 0,34 % odnosno 64,83 %. S obzirom na to da su galenit i barit minerali velike gustoće, obogaćivanje je provedeno gravitacijskom koncentracijom na uređaju tipa Mozley te flotacijom. Gravitacijskom koncentracijom udio olova povećan je na 0,6 %, nakon čega su sitne čestice izdvojene klasiranjem, kao priprema uzorka za sljedeći stupanj obogaćivanja flotacijom. Flotacijom je sadržaj olova u koncentratu povećan od 0,6 % do 53 % s iskorištenjem od 73,65 % pri učinkovitosti separacije od 73,48 %. Specifična gustoća uzorka barita povećana je s 3,9 g/cm^3 na 4,4 g/cm^3, a sadržaj barita u koncentratu sa 67 % na 95 % barita.

Ključne riječi:
olovo, barit, koncentracija, gravitacijska koncentracija, flotacija

Authors' contribution

Arash Faramarz (1) (MSc Student of Mineral Processing) performed tests and analyses. **Hassan Maleki (2)** (MSc of Mineral Processing) provided reports and wrote the article. **Mohammad Noaparast (3)** (Full Professor of Mineral Processing) proposed the idea and guided the research. **Golnaz Jozanikohan (4)** (Assistant Professor of Mineral exploration) proposed the idea and guided the research. **Hanieh Noeparast (5)** (Graduate student) Helped in translating and correcting the grammar and writing of the text.