

Article

Study of Oxidative–Reductive Potential Changes in the Enrichment of Oxidized Polymetallic Ores

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Abstract: This paper presents an analysis of the current state of processing lead–zinc ores from the Koskudyk deposit (Kazakhstan). At present, polymetallic ores are being extracted from the Ridder-Sokolnoye, Zyryanovskoye, Maleevskoye, and Achisai deposits. However, the reserves of rich and easily beneficiable ores are being depleted, and the supply of raw materials from the developed deposits does not exceed 25 years. As a result, more complex and difficult-to-enrich oxidized and mixed ores are being involved in production, and the extraction of non-ferrous metals from these ores presents a significant technological challenge. The most effective method for enriching oxidized polymetallic ores is flotation with preliminary sulfidization. Laboratory studies were conducted on a sample of oxidized lead–zinc ore from the Koskudyk deposit, which contains 79.69% oxidized lead compounds and 84.72% oxidized zinc compounds. This study examines the effect of sulfidization using sodium sulfide and determines the oxidative–reductive potential (ORP) levels for various reagent dosages. The experiments demonstrated that a sodium sulfide dosage of 700 g/t at an ORP of -200 mV yields the most effective lead flotation, resulting in a lead recovery of 50.07%. Zinc recovery remained relatively unchanged across all tests, confirming the limited response of oxidized zinc minerals under the applied sulfidization conditions. The highest beneficiation efficiency was achieved within the ORP range of -160 to -200 mV, beyond which lead recovery began to decline. The findings underscore the importance of optimizing ORP to ensure the formation of a stable sulfide film on mineral surfaces and efficient collector attachment. These results provide practical guidance for improving flotation performance of oxidized ores and demonstrate the need for additional activation strategies in zinc recovery.

Keywords: flotation; lead-zinc ores; oxidized polymetallic ores; sulfidization; ORP



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1. Introduction

The mining and metallurgical complex plays a vital role in the global economy and, in particular, in Kazakhstan, serving as the cornerstone of the national economy. At present, zinc and lead are key metals that play an important role both in Kazakhstan and in the global industry, the metallurgical sector, and the development of the circular economy. Kazakhstan is among the top ten world leaders in the extraction and production of mineral raw materials, including polymetallic ores. Approximately 16% of the world’s lead reserves and 10% of the world’s zinc reserves are concentrated in Kazakhstan’s subsoil, which gives the country a strategic advantage in the international market [1].

The major polymetallic ore deposits, such as Ridder-Sokolnoye, Zyryanovskoye, Maleevskoye, and Achisai, contain rich ores that are actively developed and processed. However, the reserves of rich and easily beneficiable ores are being depleted, and the supply of reserves from the developed deposits does not exceed 25 years. The prospects for the discovery of new lead and zinc deposits with high concentrations of valuable components remain relevant for nearly all regions of the republic. In recent years, the Shymerden deposit in Northern Kazakhstan, which is characterized by exceptionally high zinc content, has been involved in development. According to literature sources, 41% of lead–zinc deposits in Kazakhstan are concentrated in the Central region, 33% in the Eastern region, and 21% in Southern Kazakhstan. Promising areas for discovering new deposits include Central and Southern Kazakhstan, the Rudny Altai, and border areas with Russia and China [2].

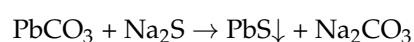
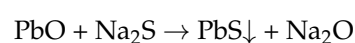
Currently, increasingly complex and difficult-to-enrich oxidized and mixed ores are being introduced into production, and extracting non-ferrous metals from them presents a technological challenge.

Oxidized polymetallic ores are complex ores containing metals such as lead, zinc, copper, and precious metals. They are formed as a result of the weathering of sulfide ores and are characterized by a high content of oxide minerals. Difficult-to-enrich ores represent a significant challenge in ensuring a high level of valuable component extraction and comprehensive utilization. The involvement of oxidized polymetallic ores in processing is an urgent task for the mining and metallurgical industry. These ores are characterized by a complex mineral composition and low reactivity of their valuable components, requiring the development of specialized technologies. To process oxidized polymetallic ores, it is necessary to integrate modern technologies (hydrometallurgy, autoclave leaching, and bioleaching), considering the mineral composition and availability of the ores. This will enable the efficient use of resources, reduce environmental damage, and ensure the sustainable development of the mining and metallurgical industry [2–11].

A method for enriching oxidized polymetallic ores is flotation after preliminary sulfidization. Sulfidization of oxidized minerals is a process of chemically modifying the surface of minerals, wherein their oxidized compounds are converted into sulfides. This method is used to enhance the hydrophobicity of minerals, improve their flotation properties, and increase the extraction of non-ferrous metals from ores. The main goal of sulfidization is to convert the oxidized forms of metals (e.g., carbonates, sulfates, and oxides) into sulfide forms, which are more easily floated. This is especially important for difficult-to-enrich oxidized and mixed ores containing metals such as copper, lead, or zinc.

Sulfidization occurs in several stages [12,13]:

1. Treatment of minerals with sulfur-containing reagents:
 - Minerals are treated with sulfidizing agents such as sodium sulfide (Na_2S), sodium or ammonium polysulfides, hydrogen sulfide (H_2S), or other compounds that can interact with the oxidized surfaces.
2. Sulfidization reaction:
 - The reagent interacts with the surface of oxidized minerals, forming a sulfide film on the surface. For example:



- As a result, the oxide and carbonate compounds of lead are converted into lead sulfide.
3. Fixation of the sulfide film:

- The sulfide film formed on the surface of the minerals improves their hydrophobicity, which facilitates the attachment of flotation reagents and air bubbles during flotation.
4. Flotation:
- The modified sulfide minerals are extracted by flotation using xanthates or other collector reagents that effectively interact with the sulfide surface.

It has also been proven that the stabilization of the electrochemical state of the dispersed system before flotation is necessary, controlled by measuring the oxidative–reductive potential (ORP) of the pulp. The oxidation–reduction potential (ORP) is a critical indicator reflecting the ratio between oxidized and reduced species in the pulp’s liquid phase. This parameter characterizes the prevailing direction of chemical processes in the system, whether oxidation or reduction. ORP significantly influences the ion–molecular composition of the liquid phase, altering it due to the occurrence of oxidation; reduction reactions. The ORP value is measured in volts or millivolts and serves as an indicator of the oxidative or reductive activity of the medium. A positive value indicates a tendency toward oxidation, while a negative value signals the predominance of reduction processes. Thus, ORP serves as an important additional factor determining the chemical behavior of the liquid phase of the pulp and the direction of the reactions occurring [8–10].

The regulation of oxidation–reduction potential (ORP) is a key factor in the enrichment of oxidized polymetallic ores. A comprehensive study of ORP changes during flotation, leaching, and precipitation processes allows the development of effective enrichment technologies and enhances the extraction of valuable metals. Proper understanding and control of ORP help optimize the selectivity of separation and increase the efficiency of extracting target components [13–17].

According to the studies by Oserov T.B., Li J., and others, sulfidization primarily occurs on the surface layer of the mineral, forming a thin sulfide film. The formation of this sulfide film significantly increases the hydrophobicity of the mineral. However, excessive use of the Na_2S reagent leads to the formation of a loose colloidal sulfide film, which easily breaks down and reduces the effectiveness of the sulfidization process. At high reagent dosages, the sulfidizer begins to act as a depressant, even for oxidized minerals [13–15]. In the work of Smaylov B., dedicated to the processing of refractory polymetallic ores, special attention is given to the flotation of oxidized forms of lead and zinc after sulfidization. It is noted that, when processing oxidized minerals, higher consumption of sodium sulfite and strict control of the redox potential (ORP) of the pulp during flotation are required. Monitoring the ORP helps avoid incomplete sulfidization or, conversely, excessive sulfidization, both of which can lead to losses in recovery [16].

In the studies by Kalichini M. et al., the flotation of oxidized copper ore was examined with the use of redox potential (ORP) control. The authors noted that controlling the optimal pulp potential significantly improved the recovery of oxidized minerals. The work confirms that maintaining a set Eh (approximately -300 mV on the Ag/AgCl electrode) during the addition of NaHS enhanced the activation of oxidized copper minerals and improved flotation efficiency compared to uncontrolled addition of the sulfidizer [17].

Based on the above, it follows that determining sodium sulfite consumption alone does not provide a comprehensive assessment of the flotation of oxidized minerals. Therefore, for the regulation and control of oxidized ores, it is necessary to know the exact level of redox potential (ORP). This is because, during the processing of oxidized ores, it is difficult to maintain consistent ore parameters. The redox potential integrally characterizes the ratio of oxidizing and reducing components in the pulp and is directly related to the presence of active sulfide ions. The flotation of oxidized ores is influenced not only by the content of the valuable minerals in the ore but also by the ore’s phase composition and the ratio of

oxidized sulfide minerals, which, in turn, significantly affect the consumption of sodium sulfite. Therefore, for successful flotation after sulfidization, it is important to maintain the potential within a specific optimal range, where both the stability of the sulfide film and the collector's ability to strongly adhere to the surface are ensured. According to reviews, the combined use of pH and ORP control is a promising flotation management method that deserves special attention in industrial conditions.

The aim of this study is to investigate the effect of sulfidization on the enrichment processes of oxidized polymetallic ores from the Koskudyk deposit, determine the optimal conditions for converting oxidized minerals into sulfide forms, improve the flotation properties of ores, and enhance the extraction of valuable components, such as lead and zinc. The study also aims to determine the optimal ORP values to ensure the hydrophobization of the surface of oxidized minerals [8–12].

2. Materials and Methods

The object of study is the oxidized lead–zinc ores from the Koskudyk deposit in Kazakhstan.

The material composition of the ore and enrichment products was determined using the group trial gravimetric analysis method in representative samples specially prepared for atomic absorption analysis, atomic emission analysis, classical chemical analysis, and photometric analysis. The phase composition was determined using a D2 Phaser diffractometer from Bruker. The study of ore minerals was conducted in reflected light on polished thin sections using the OLYMPUS BX 53 microscope (Olympus Corporation, Japan), the SIMAGIS XS-3CU (Olympus Corporation, Tokyo, Japan) video camera, and Mineral C7 (Siams, Yekaterinburg, Russia) image analysis software from SIAMS.

For the research, a sample of ore from the Koskudyk deposit was delivered to the laboratory. After natural air drying, the sample was crushed to a size of 2 mm, homogenized using the 'cone and ring' method, and analyzed for the content of major components according to particle size classes.

According to Table 1, the ore sample contains 0.85% lead, 0.65% zinc, 5.4 g/t silver, and 0.6 g/t gold.

Table 1. Chemical composition of polymetallic ore.

Component	Cu	Pb	Zn	Fe	CaO	MgO	As
Mass Fraction, %	0.02	0.90	0.70	6.42	1.33	3.35	0.023
Component	SiO ₂	Al ₂ O ₃	Au, g/ton	Au, g/ton	Na	S total	K
Mass Fraction, %	55.23	26.34	0.51	6.1	0.48	0.17	3.1

According to the data of the phase chemical analysis, the sample is classified as the oxidized type. This is confirmed by the fact that oxidized compounds make up 79.69% of the total lead mass, while for zinc, this value reaches 84.72%.

The X-ray phase composition of the ore sample is shown in Figure 1, and the data in numerical form are presented in Table 2.

The ore minerals in the sample are represented by dull ores, chalcocite, covellite, goethite, and hematite. Sulfides present include chalcopyrite, galena, pyrite, and sphalerite.

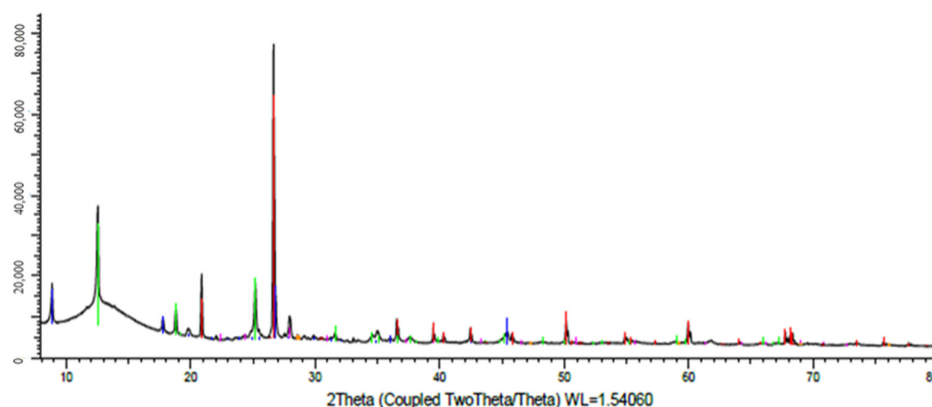


Figure 1. Results of the X-ray phase composition of the ore sample.

Table 2. Results of the X-ray phase composition.

Mineral/Phase Group	Content (%)
Sulfide minerals (total):	0.37
Galena (PbS)	0.16
Sphalerite (ZnS)	0.12
Copper sulfide minerals (e.g., Cu ₂ S)	0.04
Pyrite (FeS ₂)	0.05
Oxidized phases (total):	6.47
Oxidized and residual lead compounds	0.89
Oxidized and insoluble zinc compounds	1.07
Iron oxides (goethite, hematite)	4.51
Silicate minerals (total):	84.39
Mica group minerals	53.44
Quartz (SiO ₂)	30.95
Other phases (calcite, etc.)	8.77
Total	100.0

Sphalerite contains fine inclusions of chalcopyrite (0.003–0.01 mm) and galena (0.005–0.05 mm) (Figure 2). Galena films with a thickness of 0.003–0.005 mm are observed along the contours of sphalerite grains, with inclusions of covellite (0.011–0.03 mm). Galena veins range in length from 0.01 to 0.30 mm. Along the contours of the sphalerite grains, there are galena films (0.003–0.010 mm) and covellite inclusions (0.03 mm). Rare sphalerite veins have a thickness of 0.18 mm. Occasional galena crystals measuring 0.15 mm and discontinuous galena veins with a thickness of 0.02 mm are also observed.

The oxidized ores of the Koskudyk deposit are characterized by a high degree of oxidation, fine inclusions, and specific structural–textural features, which classify them as finely disseminated and difficult-to-beneficiate ores.

The granulometric (particle size) composition of the crushed ore (prepared to <2 mm) was analyzed, and the distribution of zinc and lead by size class is presented in Table 3. The sample was sieved into size fractions from 2 mm to below 0.045 mm, and each fraction was weighed (yield% of total) and assayed for lead and zinc. The recovery of metals in each size class was then calculated.

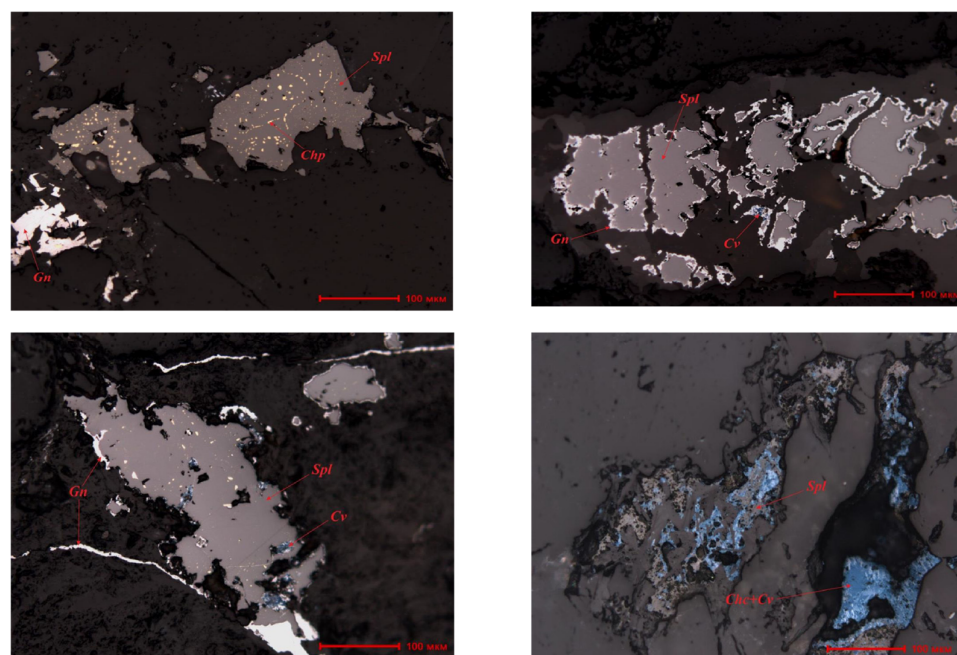


Figure 2. Characteristics of sphalerite and galena inclusions. Spl—sphalerite, Chp—chalcopyrite, Cv—covellite, Gn—galena.

Table 3. Granulometric composition of the ore sample (crushed to 2 mm), with zinc and lead distribution across size fractions.

Particle Size Class, mm	Yield, %	Content, %		Recovery, %	
		Zn	Pb	Zn	Pb
−2 + 1	39.78	0.58	0.65	32.67	28.59
−1 + 0.5	18.48	1.11	0.77	29.37	15.76
−0.5 + 0.2	14.73	0.49	0.50	10.29	8.23
−0.2 + 0.1	6.45	0.67	0.88	6.17	6.32
−0.1 + 0.071	1.99	1.10	1.80	3.12	3.99
−0.071 + 0.045	1.79	1.49	2.68	3.81	5.34
−0.045 + 0	16.78	0.61	1.70	14.57	31.77
The initial sample	100.0	0.70	0.90	100.0	100.0

As seen in Table 3, the ore’s valuable components are unevenly distributed across size fractions. A substantial proportion of lead (about 32% of Pb) and zinc (~15% of Zn) are found in the finest fraction (−0.045 mm), even though that fraction represents only ~16.8% of the mass. The coarsest fraction (−2 + 1 mm) contains a large share of both metals (around 28.6% of Pb and 32.7% of Zn in about 39.8% of the mass). The intermediate fractions contribute the rest. These data indicate that a significant amount of lead, in particular, is concentrated in the fines, which often consist of oxidized lead compounds liberated from the gangue upon crushing. The relatively low grades of Pb and Zn in all fractions (all below 2% Zn and 1% Pb) reflect the overall low tenor of the ore and the diluting effect of the abundant gangue.

The ore sample preparation for the study was carried out using the wet grinding method with a laboratory ball mill featuring a rotating axis (MShL-7; 40 ML, “Mekhanobrtsehnika”, St. Petersburg, Russia). For subsequent flotation tests, the optimal grinding fineness of −0.071 mm, with 70% passing, was determined. Flotation tests were conducted using tap water, and the results of the analysis are presented in Table 4.

Table 4. Chemical analysis of the tap water used in flotation tests.

Parameter	Content
Sodium and potassium, meq/L	4.80
Calcium, meq/L	6.40
Magnesium, meq/L	3.40
Chloride ion, meq/L	0.50
Sulfate ion, meq/L	8.81
Bicarbonate ion, meq/L	5.20
Nitrate ion, meq/L	0.09
Total hardness, °dH	9.80
Carbonate hardness, °dH	5.20
pH, units	7.50

The water analysis indicates moderately hard water (total hardness ~9.8 °dH, with calcium and magnesium present) and near-neutral pH.

Flotation concentration was carried out using standard laboratory mechanical flotation machines of the Mekhanobr type (FML-3) (St. Petersburg, Russia), with cell volumes of 3 and 1.5 L. The impeller diameter was 70 mm, the air intake rate through the impeller was 0.07 L/s, and the froth scraper rotation frequency was 0.25 s^{−1}. The ore pulp density in flotation was about 30% solids by weight (specific gravity of the ore ~2.70 g/cm³). The conditions for the laboratory tests are presented in Table 5.

Table 5. Conditions of laboratory tests.

Operation	Time (min)	Reagent Consumption (g/t)			
		Sodium Bisulfide	Thiocarbamate-Based Collector	Sodium Butyl Xanthate	Methyl Isobutyl Ketone (MIBK)
Total:		0–900	5	165	10
Grinding, −0.071 mm, 70%	10	-	-	-	-
Rougher flotation	10	0–900	5	120	5
Scavenger flotation	7	-	-	45	5

A mixture of thiocarbamate and xanthate was used as a collector due to the complex composition of the ore.

During the laboratory experiment, both the pH level and redox potential (ORP) were monitored using electronic potentiometric sensors: the pH was controlled with a HANNA HI 1230 (Hanna Instruments, Woonsocket, RI, USA) electrode (measurement range 0–14 units), and the ORP was measured using the HI 3131 (Hanna Instruments, USA) (measurement range ±399.9 mV). The ORP was recorded after the addition of reagents and before aeration began.

3. Results and Discussion

The principal technological flowchart of the laboratory experiment is shown in Figure 3.

Sodium sulfide was used to activate the oxidized minerals [13–15]. The results of the open laboratory experiments are presented in Table 6.

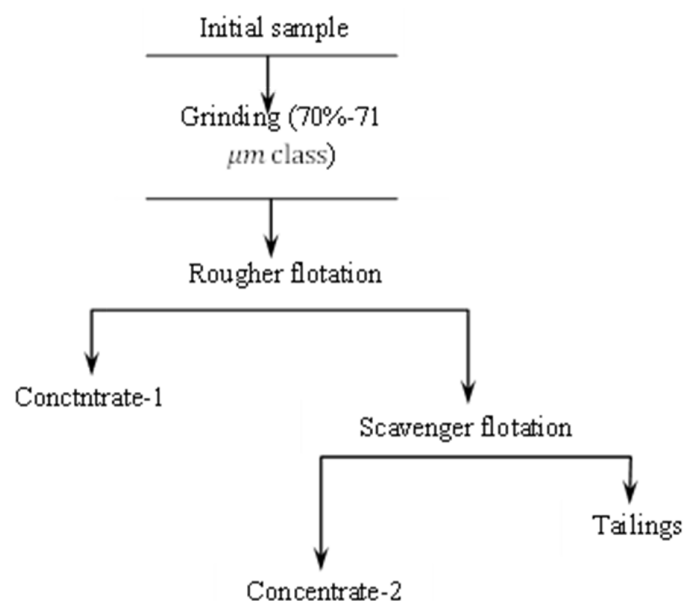


Figure 3. The principal technological flowchart of the laboratory experiment.

Table 6. Scheme of open experiments on test trials of the effect of sodium sulfide on the flotation process.

Consumption of Na ₂ S (g/t) and Redox Potential (mV)	Product	Yield, %	Content, %		Recovery, %	
			Zn	Pb	Zn	Pb
Test 1						
0 (100)	Concentrate-1	3.45	1.92	7.43	9.46	28.47
	Concentrate-2	0.98	2.34	8.28	3.28	9.02
	Σ Concentrate	4.43	2.01	7.62	12.74	37.48
	Tailings	95.57	0.64	0.59	87.26	62.52
	Total	100.0	0.70	0.90	100.0	100.0
Test 2						
500 (−120)	Concentrate-1	3.76	1.74	9.48	9.34	39.60
	Concentrate-2	1.09	2.08	5.81	3.24	7.03
	Σ Concentrate	4.85	1.82	8.65	12.58	46.63
	Tailings	95.15	0.64	0.51	87.42	53.37
	Total	100.0	0.70	0.90	100.0	100.0
Test 3						
600 (−160)	Concentrate-1	4.02	1.63	9.13	9.36	40.76
	Concentrate-2	1.19	1.95	5.21	3.31	6.89
	Σ Concentrate	5.21	1.70	8.23	12.67	47.65
	Tailings	94.79	0.64	0.50	87.33	52.35
	Total	100.0	0.70	0.90	100.0	100.0
Test 4						
700 (−200)	Concentrate-1	4.12	1.39	9.18	8.18	42.03
	Concentrate-2	1.28	1.87	5.65	3.42	8.03
	Σ Concentrate	5.40	1.50	8.34	11.60	50.07
	Tailings	94.60	0.65	0.48	88.40	49.93
	Total	100.0	0.70	0.90	100.0	100.0

Table 6. Cont.

Consumption of Na ₂ S (g/t) and Redox Potential (mV)	Product	Yield, %	Content, %		Recovery, %	
			Zn	Pb	Zn	Pb
Test 5						
800 (−260)	Concentrate-1	4.08	1.42	8.86	8.28	40.18
	Concentrate-2	1.30	1.75	5.44	3.24	7.86
	Σ Concentrate	5.38	1.50	8.03	11.52	48.04
	Tailings	94.62	0.66	0.49	88.48	51.96
	Total	100.0	0.70	0.90	100.0	100.0
Test 6						
900 (−310)	Concentrate-1	5.42	0.97	6.73	7.49	40.52
	Concentrate-2	1.06	1.97	5.15	2.98	6.06
	Σ Concentrate	6.48	1.13	6.47	10.47	46.58
	Tailings	93.52	0.67	0.51	89.53	53.42
	Total	100.0	0.70	0.90	100.0	100.0

The results of the beneficiation efficiency calculation using the Houcock-Luyken (or Hankok-Luyken) formula are presented in Figure 4.

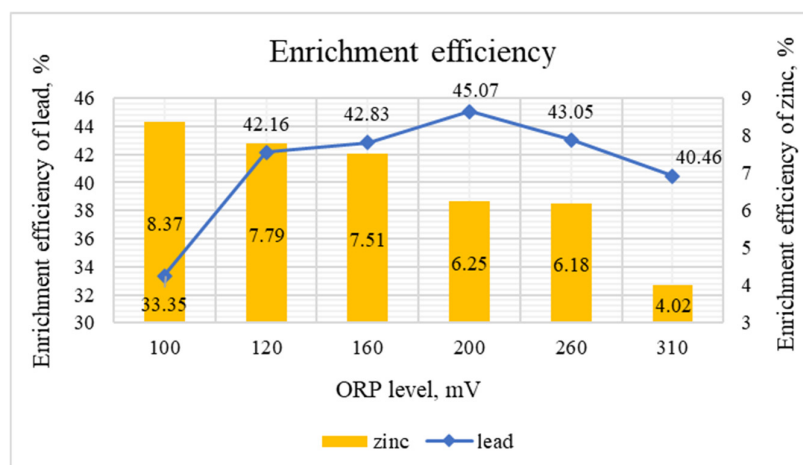


Figure 4. Results of the beneficiation efficiency calculation using the Houcock–Luyken (or Hankok–Luyken) formula.

The results from Table 1 show that without prior sulfidization, the flotation of lead is low, with a recovery of 37.48%. These results are comparable to data from the literature, which indicate the necessity of performing sulfidization.

As the dosage of Na₂S increased from 0 to 700 g/t (Tests 1 through 4), the recovery of lead steadily rose. The highest lead recovery was observed in Test 4 with 700 g/t Na₂S: about 50.07% of the lead was recovered. This is a significant improvement over Test 1's 37.48% and indicates that sulfidization successfully rendered a substantial portion of the oxidized lead minerals floatable. The ORP measured in Test 4 was around −200 mV (Ag/AgCl reference). This ORP level (approximately −200 mV) appears to be optimal for maximizing lead flotation in our experiments. It is in good agreement with values reported in other studies for effective sulfidization–flotation of oxidized lead minerals [9,12,18–21]. Maintaining the pulp potential in a moderately reducing range likely ensures the formation of a stable PbS film on the particle surfaces without causing excessive colloidal sulfur.

It should be noted that sulfidization does not significantly improve the enrichment results for zinc. It is known that additional measures are required for the flotation of oxidized zinc, such as the following [17,22]:

- Activation of zinc minerals using copper sulfate after sulfidization.
- Flotation with other collectors—for example, fatty acids are often used for carbonate zinc minerals.
- Hydrometallurgy of zinc.

Besides pulp sulfidization, there are combined methods aimed at improving the flotation of oxidized ores. These methods include [9,22,23] the following:

- Mechanochemical sulfidization—grinding oxidized ore with elemental sulfur.
- Sulfidization roasting—roasting oxidized concentrates with the addition of sulfidizing agents followed by flotation.
- Electrochemical sulfidization.

According to current results, flotation of oxidized ores with sodium sulfide enables the extraction of approximately 50% of the lead content in a single stage. Chinese researchers achieved a lead concentrate recovery of 57% by employing microwave roasting with pyrite followed by flotation.

Chepushtanova T.A. and colleagues proposed a method of sulfidization roasting followed by flotation. Antropova I.G. and colleagues conducted laboratory studies on joint roasting of oxidized lead–zinc ores with pyrite-containing ores [22,23].

Comparing these methods, it should be noted that chemical sulfidization in pulp (our case) offers advantages such as simplicity and low energy consumption, but it is limited to a recovery rate of around 50%. Combined methods (mechanochemical and thermal) potentially achieve more complete conversion of oxidized minerals to sulfides and higher recoveries but require additional processing stages (grinding with sulfur and/or roasting), thus complicating the technology. Hydrometallurgical methods (leaching zinc from tailings) can supplement flotation, providing nearly complete recovery of lead and zinc. However, this divides the process into two distinct branches—flotation for lead and hydrometallurgy for zinc.

In practice, our findings suggest that maintaining an appropriate redox environment is as important as the reagent dosage itself. Merely adding a sulfidizing agent is not enough; one must ensure that the pulp's ORP is brought into the right range and kept there during flotation. In industrial applications, ORP control could be achieved by automated dosing of Na₂S or other sulfidizers and real-time ORP measurement. Additionally, pH adjustment can influence ORP and sulfidization chemistry, so pH and ORP may need to be controlled together [14,16,24].

4. Conclusions

The oxidized ores of the Koskudyk deposit are characterized by a high degree of oxidation, fine dissemination, and specific structural–textural features, which classify them as finely disseminated and difficult-to-beneficiate ores.

The studied ore sample contains 0.85% lead, 0.65% zinc, 5.4 g/t silver, and 0.6 g/t gold.

According to the data from phase chemical analyses, the sample is classified as of the oxidized type. This is confirmed by the fact that oxidized compounds constitute 79.69% of the total lead mass, while for zinc, this value reaches 84.72%.

The ore minerals in the sample are represented by dull ores, chalcocite, covellite, goethite, and hematite. Sulfides present include chalcopyrite, galena, pyrite, and sphalerite.

Studies were carried out on the sulfidization of oxidized minerals with sodium sulfide, and the redox potential (ORP) levels were determined for each consumption rate. The high

efficiency of preliminary sulfidization of ore using Na_2S for the flotation of oxidized lead minerals has been experimentally confirmed. This approach significantly increases lead recovery compared to flotation without a sulfidizing agent ($\varepsilon = 37.48\%$).

It has been established that the redox potential (Eh) of the pulp significantly influences flotation performance. Maximum lead recovery ($\varepsilon = 50.07\%$) is achieved at a redox potential of approximately -160 to -200 mV. Under more strongly reducing conditions (Eh below -200 mV), the efficiency of lead recovery declines (to $\varepsilon = 48.04\%$). A redox potential of -200 mV corresponds to the optimal value reported in the literature. Maintaining this potential enhances the collectability of sulfide minerals and reduces the adverse effects of oxidized surface films on flotation efficiency. At the same time, changes in redox potential had virtually no effect on zinc recovery. Across all tested sulfidization regimes, zinc performance showed little to no improvement, confirming the limited effectiveness of this method in the beneficiation of oxidized zinc ores. As noted earlier, the flotation of oxidized zinc requires additional measures (e.g., the addition of auxiliary reagents, zinc hydrometallurgy, zinc activation, etc.).

The results of this study have substantial scientific and practical significance. They deepen the understanding of electrochemical factors—particularly redox potential levels—that affect the flotation efficiency of complex and refractory oxidized polymetallic ores. These findings can serve as a foundation for developing more effective technologies for processing such challenging raw materials. Practical implementation of the proposed approaches may improve the recovery of valuable metals and enhance the overall economic efficiency of beneficiation.

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