https://doi.org/10.51885/3134-7983_CATMSP_2026_2_5
SRSTI 52.45.19

IMPROVING THE QUALITY OF PB–ZN CONCENTRATE BY SODIUM SILICATE ADDITION DURING FLOTATION OF OXIDIZED ORE FROM THE KOSHKUDUK DEPOSIT

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Keywords:
oxidized lead–zinc ore; flotation; sodium silicate; calcium polysulfide; depressant; concentrate quality; Koshkuduk.

ABSTRACT

The processing of oxidized lead–zinc ores is complicated by the hydrophilic surface of valuable minerals, fine dissemination, and the significant presence of silicate and slime components, which reduce flotation selectivity and concentrate quality. This study aimed to improve the quality of a lead–zinc concentrate obtained from oxidized ore of the Koshkuduk deposit by optimizing the dosage of sodium silicate (Na_2SiO_3) during flotation after preliminary sulfidization with calcium polysulfide. Laboratory flotation tests were carried out using sodium silicate dosages ranging from 0 to 500 g/t. The process performance was evaluated based on concentrate yield, Pb, Zn, and Ag grades, recoveries, the Hancock–Luyken separation efficiency criterion, and an additional separation criterion. Without sodium silicate, the concentrate yield was 7.20 %, with Pb, Zn, and Ag grades of 3.78 %, 3.15 %, and 50.03 g/t, respectively. The optimal Na_2SiO_3 dosage was 170 g/t, at which the concentrate yield decreased to 3.62 %, while Pb, Zn, and Ag grades increased to 8.58 %, 6.44 %, and 100.46 g/t, respectively. The corresponding recoveries were 53.56 % Pb, 41.64 % Zn, and 66.12 % Ag. Further increasing the Na_2SiO_3 dosage to 200–500 g/t did not improve flotation performance. The results demonstrate that sodium silicate effectively improves concentrate quality by depressing silicate and slime minerals in oxidized lead–zinc ore flotation.

INTRODUCTION

The gradual depletion of high-grade sulfide ores has increased the need to involve oxidized and mixed lead–zinc ores in mineral processing. These ores are generally characterized by complex mineral composition, high oxidation degree of valuable minerals, fine dissemination, and a significant proportion of silicate, carbonate, and clay minerals (Nayak et al., 2022; Liu et al., 2020; Maghfouri et al., 2018). In contrast to sulfide minerals, oxidized lead and zinc minerals have more hydrophilic surfaces, which makes their direct flotation difficult and reduces the selectivity of separation (Yi et al., 2022; Elizondo-Álvarez et al., 2020; Xue et al., 2023; Zhang et al., 2022).

One of the main technological problems in the flotation of oxidized lead–zinc ores is the production of a concentrate with acceptable quality. Even when the recovery of valuable components is sufficient, the concentrate may be contaminated by gangue minerals, fine slimes, iron-bearing phases, and silicate components. This reduces the grade of lead, zinc, and associated precious metals in the concentrate and may negatively affect subsequent metallurgical processing (Zuo et al., 2020; Bai et al., 2020; Zheng et al., 2018; Ge et al., 2020).

A widely used approach to improve the floatability of oxidized minerals is preliminary sulfidization of their surface. During treatment with sulfidizing reagents, a sulfide-like layer can form on the surface of oxidized minerals, promoting the adsorption of sulfhydryl collectors and increasing particle hydrophobicity (Ke et al., 2018; Li et al., 2017; Li et al., 2014). However, the use of conventional sodium sulfide does not always provide stable results. Excess sulfide ions may depress valuable minerals, while the formed surface compounds may be unstable under flotation conditions (Luo et al., 2023; Zhang et al., 2022; Zhao et al., 2022).

In recent years, increasing attention has been paid to alternative sulfidizing reagents and combined reagent regimes. For oxidized lead–zinc ore from the Koshkuduk deposit, the effectiveness of calcium polysulfide (CaS_n) has previously been demonstrated. Calcium polysulfide provides more stable sulfidization of oxidized mineral surfaces and improves the conditions for collector adsorption (Mambetaliyeva et al., 2026). However, the selection of an effective sulfidizing agent alone is not always sufficient to obtain a high-quality concentrate, because a significant part of the loss of selectivity is associated with the entrainment of silicate and slime particles into the froth product.

To regulate flotation selectivity, depressants and dispersants of gangue minerals are widely used. Sodium silicate (Na_2SiO_3), also known as water glass, is one of the most common reagents used for depressing silicate, carbonate, and calcium-bearing minerals and for reducing the negative effect of fine slimes (Wang et al., 2024; Ni et al., 2025; Silva et al., 2012). In aqueous solutions, sodium silicate undergoes hydrolysis with the formation of silicate and polysilicic species that can adsorb on the surface of gangue minerals, increasing their hydrophilicity and preventing collector attachment (Wang et al., 2024; Silva et al., 2012). As a result, Na_2SiO_3 can reduce concentrate yield, decrease gangue contamination, and increase the grade of valuable components.

The use of sodium silicate is particularly important for complex oxidized ores in which valuable minerals are associated with quartz, aluminosilicates, carbonates, and iron-bearing phases. In such systems, Na_2SiO_3 may perform a dual function: it depresses gangue minerals and stabilizes the dispersed state of the pulp, reducing mechanical slime entrainment into the froth product (Wang et al., 2024; Ni et al., 2025; Silva et al., 2012; Marinakis & Shergold, 1985). However, the efficiency of sodium silicate strongly depends on its dosage. Insufficient reagent addition may not provide adequate gangue depression, whereas excessive dosages may reduce the floatability of sulfidized valuable minerals and decrease recovery.

Despite numerous studies on the sulfidization of oxidized lead–zinc minerals, the improvement of concentrate quality through sodium silicate dosage optimization in the presence of calcium polysulfide remains insufficiently studied. This issue is especially relevant for the Koshkuduk deposit, where the ore is characterized by a high degree of oxidation, fine mineral dissemination, and a considerable silicate component (Mambetaliyeva et al., 2026; Chepushtanova et al., 2022).

The scientific novelty of this study lies in the optimization of sodium silicate dosage for oxidized Pb–Zn ore from the Koshkuduk deposit after preliminary sulfidization with calcium polysulfide. In contrast to previous studies that mainly considered conventional sulfidization or the general depressing effect of sodium silicate on gangue minerals, this work evaluates the combined reagent regime of calcium polysulfide and Na_2SiO_3 under controlled flotation conditions. In addition, the study identifies the rational Na_2SiO_3 dosage and determines the dosage range at which further reagent addition does not improve concentrate quality and may partially depress valuable minerals.

The aim of this study is to investigate the effect of sodium silicate dosage on the flotation performance of oxidized lead–zinc ore from the Koshkuduk deposit after preliminary sulfidization with calcium polysulfide. Particular attention is paid to improving concentrate

quality, reducing froth product yield, increasing Pb, Zn, and Ag grades, and evaluating the recovery and separation efficiency of valuable components.

MATERIALS AND METHODS

The chemical composition of the oxidized polymetallic ore sample from the Koshkuduk deposit was determined using conventional silicate chemical analysis. The contents of gold and silver were measured by fire assay analysis, while the elemental composition of the sample was established by inductively coupled plasma mass spectrometry (ICP-MS). Phase composition was carried out in reflected light on polished sections using an OLYMPUS BX 53 microscope equipped with a SIMAGIS XS-3CU video camera and Mineral C7 image analysis software (SIAMS).

The chemical composition of the polymetallic ore sample is shown in Figure 1.

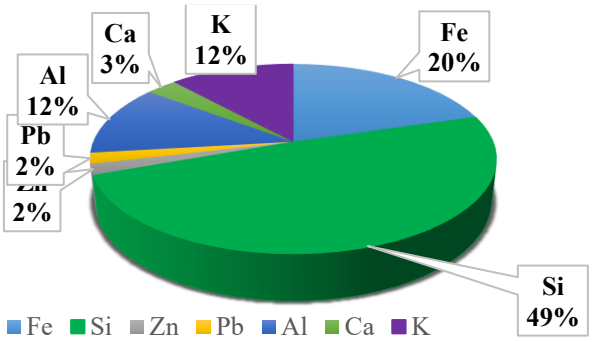


Figure 1. Chemical composition of the polymetallic ore

Note – compiled by the authors

As shown in Figure 1, the ore contains 0.58 wt.% Pb and 0.56 wt.% Zn. The copper content is low (0.02 wt.%) and does not represent industrial interest. At the same time, the ore contains a noticeable amount of precious metals, with gold at approximately 0.6 g/t and silver at 5.5 g/t.

To assess the processability of the ore, a chemical phase analysis of lead, zinc and iron was carried out. The results are presented in Table 1.

Table 1. Results of chemical phase analysis of the oxidized ore from the Koshkuduk deposit

Component forms	Content, wt.% (absolute)	Content, % (relative)
Zinc occurrence forms:		
Water-soluble	< 0.05	-
Oxidized	0.31	55.36
Sulfide	0.07	12.50
Sparingly soluble	0.18	32.14
Total Zn	0.56	100.0
Lead occurrence forms:		
Oxygen-containing	0.29	50.00
Galena	0.13	22.41
Residual forms	0.16	27.59
Total Pb	0.58	100.0
Iron occurrence forms:		
Sulfide	< 0.05	-
Ferrous iron (Fe^{2+})	3.06	48.6
Ferric iron (Fe^{3+})	3.24	51.4
Total Fe	6.30	100.0
Note – compiled by the authors		

The data in Table 1 indicate that the studied ore belongs to the oxidized type. Lead is present in oxidized and silicate forms at 77.59 % (relative). While zinc occurs predominantly in oxidized forms at 87.5 % (relative). Iron is also almost entirely oxidized, as the content of sulfide iron forms does not exceed 0.05 % (absolute).

Mineralogical analysis showed that the ore minerals are mainly represented by fahlores, chalcocite, covellite, goethite, and hematite. Sulfide minerals such as chalcopyrite, galena, pyrite and sphalerite are also present. The mineral aggregates from disseminated and vein-disseminated textures as well as replacement structures. A characteristic feature of the ore is the fine dissemination of minerals (0.005-0.015 mm) and the intensive development of oxidation and replacement processes, with sulfide minerals being replaced by goethite and hematite (Figure 2).

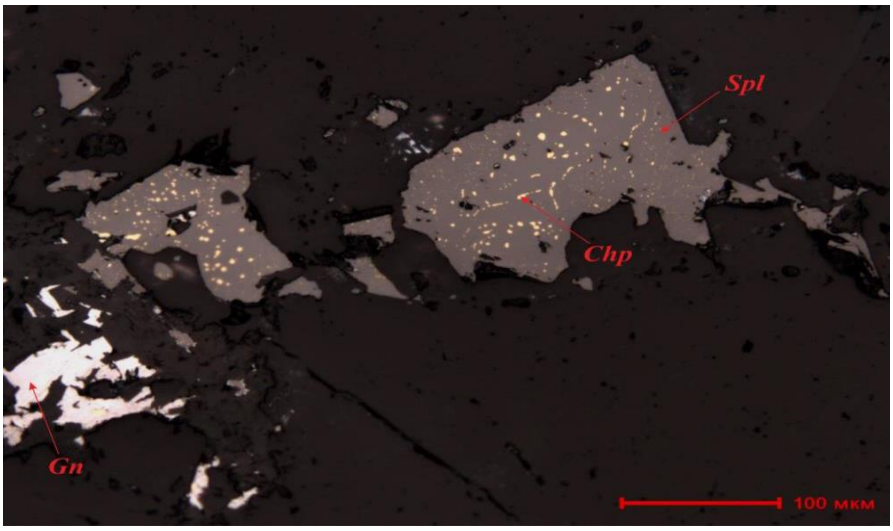


Figure 2. Polished section. Inclusions of chalcopyrite (Chp) in sphalerite (Spl) and galena (Cn) in quartz. Magnification x 200

Note – compiled by the authors

The mineralogical characteristics of the ore and its beneficiation products are further complicated by the presence of finely dispersed ore minerals, clayey material, and extensive replacement of sulfide minerals by their oxidized forms. As a result, particles of sulfide minerals potentially suitable for liberation and subsequent recovery are reduced to sizes of 5 μm or less.

The particle size distribution and the distribution of chemical elements across size fractions are presented in Table 2.

Table 2. Results of sieve analysis

Particle size class, mm	Yield, %	Grade, %, g/t*				Distribution, %			
		Zn	Pb	Fe	Ag*	Zn	Pb	Fe	Ag
-1+0.5 mm	26.46	0.37	0.30	5.83	4.26	17.88	13.80	24.70	20.7
-0.5+0.2 mm	19.90	0.51	0.48	6.21	4.79	18.62	16.62	19.77	17.5
-0.2+0.1 mm	32.89	0.54	0.55	5.92	5.17	32.51	31.90	31.14	31.21
-0.1+0.071 mm	9.49	0.73	0.91	6.90	9.27	12.55	15.13	10.48	16.14
-0.071+0.045 mm	3.31	0.83	1.06	7.43	7.00	5.00	6.16	3.93	4.25
-0.045+0 mm	7.96	0.93	1.17	7.84	6.99	13.43	16.39	9.98	10.2
Total:	100.0	0.56	0.58	6.31	5.50	100.0	100.0	100.0	100.0
Note – compiled by the authors									

The sieve analysis results show that the largest proportion of material is concentrated in the −0.2 +0.1 mm size fraction (32.89 %), while the fine fraction −0.045 +0 mm accounts for 7.96 %. Lead and zinc are predominantly concentrated in the finer size fractions, particularly in the −0.045 +0 mm fraction, where their contents reach 1.17 wt.% Pb and 0.93 wt.% Zn, respectively. The main distribution of lead and silver is observed in the −0.2 +0.1 mm size fraction.

Flotation experiments were carried out using a laboratory pneumatic–mechanical flotation machine VEKTIS with a cell volume of 3 L. Prior to flotation, a 1 kg ore sample with a particle size of −2+0 mm was ground in a laboratory ball mill MShL-7 under wet conditions. Grinding was performed at a pulp solids content of 60 % for 10 min, which provided a grinding fineness of 68.54 % of the −0.071 mm size fraction.

After grinding, the pulp was transferred to the flotation cell and diluted to a solids content of 30 % by mass. The flotation tests were conducted at a pulp temperature of 26 °C. Pulp mixing was carried out using an impeller speed of 1700 rpm. Air was supplied to the flotation zone at a pressure of not less than 5 bar, with an air flow rate of up to 10 NL/min. The froth product was removed using a froth scraper at a rotation speed of 3 rps.

The reagents were added sequentially with preliminary pulp conditioning. At the first stage, calcium polysulfide was introduced into the pulp as a sulfidizing agent to activate the surface of oxidized lead and zinc minerals. Then, sodium silicate (Na_2SiO_3) was added as a depressant of silicate and slime minerals. The dosage of Na_2SiO_3 was varied from 0 to 500 g/t to evaluate its effect on concentrate quality and flotation selectivity. Potassium butyl xanthate was used as the main collector at a dosage of 150 g/t, followed by the addition of T80 frother at 60 g/t and Aero 3418 collector at 60 g/t. The total conditioning time before flotation was 3 min. Main flotation was carried out for 10 min, followed by cleaner flotation for 6 min. The obtained products were filtered, dried, weighed, and analyzed for Pb, Zn, Ag, and Fe contents. Based on the analytical results, the yield, grade, and recovery of valuable components were calculated.

Additional confirmatory flotation tests were carried out for the optimal reagent regime in order to verify the reproducibility of the obtained results. Since the variation between repeated tests was small, Table 3 presents the results of the main laboratory tests for each sodium silicate dosage, while the repeated tests were used only to confirm the stability of the optimal Na_2SiO_3 dosage.

RESULTS

Based on the chemical, phase, mineralogical, and granulometric analyses, the oxidized lead–zinc ore from the Koshkuduk deposit can be classified as a refractory raw material. This is associated with the high oxidation degree of lead and zinc minerals, fine dissemination of ore minerals, and the significant presence of silicate and slime components. These features require the use of a combined reagent regime involving preliminary sulfidization of oxidized mineral surfaces followed by the depression of gangue minerals.

The scheme of the laboratory flotation tests is shown in Figure 3. It includes rougher flotation and a cleaner stage, which makes it possible to evaluate the effect of sodium silicate (Na_2SiO_3) not only on the recovery of valuable components but also on the quality of the resulting lead–zinc concentrate.

The results of the experiments evaluating the effect of sodium silicate dosage on flotation performance are presented in Table 3.

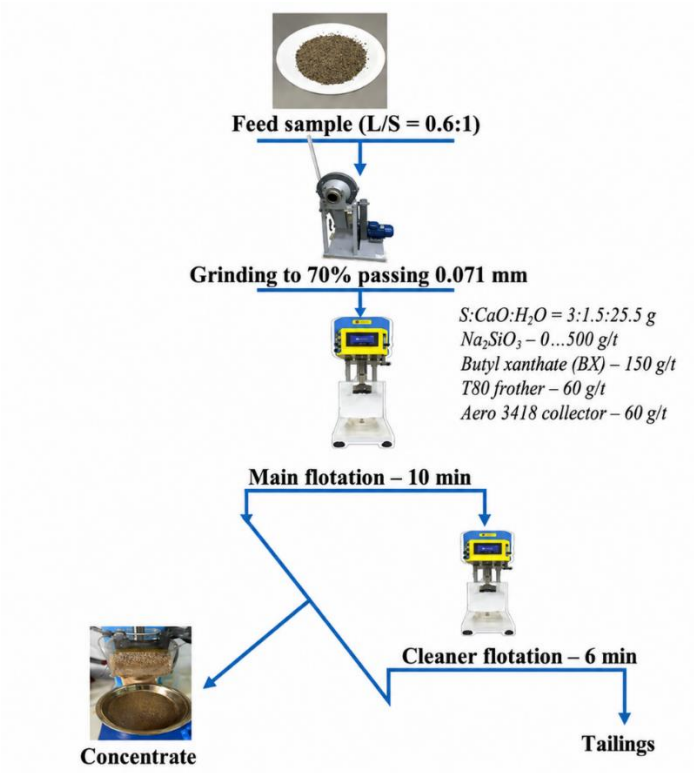


Figure 3. Technological scheme of laboratory flotation tests

Table 3. Effect of sodium silicate (Na_2SiO_3) dosage on the flotation performance of oxidized lead–zinc ore

Reagent dosage, g/t	Product	Yield, %	Grade, %, g/t*				Recovery, %			
			Pb	Zn	Ag*	Fe	Pb	Zn	Ag	Fe
Base test										
Na_2SiO_3 - 0	Concentrate	7.2	3.78	3.15	50.03	2.72	46.9	40.54	65.5	3.1
	Tailings	92.8	0.33	0.36	2.04	6.59	53.1	59.46	34.5	96.9
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0
Test 1										
Na_2SiO_3 - 30	Concentrate	4.34	6.04	5.33	78.74	5.68	45.2	41.3	62.1	3.9
	Tailings	95.66	0.33	0.34	2.18	6.34	54.8	58.7	37.9	96.1
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0
Test 2										
Na_2SiO_3 - 50	Concentrate	4.21	6.02	5.49	77.24	6.74	43.7	41.24	59.12	4.5
	Tailings	95.79	0.34	0.34	2.35	6.29	56.3	58.76	40.88	95.5
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0
Test 3										
Na_2SiO_3 - 70	Concentrate	4.19	6.19	5.47	80.24	6.66	44.7	41.0	61.1	4.4
	Tailings	95.81	0.33	0.35	2.23	6.29	55.3	59.1	38.9	95.6
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0
Test 4										
Na_2SiO_3 - 100	Concentrate	4.07	6.43	5.43	83.12	6.00	45.1	39.5	61.5	3.87
	Tailings	95.93	0.33	0.35	2.21	6.32	54.9	60.6	38.5	96.1

Reagent dosage, g/t	Product	Yield, %	Grade, %, g/t*				Recovery, %			
			Pb	Zn	Ag*	Fe	Pb	Zn	Ag	Fe
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0
Test 5										
Na_2SiO_3 - 120	Concentrate	3.91	7.20	5.88	87.63	6.62	48.51	41.04	62.3	4.1
	Tailings	96.09	0.31	0.34	2.16	6.30	51.49	58.96	37.7	95.9
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0
Test 6										
Na_2SiO_3 - 150	Concentrate	3.84	7.57	6.02	91.81	7.13	50.1	41.3	64.1	4.3
	Tailings	96.16	0.30	0.34	2.05	6.28	49.9	58.7	35.9	95.7
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0
Test 7										
Na_2SiO_3 - 170	Concentrate	3.62	8.58	6.44	100.46	6.97	53.56	41.64	66.12	4.0
	Tailings	96.38	0.28	0.34	1.93	6.29	46.44	58.36	33.88	96.0
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0
Test 8										
Na_2SiO_3 - 200	Concentrate	3.95	7.71	5.68	89.50	6.58	52.5	40.0	64.3	4.1
	Tailings	96.05	0.29	0.35	2.05	6.30	47.5	60.0	35.7	95.9
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0
Test 9										
Na_2SiO_3 - 500	Concentrate	4.01	7.41	5.62	88.05	5.66	51.23	40.24	64.2	3.6
	Tailings	95.99	0.29	0.35	2.05	6.34	48.77	59.76	35.8	96.4
	Feed (initial sample)	100.0	0.58	0.56	5.50	6.31	100.0	100.0	100.0	100.0

Note – compiled by the authors

To provide a clearer visualization of the effect of sodium silicate dosage on the recovery of valuable components, the recovery values of Pb, Zn, and Ag were plotted as a function of Na_2SiO_3 dosage. The obtained dependence is presented in Figure 4.

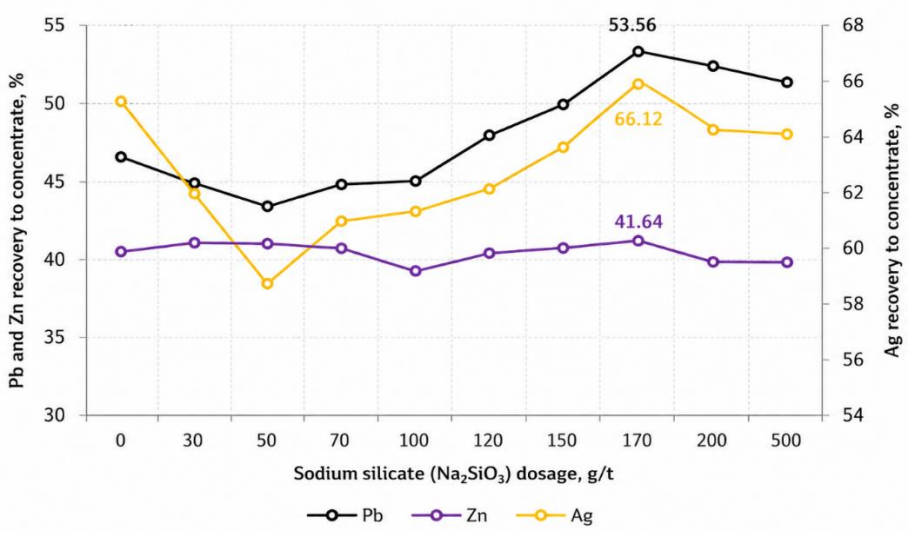


Figure 4. Effect of sodium silicate (Na_2SiO_3) dosage on Pb, Zn and Ag recovery into the concentrate

Note – compiled by the authors

As shown in Figure 4, the addition of sodium silicate (Na_2SiO_3) had different effects on the recovery of lead, zinc, and silver. In the base test without Na_2SiO_3, the recoveries of Pb, Zn, and Ag were 46.90 %, 40.54 %, and 65.50 %, respectively. When the Na_2SiO_3 dosage was increased to 170 g/t, Pb recovery increased to 53.56 %, Zn recovery to 41.64 %, and Ag recovery to 66.12 %. Thus, the highest recovery values for the valuable components were obtained at a sodium silicate dosage of 170 g/t.

A further increase in Na_2SiO_3 dosage to 200–500 g/t did not improve recovery. At 500 g/t, Pb recovery decreased to 51.23 %, Zn recovery to 40.24 %, and Ag recovery to 64.20 %. This indicates that an excess of sodium silicate may have a depressant effect not only on gangue minerals but also partially on sulfidized lead and zinc minerals. Therefore, a Na_2SiO_3 dosage of 170 g/t can be considered rational in terms of combining relatively high Pb, Zn, and Ag recoveries with improved concentrate quality.

In addition to conventional flotation indicators, such as product yield, grade, and recovery of valuable components, the Hancock–Luyken criterion was used to provide a more comprehensive assessment of separation selectivity. This criterion takes into account not only the recovery of the valuable component into the concentrate but also its losses in the tailings and the concentrate yield.

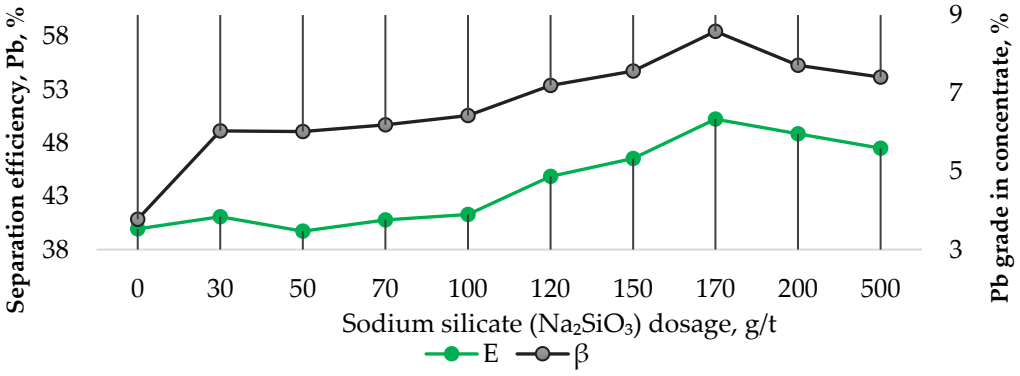
The Hancock-Luyken criterion was calculated using the following equation:

$$E = \frac{\varepsilon - \gamma}{1 - \alpha}$$

where ε is the recovery of the valuable component in the concentrate;

γ is the recovery of the valuable component in the tailings;

α is the concentrate yield.



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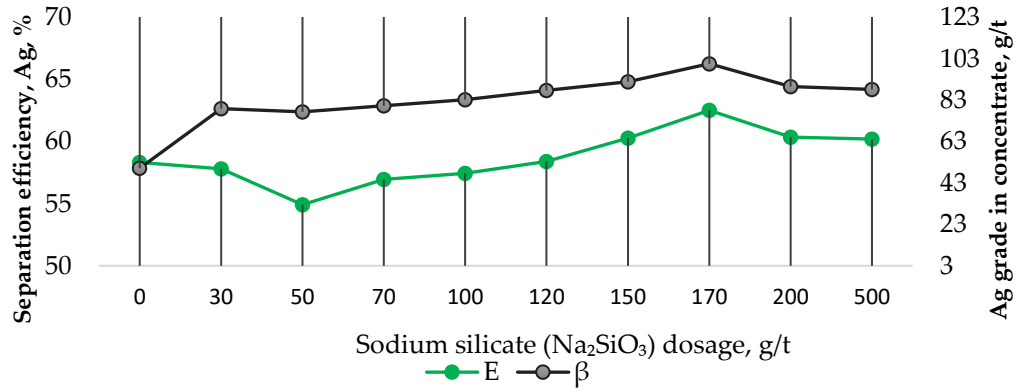

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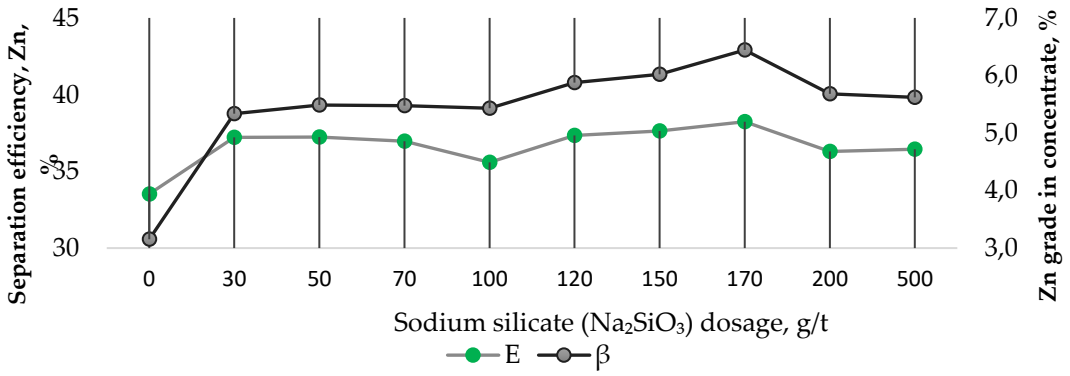

C

Figure 5. Effect of sodium silicate (Na_2SiO_3) dosage on separation efficiency and metal grade in concentrate: (A) Pb, (B) Ag, and (C) Zn.

Note – compiled by the authors

The maximum separation efficiency and metal grade were obtained at a sodium silicate dosage of 170 g/t. This dosage can therefore be considered optimal for improving the quality of the Pb–Zn concentrate under the studied flotation conditions.

The calculation results presented in Figure 5 show that the variation in separation efficiency generally correlates with changes in the metal grade and recovery of the target components. This confirms that the applied efficiency criterion adequately reflects the overall flotation performance at different sodium silicate dosages.

Various intermediate cases may also occur during flotation separation. With an increase in the partial yield of the concentrate in an ideal separation operation, while maintaining the same overall separation efficiency, the contribution of concentrate contamination increases. In this regard, the operational concentrate yield in the ideal separation operation can be used as an additional criterion complementing the Hancock–Luyken criterion.

The additional criterion is expressed by the following equation:

$$D = \frac{\varepsilon - E}{1 - E}$$

where ε is the recovery of the valuable component in the concentrate;

E is the separation efficiency determined by the Hancock-Luyken criterion.

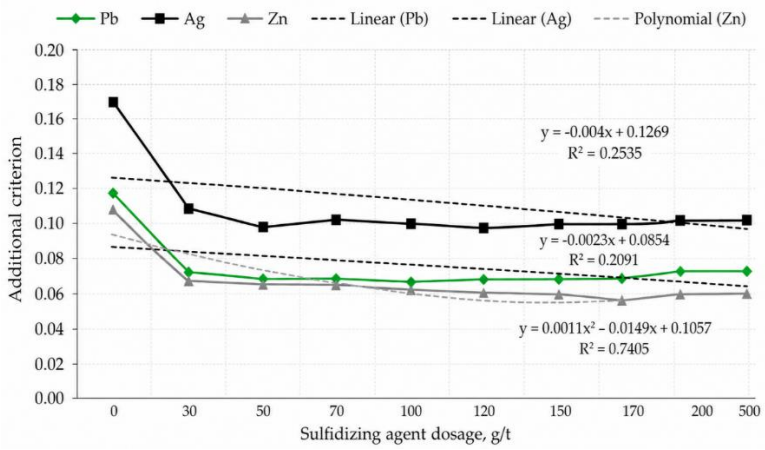

Figure 6. Additional separation efficiency criterion for Pb, Ag and Zn as a function of sodium silicate (Na_2SiO_3) dosage

Note – compiled by the authors

The analysis of the additional separation efficiency criterion shows that sodium silicate dosage has a significant effect on concentrate contamination and flotation selectivity. At low Na_2SiO_3 dosages, the additional criterion values are relatively high, especially for Ag, which indicates a greater contribution of gangue and slime entrainment into the concentrate.

With increasing Na_2SiO_3 dosage, the additional criterion for Pb, Ag and Zn generally decreases, indicating improved depression of silicate and slime minerals and reduced contamination of the froth product. The most favorable values are observed in the dosage range of 150–170 g/t, where high metal grades and recoveries are achieved together with improved separation selectivity.

Further increasing the Na_2SiO_3 dosage to 200–500 g/t does not lead to a significant improvement in the additional criterion. This suggests that excessive sodium silicate addition does not further enhance separation efficiency and may partially depress valuable mineral particles. Therefore, a sodium silicate dosage of 170 g/t can be considered rational for improving the quality of the Pb–Zn concentrate while maintaining acceptable recovery of Pb, Zn and Ag.

DISCUSSION

The obtained results can be explained by the mechanism of sodium silicate (Na_2SiO_3) action as a depressant and dispersant for silicate and slime minerals. According to the literature, sodium silicate undergoes hydrolysis in aqueous pulp with the formation of monomeric and polymeric silicate species, including H_4SiO_4 and $SiO(OH)_3^-$. These species can adsorb on the surface of gangue minerals, increasing their hydrophilicity and reducing the probability of collector attachment (Wang et al., 2024; Ni et al., 2025; Silva et al., 2012).

At low Na_2SiO_3 dosages, the depressant effect of the reagent is insufficient; therefore, part of the silicate, carbonate, and slime particles is mechanically entrained into the froth product. This results in an increased concentrate yield and lower Pb, Zn, and Ag grades. In the base test without sodium silicate, the concentrate yield was 7.20 %, whereas the grades of Pb, Zn, and Ag were relatively low. This indicates significant contamination of the concentrate by gangue minerals.

Increasing the Na_2SiO_3 dosage to 170 g/t promoted more effective depression of gangue and slime minerals. As a result, the mechanical entrainment of non-valuable particles into the froth product decreased, while the grades of valuable components in the concentrate increased. At this dosage, the maximum grades of Pb, Zn, and Ag were obtained: 8.58 %, 6.44 %, and 100.46 g/t, respectively. At the same time, relatively high recoveries of Pb, Zn, and Ag were maintained, confirming the improvement in flotation selectivity.

The depressant mechanism of Na_2SiO_3 is associated with the formation of hydrophilic silicate coatings on the surface of gangue minerals. It has been shown that sodium silicate is an effective depressant for calcium-bearing minerals and can reduce the floatability of calcite, fluorite, dolomite, and other minerals due to the adsorption of silicate species on their surfaces (Wang et al., 2024). In addition, Na_2SiO_3 is widely considered a flotation regulator capable of depressing quartz and silicate minerals; at elevated dosages, it may also exert a depressant effect on sulfide minerals (Ni et al., 2025).

A further increase in Na_2SiO_3 dosage to 200–500 g/t did not improve concentrate quality. On the contrary, a decrease in Pb, Zn, and Ag grades was observed. This may be related to excessive adsorption of silicate species in the pulp and partial depression of sulfidized surfaces of valuable minerals. Such behavior is consistent with literature data indicating that excessive sodium silicate addition can reduce the flotation activity not only of gangue minerals but also of valuable minerals due to the formation of hydrophilic surface layers (Ni et al., 2025; Silva et al., 2012).

Thus, the effect of Na_2SiO_3 is clearly dosage-dependent. At insufficient reagent dosage, gangue depression remains incomplete, whereas excessive sodium silicate addition may

deteriorate the flotation of valuable minerals. A dosage of 170 g/t can be considered rational, since it provides an optimal balance between the depression of silicate–slime components and the preservation of the floatability of sulfidized Pb-, Zn-, and Ag-bearing minerals.

CONCLUSIONS

This study investigated the effect of sodium silicate (Na_2SiO_3) dosage on the flotation performance of oxidized lead–zinc ore from the Koshkuduk deposit after preliminary sulfidization with calcium polysulfide.

It was established that the introduction of Na_2SiO_3 into the reagent regime has a pronounced effect on flotation selectivity and the quality of the obtained concentrate. In the base test without sodium silicate, the concentrate yield was 7.20 %, with Pb, Zn, and Ag grades of 3.78 %, 3.15 %, and 50.03 g/t, respectively. The addition of Na_2SiO_3 reduced the concentrate yield and increased the grades of valuable components, which is associated with the depression of silicate and slime minerals.

The best technological results were obtained at a Na_2SiO_3 dosage of 170 g/t. Under this condition, the concentrate yield was 3.62 %, while the Pb grade reached 8.58 %, the Zn grade reached 6.44 %, and the Ag grade reached 100.46 g/t. The recoveries of Pb, Zn, and Ag were 53.56 %, 41.64 %, and 66.12 %, respectively. Compared with the base test, the Pb grade in the concentrate increased by 2.27 times, the Zn grade by 2.04 times, and the Ag grade by 2.01 times.

Calculations using the Hancock–Luyken criterion and the additional separation efficiency criterion confirmed that a Na_2SiO_3 dosage of 170 g/t provides the most favorable combination of valuable component grades, recoveries, and separation selectivity. A further increase in sodium silicate dosage to 200–500 g/t did not improve flotation performance, which can be explained by the excessive depressant effect of the reagent and the possible reduction in the floatability of sulfidized valuable minerals.

Thus, a sodium silicate dosage of 170 g/t can be considered rational for improving the quality of Pb–Zn concentrate from oxidized ore of the Koshkuduk deposit. The obtained results confirm the feasibility of using Na_2SiO_3 as a depressant for silicate and slime components during the flotation of refractory oxidized lead–zinc ores.

CONFLICT OF INTEREST: The authors declare no conflict of interest..

FUNDING: This research was funded by the Committee of Science of the Ministry of Education and Science of the Republic of Kazakhstan, grant project No. AR23489765.

INSTITUTIONAL REVIEW BOARD STATEMENT: Not applicable. This study was purely technical and did not involve human participants or animals.

INFORMED CONSENT STATEMENT: Not applicable. This study did not involve human participants.

DATA AVAILABILITY STATEMENT: The data supporting the findings of this study are available from the corresponding author upon reasonable request.

ACKNOWLEDGEMENTS: The authors express their gratitude to the laboratory staff for technical support during sample preparation, flotation experiments, and analytical measurements.

STATEMENT ON THE USE OF ARTIFICIAL INTELLIGENCE TECHNOLOGIES: Artificial intelligence tools were used for language editing, text structuring, and improvement of academic style during manuscript preparation. The authors reviewed and approved the final version of the manuscript and take full responsibility for its scientific content, data interpretation, and conclusions.

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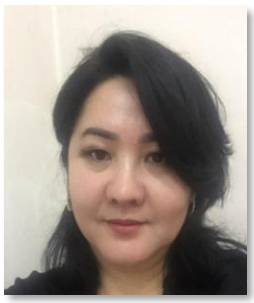

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