RESEARCH PAPER

The Possibility of Flotation and Gravity Separation Methods for a Mixed Pb and Zn Oxide-Sulfide Ore

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ABSTRACT

With the depletion of lead and Zn sulphide reserves, processing of oxide and oxide-sulphide mixture deposits has gained significant importance. In the present research, lead and zinc minerals were concentrated from an oxide-sulphide ore containing 2.92% Zn (2.42% oxide) and 1.61% lead (1.23% oxide) with silicate gangue. Flotation experiments were conducted with two approaches: differential flotation of sulphide and oxide minerals in several stages, and cumulative flotation of mixed sulphide-oxide minerals. The effects of desliming and flotation reagents were examined. Results indicated that differential flotation of sulphide and oxide minerals is impractical for this ore. Optimal conditions for ore concentration were identified as flotation of mixed sulphide-oxide minerals of lead with d_{80} of 90 µm and the addition of 6000 g/t sodium sulphide in the rougher stage and 1000 g/t in the scavenger stage. Bromoform heavy liquid tests revealed that Zn recovery in the heavy fraction remained relatively unchanged with particle size reduction, whereas Pb recovery increased from 74.22% to 89.09%. Finally, a combination of mixed sulphide-oxide flotation along with gravity separation using a shaking table and spirals was proposed.

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INTRODUCTION

Lead and zinc (Pb-Zn) minerals are naturally associated with each other. Pb-Zn deposits typically exhibit three types of mineralization: a) Sulphide Minerals that are mainly found in primary hypogene sulphide ore, where mineralization occurs in several sequential stages. Sphalerite (ZnS) and galena (PbS) are the primary sulphide minerals for Zn and Pb, respectively. b) Non-sulphide (oxide) ores that can be classified into two types based on hypogene or supergene alteration. The primary sulphide ore body is usually protected from supergene alteration by oxidized hypogene ores. Supergene oxidation ores are characterized by the absence of sulphide minerals and their outcrops at or near the surface. c) Mixed sulphide-oxide ores that have very complex mineralogy and are often found in transition zones and occasionally in oxidized zones [1].

Significant reserves of mixed sulphide-oxide ores are found around the world. Processing of these ores are challenging due to their complex mineralogy. The pre-concentration and separation using the flotation method are traditionally

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employed for processing the mixed sulphide-oxide ores [2-4]. The presence of sulphide and oxide minerals makes flotation very difficult[5]. Therefore, the appropriate reagents and their optimal dosages must be carefully controlled [6-8].

Single-stage flotation is inadequate for these types of ores, so employing multiple flotation stages and stepwise reagent addition is important. Hence, a practical design of the flotation process is necessary for treating mixed sulfide-oxide Pb-Zn ores. The selection of the reagent scheme depends on the gangue mineral composition and the specific types of Pb and Zn oxide minerals of the ore. A possible variation in the flowsheet could be the inclusion of a gravity concentration stage prior to the flotation process [2].

In some Pb-Zn processing plants, heavy media concentration method is used as a pre-concentration method [2, 9, 10]. In some cases, extensive grinding is needed to achieve sufficient mineral liberation because of the fine-grained nature and interlocking of oxide minerals with sulfide minerals [11] and the grinding media has an important effect on the flotation behavior of sulfide minerals[12]. However, excessive grinding produces large amounts of slime, which affects the flotation behavior of target minerals. Additionally, the flotation of Pb-Zn oxide minerals presents a challenge in processing of these ores. For successful flotation of Pb and Zn oxide minerals, precise control of the sulphidizing agent amount and slurry pH is required [6, 7, 13]. Wei et al. (2021) concluded that flotation at pH around 9 without desliming improved the recovery of Pb and Zn from mixed sulfide-oxide ores. Their results showed that this flotation method is practical for both laboratory and industrial scales [14]. Due to the variability in Pb and Zn deposits globally, flotation processes differ significantly not only between deposits but also within a single deposit [2].

In the present research, a suitable process plan was investigated for producing separate Pb and Zn concentrates from a mixed oxide-sulphide Pb and Zn ore. Two approaches were tested in the flotation experiments: a) differential flotation of sulfide and oxide minerals, and b) flotation of mixed sulfide-oxide minerals. The effects of desliming and optimizing the flotation reagents were investigated. Additionally, to explore the potential for gravity concentration of Pb and Zn in the sample, bromoform heavy liquid tests were conducted. Thus, the novelty of the present paper lies in its comprehensive approach to optimizing the flotation process for producing separate Pb and Zn concentrates from a mixed oxide-sulphide ore and testing the feasibility of gravity concentration for Pb and Zn minerals of the sample.

MATERIALS AND METHODS

Sample preparation and characterization

A 250 kg representative sample was collected from the Robat Pb and Zn mine located in the Markazi province (Iran). The sample, with a d_{100} of approximately 100 mm, was crushed using

Table 1. Chemical analysis of the sample.

Component	wt. %	Component	g/t
SiO_2	69.39	Cu	188
CaO	7.57	Ti	177
Fe_2O_3	6.63	Sr	136
Pb	1.61 (1.23 oxide)	As	60
Zn	2.92 (2.42 oxide)	Cd	54
LOI	6.92	Sb	25
Al	0.57	Ni	20
SO_3	0.52	Cr	16
MgO	0.49	Ag	16
MnO	0.43	Zr	8
K_2O	0.36	Co	6
BaO	0.35	Ce	5

laboratory jaw and roller crushers. Then, the crushed sample was divided into sub-samples using a Jones riffler. The chemical composition of the sample was analyzed by Zarazma Company (Tehran, Iran) using alkaline fusion (Peroxide Fusion 02) and ICP-MS methods. According to Table 1, the representative sample contains 1.61% Pb and 2.92% Zn. Approximately 74% of the total Pb and 83% of the total Zn in the feed are in oxide form, which is important in the flotation behavior of target minerals.

Mineralogy and degree of freedom studies

Polished and thin sections were prepared to study the Pb, Zn, and gangue minerals. The results of microscopic studies showed that the abundance of sulfide minerals is much less than that of oxide minerals (Table 1). From the total Pb and Zn content of 1.61% and 2.92% of the sample, 1.23% and 2.42% are in the form of oxide minerals,

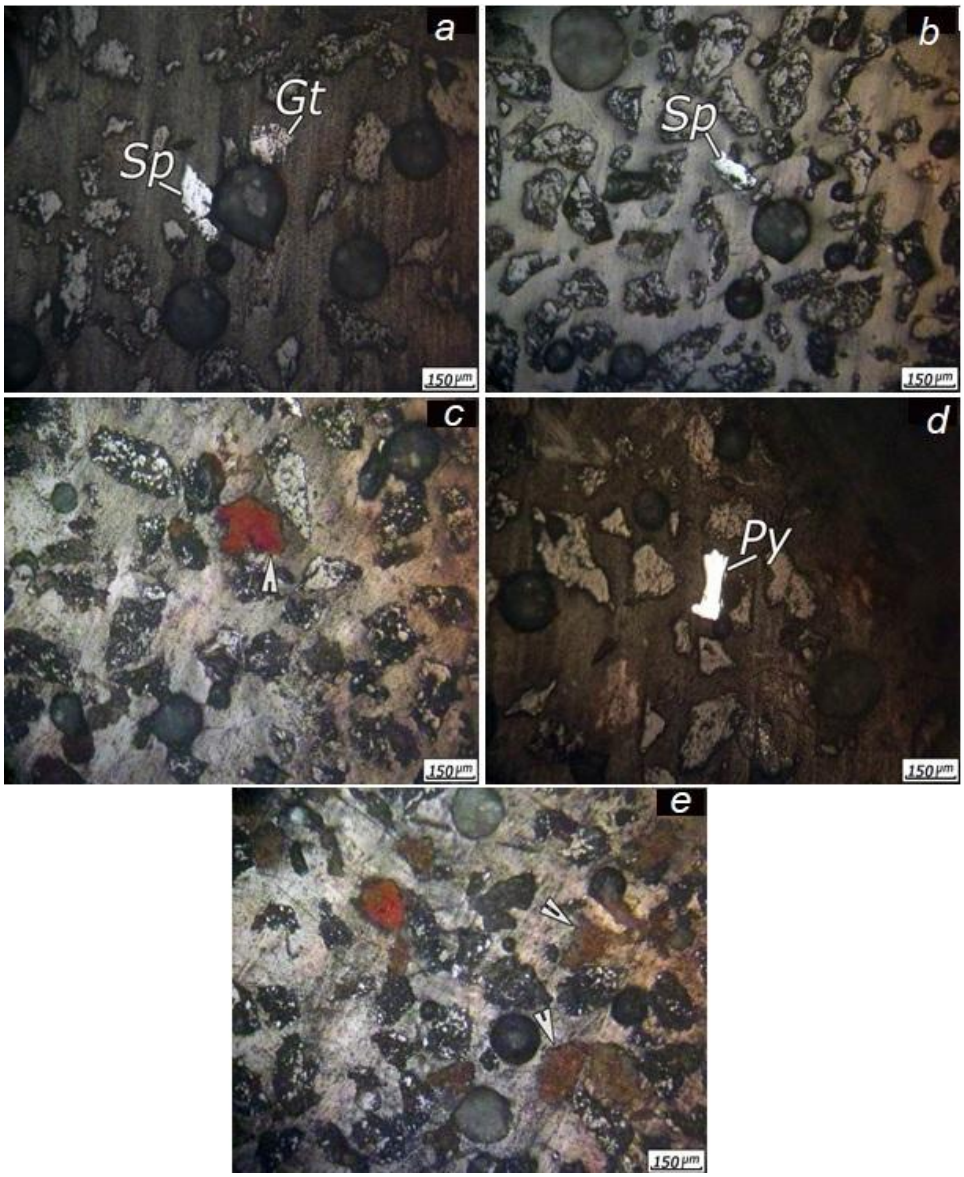

Fig. 1. a) Sphalerite crystal (Sp) in free form. b) Sphalerite and goethite (Gt). c) Pyrite crystal (Py) in free form. d) Zn carbonate mineral in free form. e) The red and brown minerals are Zn carbonate minerals; those with lower color intensity and tending towards brown (arrowhead) have a lower Zn grade.

respectively. To better examine oxide Zn minerals in the microscopic studies, a special solution (Zinc Zap) was used. The Zinc Zap solution changes the color of Zn carbonate minerals to a red-brown color.

Opaque (sulfide) minerals were studied in the polished sections. Sphalerite (Fig. 1a and b), pyrite (Fig. 1d), goethite (Fig. 1a), and galena are the main opaque minerals of the sample. The abundance of sphalerite and galena is much less than that of pyrite and goethite. Zn carbonate minerals, are shown in Fig. 1c and e (changed color to red-brown with the Zinc Zap solution).

Microscopic studies of thin sections in Fig. 2 showed that it was found that quartz is the most abundant non-metallic mineral of the sample. Quartz appears in two forms: quartz microcrystal accumulations (polycrystalline quartz) and large single crystals. Along with quartz, fragments of carbonate minerals were also observed (Fig. 2b). Jarosite is another non-metallic mineral detected in the thin sections with low abundance (Fig. 2d).

The results of the degree of freedom study of minerals is presented in Fig. 3. The degree of freedom of minerals was measured in three size fractions: +106 µm, +75-106 µm, and +38-75 µm. The study shows that sphalerite and galena are mostly liberated even in the coarse fractions. Sphalerite particles are often locked with gangue minerals. For Pb and Zn oxide minerals, the degree of freedom increases with the reduction of particle size. The degree of freedom for Zn and Pb oxide minerals in the +106 µm size fraction is 60.97% and 71.3%, respectively, while in the +38-75 µm size fraction, it reaches 94.42% and 94.68%, respectively. In the study of the liberation of Zn carbonate minerals, contact locking and gangue inclusions within the Zn carbonate minerals were observed. Additionally, Zn carbonate minerals are sometimes found in the gangue minerals as inclusions. Higher grade Zn carbonate minerals appear completely red-brown, while lower grade

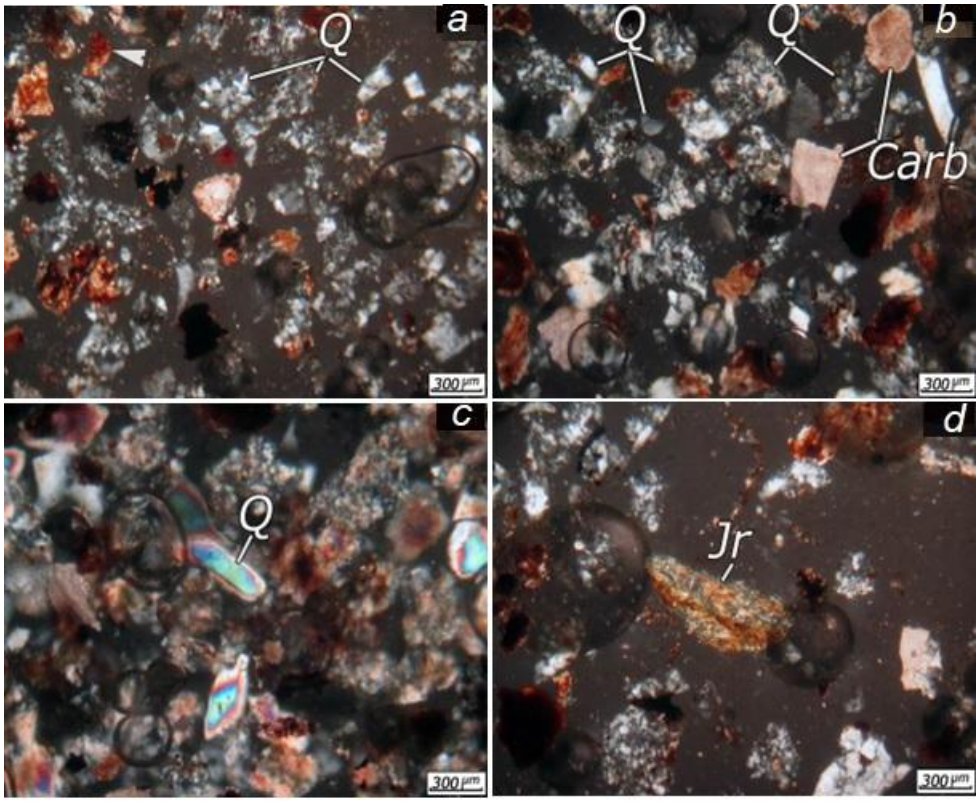

Fig. 2. Quartz crystals (Q) in gray and white, often in the form of fine aggregates (polycrystalline). Single quartz crystals are less abundant. Carbonate minerals (Carb), tending to brown, are also seen in the section. b) Quartz, which is significantly abundant in the studied section, and red-brown particles of Zn minerals. c) Jarosite (Jr) coarse crystal along with quartz.

parts are less red-brown after the addition of Zinc Zap solution.

Flotation Experiments

A Denver model 1.6-liter laboratory flotation machine equipped with rotor and stator was used for the flotation experiments. The experiments were conducted using two approaches: differential flotation of sulfide and oxide minerals, and cumulative flotation of mixed sulfide-oxide minerals.

Differential flotation of sulfide and oxide minerals

In the differential flotation of sulfide and oxide minerals, Pb sulfide (PbS) was floated first, followed by Zn sulfide (ZnS), Pb oxide, and finally Zn oxide. Thus, flotation reagents were added in four different steps for the sample with a d_{80} of about 90 µm.

For Pb sulfide flotation, 50 g/t of potassium ethyl xanthate (PAX) was used as a collector, along with 300 g/t of $ZnSO_4$ and 50 g/t of sodium cyanide to depress sphalerite and pyrite. Finally, 1 drop of Rc250 frother was used and froth collection was conducted for 90 seconds.

In the second stage, the conditions for Zn sulfide flotation included 40 g/t of PAX, 200 g/t of copper sulfate activator, 1 drop of Rc250 frother, and 2 minutes of froth collection.

In the third stage (aimed at floating oxide Pb minerals), 4000 g/t of sodium sulfide ($Na_2S.9H_2O$), 200 g/t of PAX, 1 drop of MIBC frother, and a froth collection time of 3 minutes were used.

Finally, for the flotation of oxide Zn minerals, after desliming at 10 µm, 4000 g/t of $Na_2S.9H_2O$, 140 g/t of Armac C, 110 g/t of PAX, and 12 minutes of froth collection were used. Armac C and PAX were added in several stages.

Flotation of mixed sulfide-oxide Pb and Zn minerals

In the simultaneous flotation tests of mixed sulphide-oxide Pb and Zn minerals, flotation reagents were added in the following sequence. During the flotation of mixed sulphide-oxide Pb minerals, 4000 to 6000 g/t of sodium sulphide was used in the rougher stage, and 0 to 1000 g/t of sodium sulphide was used in the scavenger stage. In the rougher stage, 500 g/t of PAX collector and one drop of MIBC frother were added, and froth collection lasted for 3 minutes. In the scavenger stage, 100 g/t of PAX collector was used. For the rougher-scavenger flotation of mixed sulphide-oxide Zn minerals, particles finer than 10 µm were deslimed beforehand. Subsequently, 5000 g/t of sodium sulphide, 1000 g/t of sodium silicate, 250 g/t of Armac C (in multiple stages), 50 g/t of MIBC, and 10 minutes of froth collection were employed. The impact of particle size was also examined at two d_{80} levels of 75 and 90 µm. It should be emphasized that the flotation conditions mentioned above were used under optimal conditions.

Heavy liquid experiments

To examine the feasibility of gravity concentration of Pb and Zn minerals in the sample,

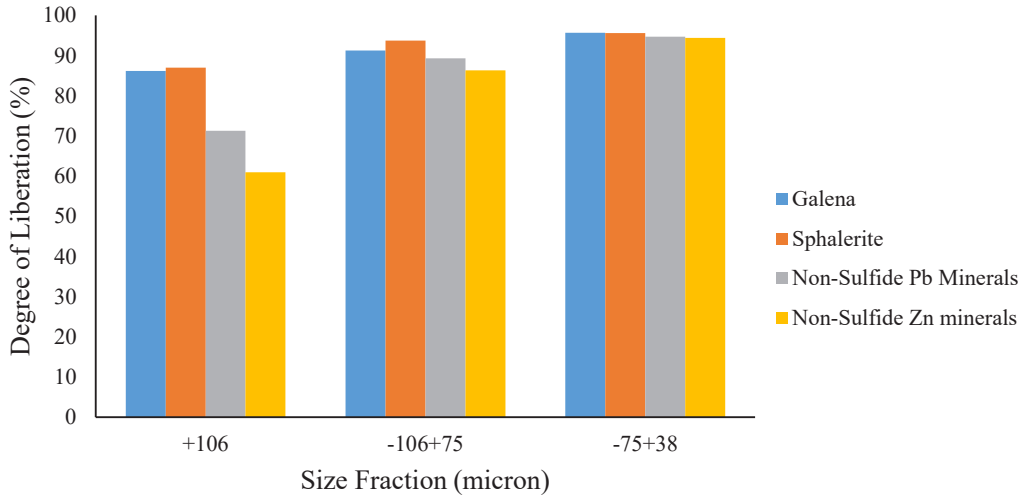

Fig. 3. Results of degree of liberation studies of Pb and Zn minerals.

considering the density range of target and gangue minerals, bromoform heavy liquid with a density of 2.89 g/cm³ was used as the heavy medium. To prepare the necessary samples for heavy liquid tests, the representative sample was crushed using jaw and roller crushers to a size of up to 3.35 mm. The sample was then classified into the following size fractions: -3350+2000 µm, -2000+850 µm, -850+106 µm, and -106 µm.

RESULTS AND DISCUSSION

As discussed in the ore characterization section (2.1.1.), 75.13% of the total Pb and 85.05% of the total Zn in the sample are in the form of oxide minerals. Therefore, the flotation method (both differential and cumulative) was employed to produce separate Pb and Zn concentrates. Additionally, the feasibility of gravity separation of Pb and Zn minerals was investigated using bromoform heavy liquid with a density of 2.89 g/cm³.

Concentration using the flotation method

Differential flotation of sulfide and oxide minerals

In the differential flotation method, PbS was floated first, followed by ZnS, then oxide Pb minerals, and finally oxide Zn minerals. In this method, due to the very small volume of the PbS concentrate collected, the sulfide-oxide concentrate was combined and analyzed. The results of the differential flotation tests (Fig. 4) indicated that the recovery of Pb and Zn in the Pb rougher concentrate was 53.7% and 4.41%, respectively. The grades of Pb and Zn in the Pb rougher concentrate were 19.1% (including 15.74% oxide) and 2.5% (including 2.03% oxide), respectively.

The recovery of Zn and Pb in the ZnS concentrate was 3.04% and 3.41%, respectively, indicating that the flotation of the Zn sulfide phase was not successful. The grades of the Zn and Pb in the Zn sulfide concentrate were 5.27% and 3.71%, respectively, suggesting that producing a separate

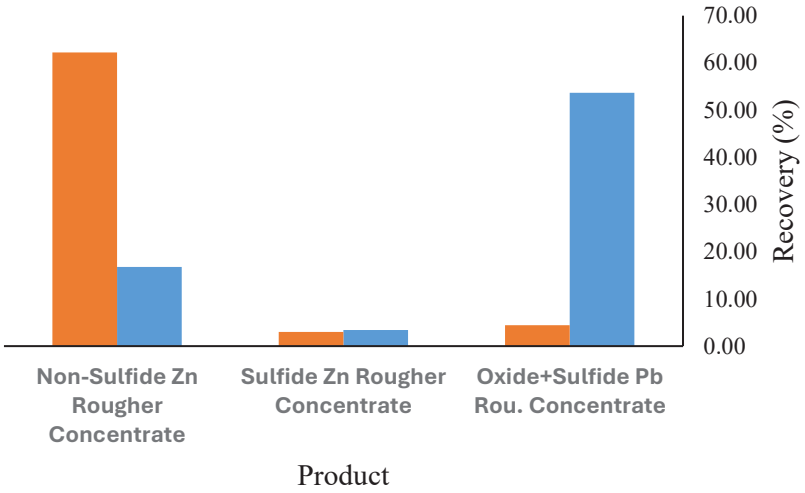

Fig. 4. The results of differential flotation tests of sulphide and oxide Pb and Zn minerals.

Table 2. Effect of sodium sulfide dosage on the grade and recovery of Pb and Zn in the rougher and scavenger flotation stages.

Sodium Sulfide Addition (g/t)		Grade (%)		Recovery (%)	
Rougher	Scavenger	Pb	Zn	Pb	Zn
4000	1000	6.86	2.75	68.99	17.64
4000	0	9.26	2.94	64	11.84
6000	1000	8.73	2.5	70.22	9.89

Zn sulfide concentrate is not feasible.

The recovery of Zn and Pb in the Zn oxide concentrate was 62.21% and 16.80%, respectively. This indicates that a significant portion of the Pb minerals were recovered in the oxide Zn concentrate, suggesting a need for more sodium sulfide during flotation. The grades of Zn and Pb in the oxide Zn concentrate were 9.67% (8.74% oxide) and 1.64% (1.21% oxide), respectively.

The differential flotation test of sulfide and oxide minerals revealed that, due to the small volume of sulfide minerals compared to oxide minerals, producing separate sulfide and oxide concentrates for Pb and Zn is not practically feasible. Therefore, the differential flotation method is not practical for this ore.

Cumulative flotation of mixed sulfide-oxide minerals

The possibility of cumulative flotation of mixed sulfide-oxide minerals was investigated for both Pb and Zn. Given the importance of sodium sulfide consumption and the particle size of the flotation feed, the effects of these two factors were examined in the final flotation experiments.

Effect of sodium sulfide activator dosage

The effect of sodium sulfide dosage on the grade and recovery of Pb and Zn in the rougher and scavenger flotation stages of sulfide-oxide Pb minerals are presented in Table 2. According to these results, increasing the amount of sodium sulfide from 4000 to 6000 g/t in the rougher stage did not significantly increase the Pb recovery, which only increased from 68.99% to 70.22%. Additionally, not adding sodium sulfide in the scavenger stage caused a 5% decrease in the Pb recovery.

Effect of particle size

The effect of particle size on the grade and recovery of Pb and Zn in the rougher and scavenger stages of the cumulative flotation test is presented in Table 3. The degree of freedom study in Fig. 3 showed that sphalerite and galena minerals are free even in the coarse fractions, but the degree

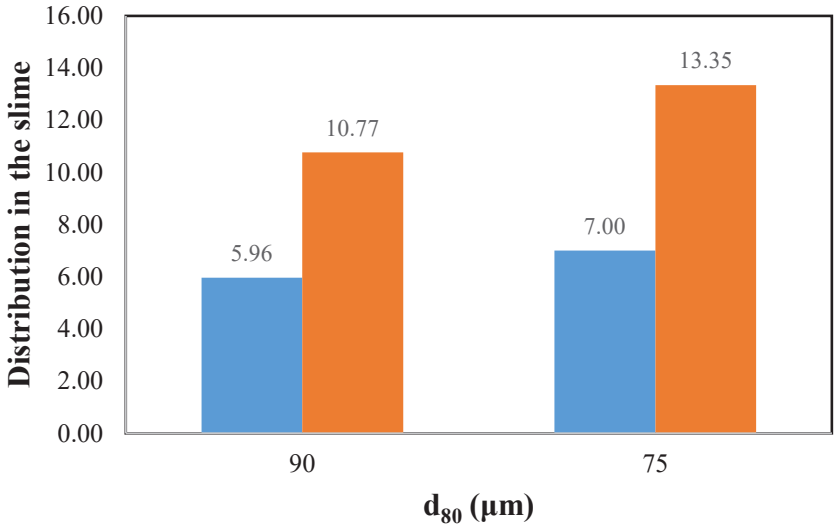

Fig. 5. The amount of Pb and Zn in the slime fraction.

Table 3. Effect of particle size (d_{80}) on the grade and recovery of Pb and Zn in the rougher and scavenger flotation stages.

d80 (micron)	Grade (%)		Recovery (%)	
	Pb	Zn	Pb	Zn
90	8.73	2.5	70.22	9.89
75	11.2	2.5	66.64	8.81

of freedom of Pb and Zn oxide minerals increases with a decrease in particle size. Therefore, it was expected that reducing the d_{80} would increase Pb recovery due to the increase in the free surface area of the particles. However, according to Table 3, reducing the particle size from 90 to 75 µm caused a decrease in Pb recovery from 70.22% to 66.64% and an increase in Pb grade from 8.73% to 11.2%.

Reducing the particle size caused an increase in the amount of slime (resulting from the desliming of particles finer than 10 µm) from 13.88% to 14.93%. At a d_{80} of 90 µm, the loss of Pb in the slime increased from 5.96% to 7%, and the loss of Zn in the slime fraction increased from 10.77% to 13.35% compared to a d_{80} of 75 µm (Fig. 5). Therefore, it can be said that the loss of particles in the slime is one of the factors reducing Pb recovery. Also, with the reduction of particle size, the recovery decreased due to higher reagent consumption by slime particles. The decrease in particle size increased the fraction with dimensions smaller than 10 µm, and by desliming, more gangue particles were removed, increasing the concentrate grade and, consequently, decreasing recovery.

The only advantage of further grinding the sample was the reduction of Zn recovery in the Pb concentrate from 9.89% to 8.81%. Therefore, a d_{80} of 90 µm is more suitable for the flotation of ores with a higher amount of oxide minerals.

The number of cleaner stages in both differential and cumulative flotation methods

The first and second cleaner stages were carried out using both differential and cumulative flotation methods to examine the possibility of increasing the Pb concentrate grade to an

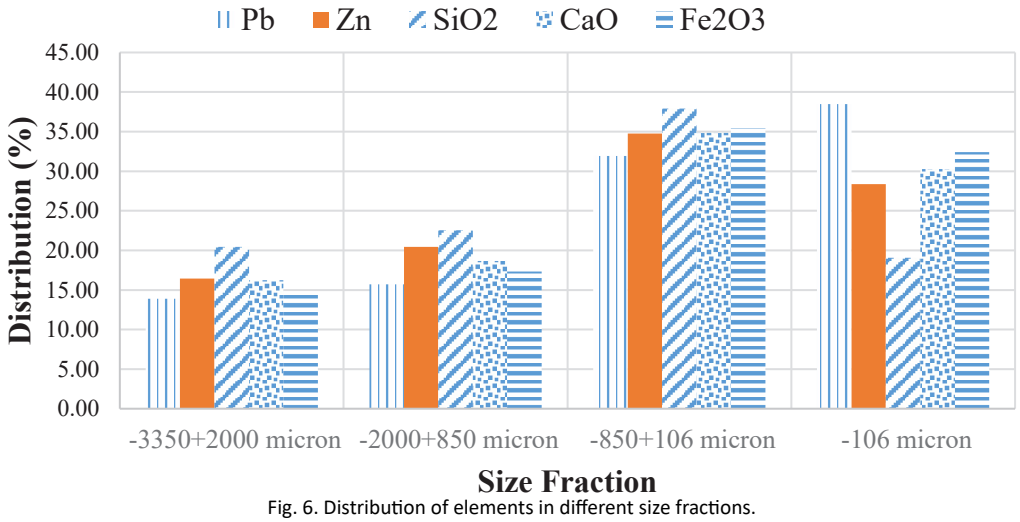

Fig. 6. Distribution of elements in different size fractions.

Table 4. Factors affecting lead grade in the cleaner stages.

Flotation Method	Sulfide -Oxide differential Flotation		Mixed Oxide-Sulfide Flotation							
Sodium Sulfide addition (Rougher+Scavenger)	4000+1000		4000+1000		4000+0		6000+1000		6000+1000	
d80 (micron)	90		90		90		90		75	
Percent	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
Rougher+Scavenger Concentrate	19.10	53.70	6.86	68.99	9.26	64.00	8.73	70.22	11.20	66.64
1st Cleaner Concentrate	56.40	47.39	34.91	55.97	40.00	53.99	32.9	59.2	38.00	58.5
2nd Cleaner Concentrate	-	-	-	-	-	-	55.20	53.76	58.60	53.05

acceptable level (at least 50%). According to Table 4, in the differential flotation method, the grade of the rougher concentrate was 19.1%, which increased to 56.4% in the first cleaner stage. However, under these conditions, the recovery of Pb in the rougher and cleaner stages was 53.7% and 47.39%, respectively.

A comparison of the results of the differential and cumulative flotation methods showed that the recovery in the rougher and scavenger stages using cumulative flotation was 68.99%, whereas it was 53.7% for the differential flotation method. The addition of sodium sulfide in the cumulative flotation of mixed sulfide-oxide Pb minerals clearly increased Pb recovery from 64% to 68.99%. However, the grade of the rougher-scavenger concentrate decreased from 9.26% to 6.86%.

Comparison of the results indicated that the best grade and recovery of Pb (for the d_{80} of 90 µm) were achieved by performing flotation of the oxide-sulfide mixture with two cleaner stages adding 6000 g/t of sodium sulfide in the rougher stage and 1000 g/t in the scavenger stage.

Although reducing particle size increased the grade of the rougher stage from 8.73% to 11.20%, the first cleaner stage from 32.9% to 38%, and the second cleaner stage from 55.2% to 58.6%, it also caused a decrease in recovery in the rougher stage from 70.22% to 66.64% due to higher reagent consumption. Meanwhile, when d_{80} was 75 µm, the final recovery in the second cleaner stage did not change significantly compared to the particle size of 90 µm.

Therefore, the optimal condition for the flotation of this ore is to use the cumulative flotation method with a d_{80} of 90 µm is to perform the rougher and scavenger stages with the addition of 6000 g/t of sodium sulfide in the rougher stage

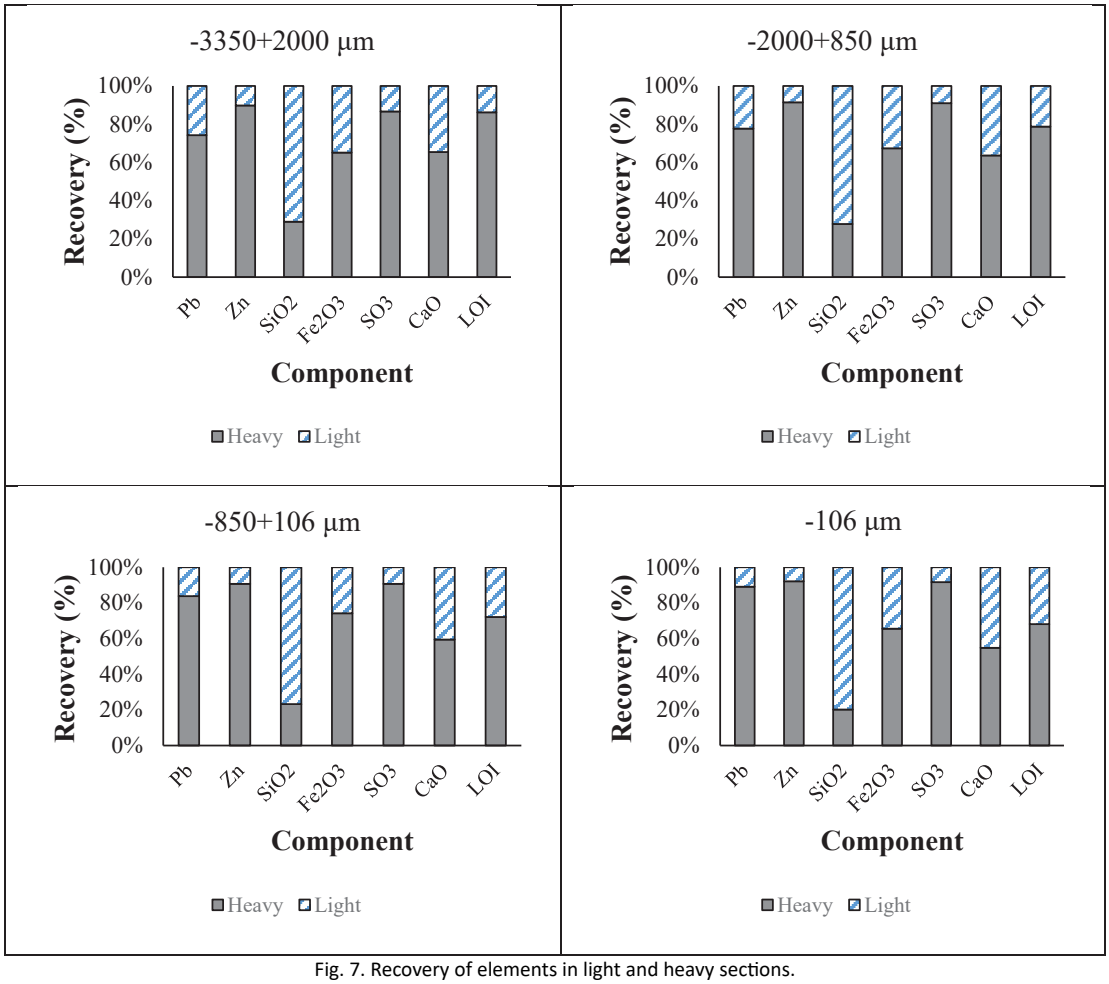

Fig. 7. Recovery of elements in light and heavy sections.

and 1000 g/t in the scavenger stage followed by two cleaner stages.

Heavy liquid tests

Gravity separation of Pb and Zn minerals from gangue minerals was examined using 98% bromoform heavy liquid with a density of 2.89 g/cm³ for four different size fractions: -3350+2000 µm, -2000+850 µm, -850+106 µm, and -106 µm. The results of the element distribution in these size fractions are shown in Fig. 6. The Pb recovery increased in the finer fractions. For Zn, as the particle size decreased, the distribution of Zn also decreased except for the 106 µm fraction.

The grade and recovery of Pb, Zn, and gangue minerals in the light and heavy fractions of the heavy liquid tests are shown in Table 5 and Fig. 7. In the coarse size fraction of -3.3 mm +2 mm, Zn grade increased from 2.41% to 5.43%, indicating a concentration factor of approximately 2.25 times. For this size fraction, the Pb grade increased from 1.1% to 2.05%, resulting in a concentration factor of 1.86. The recovery rates of Zn and Pb in the heavy fraction were 89.74% and 74.22%, respectively (Fig. 7).

According to Table 5, the concentration factor of Zn in the -2000 +850 µm size fraction was approximately 2.35, with the Zn grade increasing from 2.77% to 6.52%. Additionally, Fig. 7 shows that the recovery of Zn in the heavy fraction was 91.4%. The concentration factor of Pb in this size fraction was 2, with a recovery rate of 77.68%. Therefore, both Pb and Zn can be effectively concentrated using gravity separation in the -2000 +850 µm size fraction.

By reducing the particle size to the -850 +106 µm fraction, 90.75% of Zn entered the heavy fraction, and the Zn grade increased from 2.64% to 6.41%. As particle size decreases, Pb minerals tend to be crushed more, resulting in a higher Pb grade in this finer fraction compared to coarser fractions. The Pb grade increased from 1.31% to 2.93%, with a recovery rate of 83.73%.

The distribution results of Pb, Zn, and gangue minerals in the light and heavy sections using bromoform heavy liquid for gravity separation showed that Pb-containing minerals are more prone to crushing than other minerals. With decreasing particle size, the efficiency of gravity separation for Pb increased, with the concentration factor for Pb increased from 1.86 in the -3.3 mm +2 mm fraction to 2.5 in the -106 µm fraction. Similarly, the concentration factor for Zn increased from 2.25 in the -3.3 mm +2 mm fraction to 2.5 in the -106 µm fraction. Fig. 7 shows that the recovery of Zn in the heavy fraction did not change significantly with particle size reduction, but Pb recovery increased from 74.22% to 89.09%.

Table 5. Heavy liquid test results on different size fraction.

(-3350+2000 micron)	Wt.(%)	Pb	Zn	SiO_2	Fe_2O_3	SO_3	CaO	LOI
Heavy	39.76	2.05	5.43	53.16	8.57	0.69	9.52	7.63
Light	60.24	0.47	0.41	86.10	3.03	0.07	3.31	0.81
Feed	100.00	1.10	2.41	73.00	5.23	0.32	5.78	3.52
(-2000+850 micron)								
Heavy	38.86	2.30	6.52	53.43	9.83	0.95	10.10	9.73
Light	61.14	0.42	0.39	88.18	3.03	0.06	3.66	1.67
Feed	100.00	1.15	2.77	74.67	5.67	0.41	6.16	4.80
(-850+106 micron)								
Heavy	37.39	2.93	6.41	43.61	12.87	0.98	10.22	14.08
Light	62.61	0.34	0.39	86.15	2.68	0.06	4.17	3.26
Feed	100.00	1.31	2.64	70.25	6.49	0.40	6.43	7.31
(-106 micron)								
Heavy	35.71	6.54	9.23	33.08	18.09	1.96	14.23	19.69
Light	64.29	0.44	0.44	72.81	5.29	0.10	6.52	5.11
Feed	100.00	2.62	3.58	58.62	9.86	0.76	9.27	10.32

Considering these findings, it is recommended to perform gravity separation experiments on finer size fractions using devices such as shaking tables or spirals.

CONCLUSION

The objective of the present paper was to find an effective process plan for producing separate concentrates of Pb and Zn from a mixed oxide-sulfide Pb and Zn ore. Flotation experiments were conducted using two approaches: differential flotation of sulfide and oxide minerals and cumulative flotation of mixed sulfide-oxide minerals. Characterization studies showed that 75.13% of the total Pb and 85.05% of the total Zn in the sample were in the form of oxide minerals. The results of flotation experiments indicated that the differential flotation method is impractical for this ore. Subsequently, the effects of desliming, particle size reduction, and optimization of sodium sulfide dosage were examined in the cumulative flotation experiments. The findings revealed that reducing the particle size from a d_{80} of 90 to 75 µm decreased Pb recovery from 70.22% to 66.64% while increasing the Pb grade from 8.73% to 11.2%. Additionally, particle size reduction increased the weight of the slime fraction (finer than 10 µm) from 13.88% to 14.93%. Increasing the sodium sulfide dosage in the rougher stage did not significantly enhance Pb recovery. Furthermore, omitting sodium sulfide in the scavenger stage led to a 5% decrease in Pb recovery and an increase in Pb grade in the rougher-scavenger stage concentrate. The results showed that the cumulative flotation method for both sulfide and oxide minerals of Pb, followed by Zn, is practical for this ore. The optimal conditions for the flotation of this ore are to use the cumulative flotation method with a d_{80} of 90 µm and to perform the rougher and scavenger stages with the addition of 6000 g/t sodium sulfide in the rougher stage and 1000 g/t sodium sulfide in the scavenger stage along with two stages of cleaner flotation. Heavy liquid tests using bromoform with a density of 2.89 g/cm^3 were also conducted. The results of the sample distribution in the light and heavy sections indicated that the recovery of Zn in the heavy section did not change significantly with the reduction of particle size. However, the recovery of Pb increased from 74.22% to 89.09%. Additionally, reducing the particle size improved the efficiency of gravity separation for Pb. Therefore, it is recommended to perform gravity separation experiments on finer size fractions using equipment such as shaking tables and spirals.

CONFLICT OF INTEREST

The authors declare that there is no conflict of interests regarding the publication of this manuscript.

REFERENCES

1. Gilg HA, Boni M, Balassone G, Allen CR, Banks D, Moore F. Marble-hosted sulfide ores in the Angouran Zn-(Pb–Ag) deposit, NW Iran: interaction of sedimentary brines with a metamorphic core complex. Mineralium Deposita. 2006;41(1):1-16.
2. Bulatovic SM. Flotation of Mixed Lead Zinc Sulphide Oxide and Oxide Lead and Zinc Ores. Handbook of Flotation Reagents: Chemistry, Theory and Practice: Elsevier; 2010. p. 67-86.
3. Moradi S, Monhemius AJ. Mixed sulphide–oxide lead and zinc ores: Problems and solutions. Miner Eng. 2011;24(10):1062-1076.
4. Popli K, Afacan A, Liu Q, Prasad V. Development of online soft sensors and dynamic fundamental model-based process monitoring for complex sulfide ore flotation. Miner Eng. 2018;124:10-27.
5. Hajati A, Soltani F. Concentrating challenges of the Zarigan complex Pb-Zn-Fe non-sulfide ore. Can Metall Q. 2022;62(2):301-310.
6. Kursunoglu S, Kursunoglu N, Hussaini S, Kaya M. Selection of an appropriate acid type for the recovery of zinc from a flotation tailing by the analytic hierarchy process. Journal of Cleaner Production. 2021;283:124659.
7. Yanbo S, Xin T. Study of New Process Technology for Low-grade Refractory Zinc Oxide Ore. Procedia Environmental Sciences. 2016;31:195-203.
8. Darban Z, Patel B, Gaur R, Shahabuddin S. Polyaniline/Sodium Alginate (Pani/Sa) Nanocomposites as an Efficient Adsorbent for Cr (Vi) Elimination. Elsevier BV; 2023.
9. Marketing of Bambaranut and Its Sustainabilty in Alkaleri Local Government Area of Bauchi State, Nigeria. Iconic Research and Engineering Journals. 2025;9(3).
10. Oyelola AO, Abdu DA, Victor AD, Igonwelundu MT, Bosan BM, Oyebode AB. Extraction of a low grade zinc ore using gravity and froth flotation methods. Journal of Applied Sciences and Environmental Management. 2017;20(4):903.
11. Zhang P-y, Ou L-m, Zeng L-m, Zhou W-g, Fu H-t. MLA-based sphalerite flotation optimization: Two-stage roughing. Powder Technol. 2019;343:586-594.
12. Soltani F, Koleini SMJ, Abdollahy M. Optimization of Cu–Zn Massive Sulphide Flotation by Selective Reagents. Journal of The Institution of Engineers (India): Series D. 2014;95(2):125-134.
13. Kara Y, Tatar A, Hasyamanlar A. Eti Copper Siirt flotation plant revision studies. Physicochemical Problems of Mineral Processing. 2023.
14. Wei Q, Dong L, Qin W, Jiao F, Qi Z, Feng C, et al. Efficient flotation recovery of lead and zinc from refractory lead-zinc ores under low alkaline conditions. Geochemistry. 2021;81(4):125769.