MINERAL PROCESSING OF NON-FERROUS METALS / ОБОГАЩЕНИЕ РУД ЦВЕТНЫХ МЕТАЛЛОВ

UDC 622.765
https://doi.org/10.17073/0021-3438-2025-4-5-17

Research article
Научная статья

Effect of metal-containing modifier compositions with sodium sulfide on the selective flotation of copper and zinc sulfides

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Abstract: The most efficient selective reagent modes for the flotation of a copper–zinc pyrite ore from one of the Ural deposits have been developed, based on the use of compositions of metal-containing reagent modifiers in combination with sodium sulfide. The study analyzed the most effective conditions for separating copper and zinc minerals from pyrite during the bulk flotation of copper–zinc ore, as well as the conditions for improving the selective separation of the bulk copper–zinc concentrate. The influence of reagent–modifier compositions introduced into the bulk flotation cycle on the process parameters of selective flotation of the bulk concentrate was evaluated. The results of fractional analysis of the floatability of copper, zinc, and iron minerals were presented, taking into account the flotation kinetics and the distribution of these minerals in the floated concentrate by fractions: poorly floatable, moderately floatable, and easily floatable. The reagent–modifier compositions used not only depressed pyrite flotation but also ensured efficient separation of copper and zinc minerals into individual concentrates. It was found that the most effective selectivity in flotation separation of copper and zinc minerals was achieved by introducing a composition of ferrous sulfate and sodium sulfide into the bulk copper–zinc flotation circuit in equal proportions (50 and 50 g/t). As a result, a copper–pyrite concentrate containing 12 wt. % Cu with a copper recovery of 74.45 % and a zinc concentrate containing 5 wt. % Zn with a zinc recovery of 73.68 % from the ore were obtained. Analysis of flotation kinetics showed that the introduction of this reagent mixture contributed to the highest flotation rate of copper, ensuring a maximum copper recovery to the froth (copper–pyrite) product of 86.74 %.

Key words: copper-zinc ores, recovery, flotation, sodium sulfide, flotation kinetics, separation, selectivity, modifiers, sulfides.

For citation: Htet Zaw Oo, Kyaw Zay Ya, Goryachev B.E. Effect of metal-containing modifier compositions with sodium sulfide on the selective flotation of copper and zinc sulfides. *Izvestiya. Non-Ferrous Metallurgy*. 2025;31(4):5–17.
https://doi.org/10.17073/0021-3438-2025-4-5-17

Оценка влияния композиции металлосодержащих модификаторов с сернистым натрием на селективность флотационного разделения сульфидов меди и цинка

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Аннотация: Разработаны наилучшие селективные реагентные режимы для флотации медно-цинковой колчеданной руды одного из месторождений Урала, основанные на применении композиций металлосодержащих реагентов-модификаторов в сочетании с сернистым натрием. Проанализированы наиболее эффективные условия флотации минералов меди и цинка от пирита в коллективном цикле флотации медно-цинковой руды, а также условия для повышения селективного разделения коллективного медно-цинкового концентрата. Оценено влияние композиций реагентов-модификаторов, вводимых в коллективный цикл флотации, на технологические показатели селективной флотации коллективного концентрата. Приведены результаты фракционного анализа флотируемости минералов меди, цинка и железа с учетом кинетики флотации и распределения этих минералов во флотируемый концентрат по фракциям: труднофлотируемой, среднефлотируемой и легкофлотируемой. Используемые композиции реагентов-модификаторов не только подавляли флотацию пирита, но и обеспечивали эффективное разделение минералов меди и цинка в отдельные концентраты. Установлено, что наиболее эффективное влияние на селективность флотационного разделения минералов меди и цинка оказывает дозирование композиции железного купороса и сульфида натрия в коллективную медно-цинковую флотацию в равных долях (50 и 50 г/т). В результате применения данной композиции реагентов получены медно-пиритный концентрат с содержанием меди 12 % при извлечении меди 74,45 % и цинковый концентрат с содержанием цинка 5 % при извлечении 73,68 % от руды. Анализ кинетики флотации показал, что введение указанной смеси реагентов способствует наилучшей скорости флотации меди, обеспечивая максимальное извлечение меди в пенный (медно-пиритный) продукт на уровне 86,74 %.

Ключевые слова: медно-цинковые руды, извлечение, флотация, сернистый натрий, кинетика флотации, разделение, селективность, модификаторы, сульфиды.

Для цитирования: Хтет Зо У, Чжо Зай Яа, Горячев Б.Е. Оценка влияния композиции металлосодержащих модификаторов с сернистым натрием на селективность флотационного разделения сульфидов меди и цинка. *Известия вузов. Цветная металлургия.* 2025;31(4):5–17. https://doi.org/10.17073/0021-3438-2025-4-5-17

Introduction

Copper—zinc pyrite ores from the Ural deposits are predominantly pyrite-type and rank among the most challenging to process [1]. This is due to the high pyrite content (85—90 wt. %) and fine, irregular intergrowth of sulfide minerals with each other and with gangue minerals [2; 3].

Flotation is the most efficient process for separating minerals from finely disseminated non-ferrous metal ores such as copper—zinc pyrite ores [4—7]. The processing of large volumes of refractory raw materials requires the development and implementation of new methods and technologies [8]. The use of selective reagent modes and optimization of pulp conditioning can increase the contrast of surface properties of minerals with similar technological characteristics [9].

At present, the most common scheme at concentrator plants is the bulk—selective flotation circuit, in which copper and zinc minerals are recovered from the bulk copper—zinc concentrate while pyrite flotation is suppressed [10; 11]. This technology enables high-quality recovery of metals from finely disseminated and refractory ores, followed by the production of concentrates for various applications, and significantly enhances the selectivity of mineral separation [12; 13].

Despite significant advances in the mineral processing industry, the selective flotation of non-ferrous metal sulfide ores — particularly copper—zinc pyrite ores — remains an urgent challenge [10]. It can be addressed by improving existing technologies through modifications to process flowcharts, optimizing reagent application modes and broadening the range of reagents used in the flotation circuit [10]. One of the key approaches to improving the efficiency of processing copper—zinc ores is the development of new reagent systems, including compositions of metal-containing reagent modifiers [14].

Previous studies [15—17] have demonstrated the positive effect of metal-containing reagent—modifier compositions on the results of bulk flotation of copper—zinc ore. However, their role in the selective flotation of bulk copper—zinc concentrate has not been investigated. The aim of this study is to examine the effect of metal-containing modifier compositions combined with sodium sulfide, introduced into the bulk flotation circuit, on the efficiency of selective flotation of this concentrate.

Materials and methods

Research objects

Samples of copper—zinc pyrite ore from a Ural deposit were used as the test material. The average contents of copper, zinc, and iron in the run-of-mine ore were 0.65 ± 0.02, 1.30 ± 0.08 и 38.48 ± 1.52 wt. %, respectively [15]. The following reagents were used in the experiments: potassium butyl xanthate (PBX) as

Table 1. **Quantitative *X*-ray diffraction (XRD) phase analysis of the raw ore (powder)**

Таблица 1. Рентгенографический фазовый анализ исходной руды (порошка)

Mineral	Ideal formula	Content, wt. %
Pyrite	FeS_2	61
Chalcopyrite	$CuFeS_2$	3
Sphalerite	ZnS	5
Tetrahedrite	$(Cu,Fe,Zn,Ag)_{24}(Sb,As)_8S_{26}$	0.5
Quartz	SiO_2	21
Calcite	$Ca(CO_3)$	3
Barite	$BaSO_4$	2
Chlorite	$(Mg,Fe)_{4.75}Al_{1.25}(Si_{2.75}Al_{+.25}O_{10})(OH)_8$	2
Illite	$KAl_2(AlSi_3O_{10})\cdot(OH)_2$	1
	Σ	**98.5**

Table 2. **Mineralogical analysis of the raw ore (polished section)**

Таблица 2. Минералогический анализ исходной руды (полированный шлиф)

Mineral	Mineral content in analyzed samples, wt. %										Mean	Confidence interval
	1	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*		
Pyrite	84	75	91	77	92	75	74	73	81	96	81.80	±5.27
Quartz	15	20	6	16	5	22	20	23	14.5	2	14.35	±4.67
Sphalerite	–	–	–	3	–	–	–	2.5	–	–	2.75	±0.72
Chalcopyrite	–	–	1.5	0.5	1.5	1.5	3	1	2	1	1.50	±0.57
Fahlore	–	–	–	–	1	–	1.5	–	1.5	0.5	1.13	±0.40
Carbonate	–	3	1	2	–	–	1.5	–	1	0.5	1.50	±0.63
Others	1	2	0.5	1.5	0.5	1.5	–	0.5	–	–	1.07	±0.44

the collector, pine oil as the frother, lime (CaO) as the pH regulator, and the modifier reagents ferrous sulfate ($FeSO_4{\cdot}7H_2O$), copper sulfate ($CuSO_4{\cdot}5H_2O$), zinc sulfate ($ZnSO_4{\cdot}7H_2O$), and sodium sulfide ($Na_2S{\cdot}9H_2O$).

Quantitative *X*-ray diffraction (XRD) and reflected-light ore microscopy of the raw ore sample were performed using a Tongda TD-3700 *X*-ray diffractometer and an Olympus BX51 optical microscope. The results are summarized in Tables 1 and 2.

Procedure for bulk flotation of copper—zinc ore

Ore preparation was carried out as follows: the initial ore was crushed in a jaw crusher (DSh 80×150) to a particle size of 3 mm. After averaging and quartering, 1 kg ore samples were taken from the crushed material. These samples were ground in a laboratory ball mill (MShL-7) to a fineness of 80 % passing –0.074 mm, after which the ground material was subjected to bulk flotation.

The flotation tests were conducted in mechanical flotation machines (Mekhanobr and FL-137) with cell volumes of 0.5 L and 3.0 L. Under laboratory conditions, flotation experiments were carried out according to a process flowsheet that included Flotation *I* (copper head flotation) and Flotation *II* (copper—zinc flotation) [15].

The flowsheet of the bulk flotation of copper—zinc ore and the production of the bulk copper—zinc concentrate is shown in Fig. 1.

In Flotation *I*, only the collector and frother were used, and the flotation process was carried out for 2 min. The tailings from Flotation I served as the feed for Flotation *II*, in which lime, modifier reagents, potassium butyl xanthate, and pine oil were added [15]. The total consumption of modifier reagents in all tests was 100 g/t. Flotation *II* was then performed for 8 min. The concentrates obtained from Flotation *I* and Flotation *II* were combined to produce the bulk copper—zinc concentrate.

Procedure for selective flotation of the bulk concentrate

The process of selective flotation of the bulk copper—zinc concentrate included the following stages: desorption of the collector from the surface of sulfide minerals in a sodium sulfide medium to achieve strong depression of sphalerite; thickening and washing of the concentrate to remove excess alkalinity; regrinding of the coarse bulk copper—zinc concentrate to 92—95 % passing –0.044 mm; and primary copper—pyrite flotation [18]. The flowsheet of the selective flotation process for the bulk copper—zinc concentrate is shown in Fig. 2.

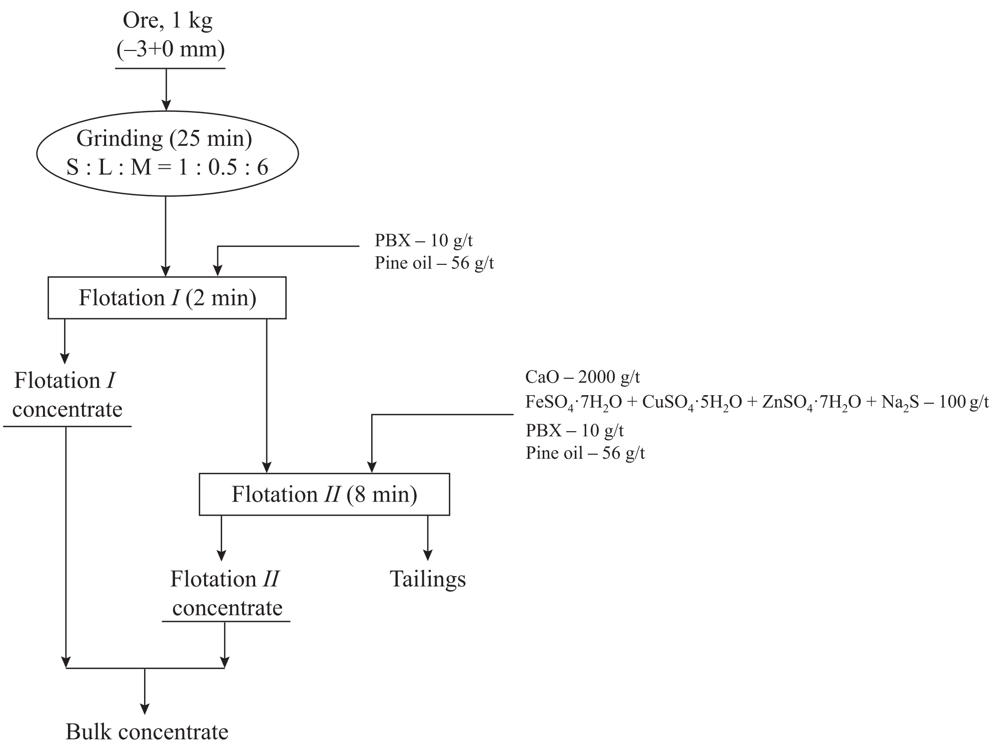

Fig. 1. Flowsheet for producing the bulk copper–zinc concentrate

Рис. 1. Схема получения коллективного медно-цинкового концентрата

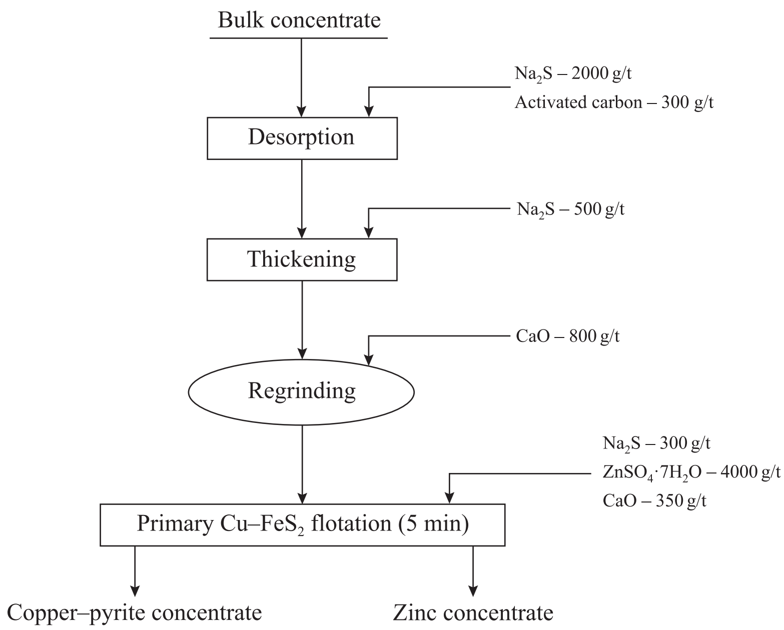

Fig. 2. Flowsheet of the selective flotation of the bulk copper–zinc concentrate

Рис. 2. Схема селекции коллективного медно-цинкового концентрата

According to the selective flotation flowsheet (Fig. 2), the bulk concentrate was first subjected to desorption using sodium sulfide (2000 g/t) and activated carbon (300 g/t). After desorption, the pulp was thickened to a solids content of 60 % with the addition of sodium sulfide (500 g/t) and reground to 95 % passing –0.044 mm with lime addition (800 g/t) [19]. The reground product was then subjected to primary copper—pyrite flotation.

In practice, separation of copper and zinc minerals is most often carried out using a scheme based on sphalerite depression [20—22]. In the primary copper—pyrite flotation process, sphalerite depression was achieved through the use of sodium sulfide (300 g/t) and zinc sulfate (4000 g/t). This operation was conducted under alkaline conditions maintained by the addition of lime (350 g/t), ensuring a pH of 8.5—9.0. Upon completion of the flotation tests, the products were dried, and the copper, zinc, and iron contents were analyzed using an ElvaX *X*-ray fluorescence spectrometer.

Results and discussion

Effect of reagent–modifier compositions on the efficiency of selective flotation of the bulk concentrate

The results of selective flotation of the bulk copper—zinc concentrate with the addition of reagent—modifier compositions in Flotation *II* are presented in Figs. 3 and 4.

According to the data in Fig. 3, the use of the studied reagent—modifier compositions in Flotation *II* has a significant effect on the process performance of the selective flotation of the bulk concentrate. The compositions of ferrous and copper sulfates ($0.5FeSO_4$ + + $0.5CuSO_4$) and ferrous sulfate with sodium sulfide ($0.5FeSO_4 + 0.5Na_2S$) provided the best results for copper recovery into the concentrate (Fig. 3, *a*). In the first case, the copper—pyrite concentrate contained 13 % Cu with a copper recovery of 69.87 %, and in the second — 12 % Cu with a recovery of 75.45 %. Compared with the baseline test without modifiers (in which the copper content was 11.32 % and the recovery was 55.72 %), the use of these reagent—modifier mixtures increased copper recovery by 14.15 % and 19.73 %, respectively.

Notably, when using the $0.5FeSO_4 + 0.5Na_2S$ mixture, copper loss with the scavenger product was the lowest, amounting to 11.54 %. Nevertheless, the highest copper content in the copper—pyrite concentrate (13.32 %) was achieved when introducing the $0.25CuSO_4 + 0.75Na_2S$ mixture into Flotation *II*, while the zinc content in this concentrate remained at 2.29 % (Fig. 3, *b*).

As shown in Fig. 4, a similar trend was observed in the zinc (scavenger) concentrate, where the selectivity of copper and zinc separation increased. It was established that introducing into Flotation *II* the mixtures of ferrous and zinc sulfates ($0.5FeSO_4 + 0.5ZnSO_4$)

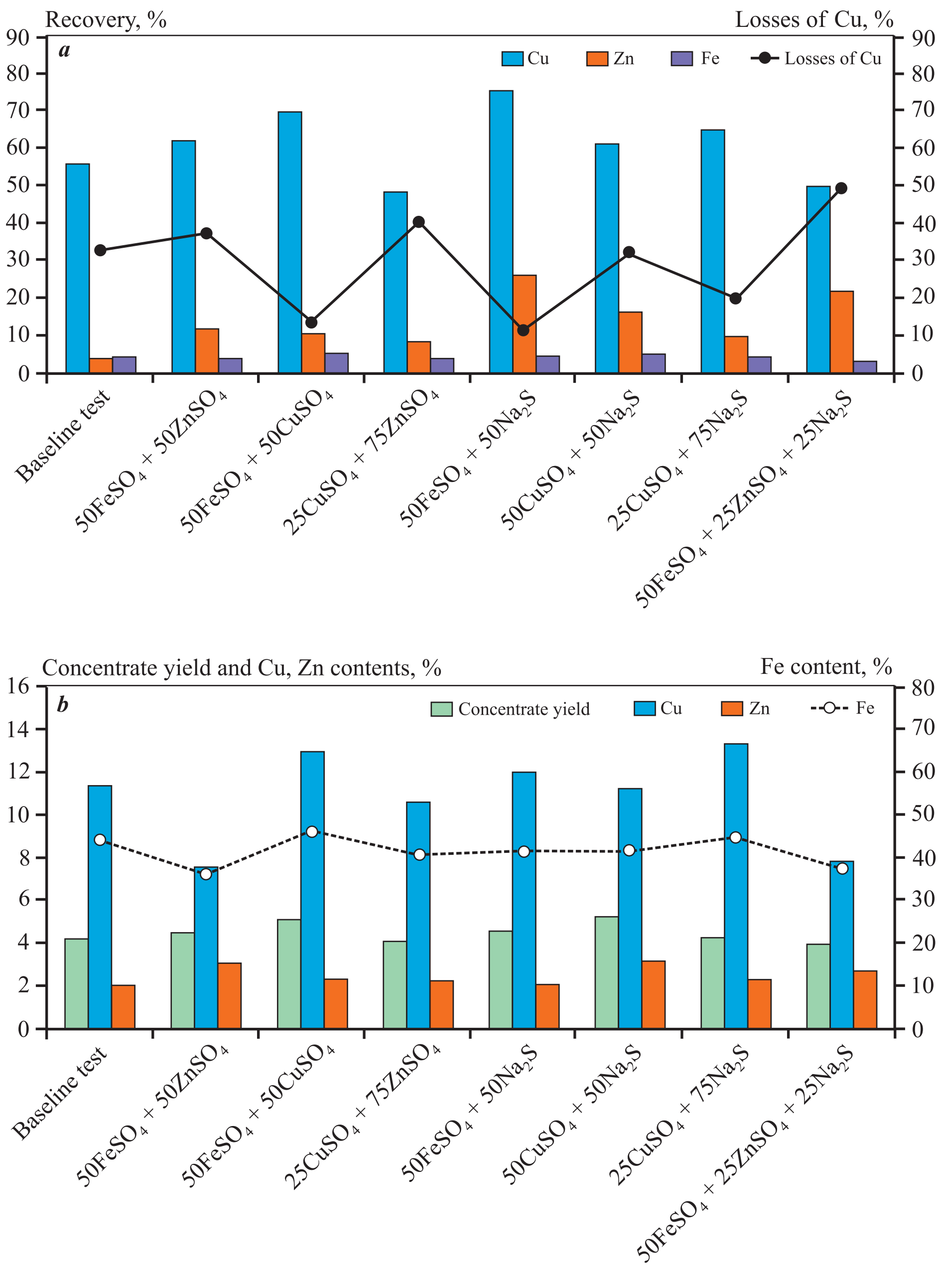

Fig. 3. Recovery of copper, zinc, and iron into the copper–pyrite concentrate and copper losses with the scavenger product (***a***), as well as the concentrate yield and copper, zinc, and iron contents (***b***) depending on the reagent–modifier compositions introduced into Flotation *II*

Рис. 3. Извлечение меди, цинка и железа в медно-пиритный концентрат и потери меди с камерным продуктом (***a***), а также выход концентрата и содержание меди, цинка и железа в нем (***b***) при расходе композиций модификаторов во флотацию *II*

and ferrous sulfate, zinc sulfate, and sodium sulfide ($0.5FeSO_4$ + $0.25ZnSO_4$ + $0.25Na_2S$) ensured the highest zinc recovery into the concentrate (Fig. 4, *a*): 78.86 % and 77.93 %, respectively. This corresponds to an increase in zinc recovery by 49.67 % and 48.71 % compared with the baseline mode without modifiers (29.19 %).

It is noteworthy that when using the mixture of ferrous and copper sulfates ($0.5FeSO_4$ + $0.5CuSO_4$), the obtained concentrate demonstrated the highest zinc content (Fig. 4, *b*): 8.3 % Zn, while the copper content remained at 1.72 %. This increased the zinc grade of the obtained zinc concentrate by 4.83 % compared with the baseline test without modifiers (3.47 %).

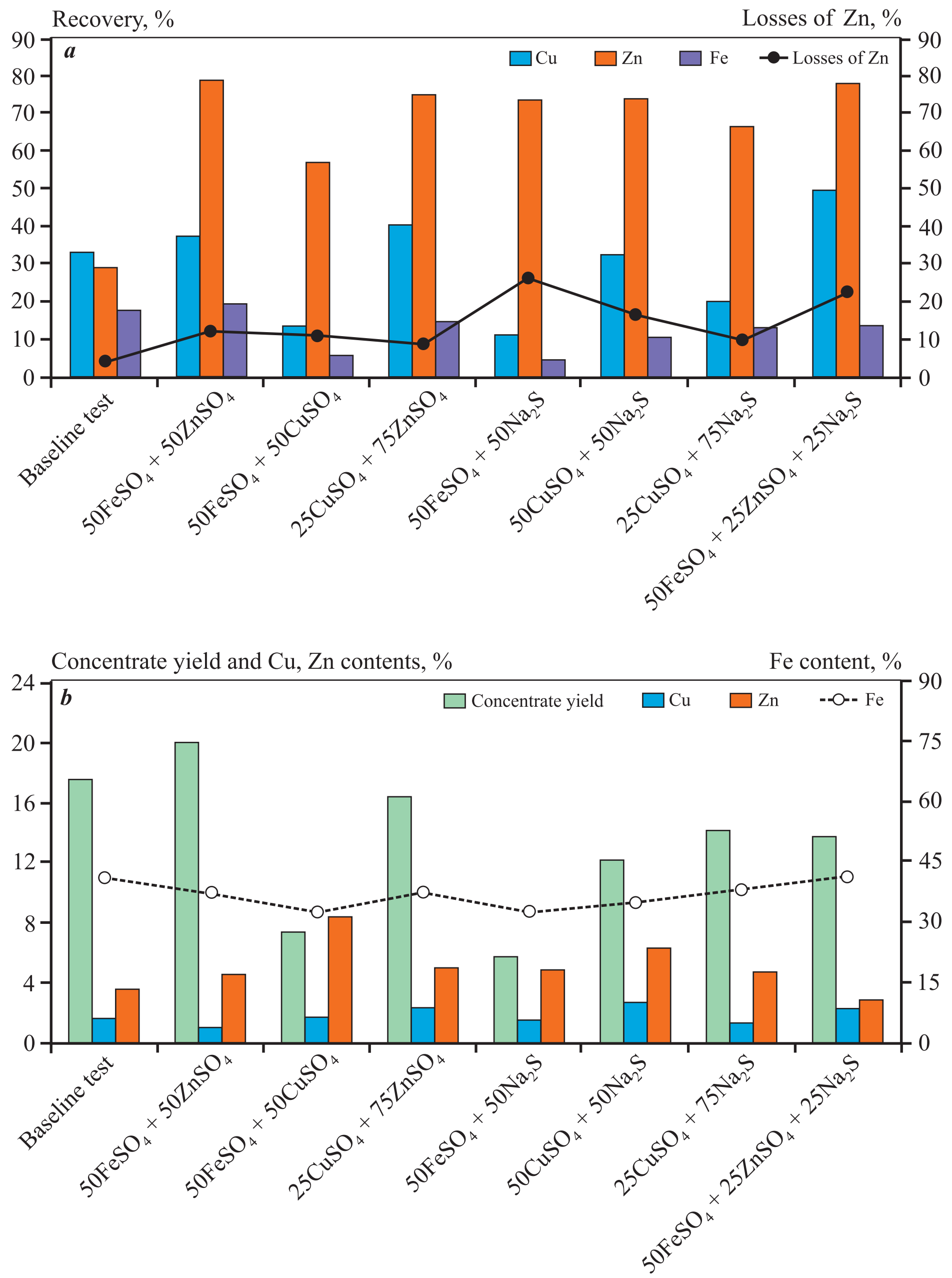

Fig. 4. Recovery of copper, zinc, and iron into the zinc concentrate and zinc losses with the froth product (***a***), as well as the concentrate yield and copper, zinc, and iron contents (***b***) depending on the reagent–modifier compositions introduced into Flotation *II*

Рис. 4. Извлечение меди, цинка и железа в цинковый концентрат и потери цинка с пенным продуктом (***a***), а также выход концентрата и содержание меди, цинка и железа в нем (***b***) при расходе композиций модификаторов во флотацию *II*

Kinetics of selective flotation of the bulk concentrate and fractional analysis of the floatability of copper, zinc, and iron minerals

The study of the kinetics of selective flotation of the bulk concentrate involved the fractional collection of the froth product at flotation time intervals of 0.21 min (corresponding to three scrapes of the froth layer), 0.43 min (six scrapes), 0.64 min (nine scrapes), 1, 2, and 5 min. During the investigation of the flotation kinetics of the bulk concentrate, the distribution of materials was analyzed according to fractions differing in the floatability of the respective minerals in the froth product of the main copper–pyrite flotation. Using the Spectr software developed by Prof. D.V. Shekhirev, the floatability spectra of copper and zinc minerals in the concentrate were evaluated [15; 23; 24].

Fractional analysis of selective flotation involved partitioning the floated material into six fractions according to floatability, each defined by a specific range of flotation rate constants (K, min^{-1}) and described by the equation proposed by K.F. Beloglazov: 1st fraction, $0 < K < 0.0001$; 2nd, $0.0001 < K < 0.001$; 3rd, $0.001 < K < 0.01$; 4th, $0.1 < K < 1$; 5th, $1 < K < 10$; and 6th, $10 < K < 100$ [15; 25—28].

Fig. 5 presents the results of copper flotation kinetics into the copper—pyrite product when reagent—modifier compositions were introduced into Flotation *II*.

As seen in Fig. 5, *a*, the introduction of reagent—modifier compositions into Flotation *II* resulted in a substantial increase in the copper flotation rate compared with the baseline test without their use. The highest copper flotation rates were obtained with the reagent mixtures containing combinations of ferrous and copper sulfates ($0.5FeSO_4 + 0.5CuSO_4$) and ferrous sulfate with sodium sulfide ($0.5FeSO_4 + 0.5Na_2S$). At the early stages of flotation (0.21 min), copper recovery in these variants reached 32.39—32.98 %, which is more than seven times higher than that of the control test (4.3 %). After 2 min of flotation, copper recovery increased to 80.46 % in the presence of these reagents, while in the baseline test it was only 22.98 %. The maximum copper recovery into the froth product (86.74 %) was achieved using the $0.5FeSO_4 + 0.5Na_2S$ mixture at $\tau = 5$ min.

The results of fractional analysis (Fig. 5, *b*) showed that copper minerals were distributed among the 1st (poorly floatable), 3rd and 4th (moderately floatable), and 5th (easily floatable) fractions. In the control test, the proportions of poorly, moderately, and easily floatable fractions were 0.61, 0.35, and 0.04 rel. units, respectively. The most pronounced effect on the floatability of copper minerals was observed with the compositions $0.5FeSO_4 + 0.5CuSO_4$ and $0.5FeSO_4 + 0.5Na_2S$. In the first case, the proportion of poorly floatable fractions decreased to 0.098, moderately floatable to 0.301, while the easily floatable fraction increased to 0.601 rel. units. In the second case, the poorly floatable fraction decreased further to 0.009, the moderately floatable fraction increased to 0.481, and the easily floatable fraction reached 0.51 rel. units. Under these conditions, copper recovery into the copper—pyrite concentrate reached its maximum.

Fig. 6 shows the results of zinc flotation kinetics into the copper—pyrite product when reagent—modifier compositions were introduced into Flotation *II*.

The analysis of the data in Fig. 6, *a* indicates that the addition of reagent—modifier compositions to Flotation *II* caused a slight increase in the zinc flotation rate compared with the baseline test. The mixtures $0.5FeSO_4 + 0.5CuSO_4$ and $0.5FeSO_4 + 0.5Na_2S$ provided the highest zinc flotation rates, with zinc recoveries of 29.98 % and 26.32 % into the froth product after 5 min, exceeding the control value (12.59 %). The least influence on the zinc flotation rate was observed with the $0.5FeSO_4 + 0.5ZnSO_4$ mixture, giving only 15.46 % zinc recovery. Thus, despite the suppression of sphalerite

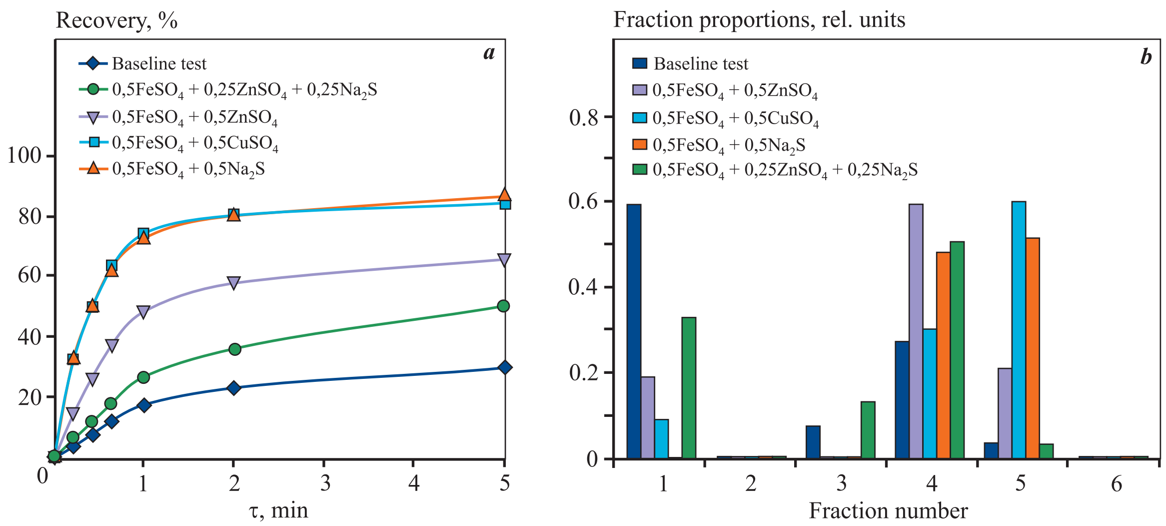

Fig. 5. Kinetics of copper flotation into the copper-pyrite product (***a***) and its distribution by fractions (***b***) upon addition of reagent—modifier compositions into Flotation *II*

Рис. 5. Кинетика флотации меди в медно-пиритный продукт (***a***) и ее распределение по фракциям (***b***) при дозировании композиций реагентов-модификаторов во флотацию *II* flotation in the lime medium when sodium sulfide and zinc sulfate were used, these reagent—modifier mixtures slightly accelerated zinc flotation.

According to Fig. 6, *b*, zinc minerals were distributed among the 1st and 2nd (poorly floatable), 3rd and 4th (moderately floatable), and 5th (easily floatable) fractions. In the baseline test without modifiers, the proportions of poorly, moderately, and easily floatable fractions were 0.742, 0.257, and 0.001 rel. units, respectively. When the $0.5FeSO_4$ + $0.5CuSO_4$ mixture was used, the easily floatable fraction amounted to only 0.012 rel. units. The introduction of $0.5FeSO_4$ + $0.5Na_2S$ and $0.5FeSO_4$ + $0.25ZnSO_4$ + $0.25Na_2S$ mixtures into Flotation *II* considerably reduced the proportion of poorly floatable fractions to 0.42 and 0.38 rel. units, respectively. The proportion of moderately floatable fractions increased to 0.58 and 0.62 rel. units, while easily floatable fractions were entirely absent. This explains the influence of these reagent—modifier mixtures on the poorly floatable fractions of zinc minerals into the copper—pyrite concentrate.

Fig. 7 presents the results of iron flotation kinetics into the copper—pyrite product when reagent—modifier compositions were introduced into Flotation *II*.

The kinetic curves of iron flotation (Fig. 7, *a*) demonstrate a notable acceleration of the process upon addition of reagent—modifier compositions to Flotation *II* compared with the baseline test without them. The highest iron flotation rate was obtained with the $0.5FeSO_4$ + + $0.5Na_2S$ composition, yielding 50.88 % recovery after 5 min of flotation. A similar trend was observed with the $0.5FeSO_4$ + $0.5CuSO_4$ mixture, where iron reco-

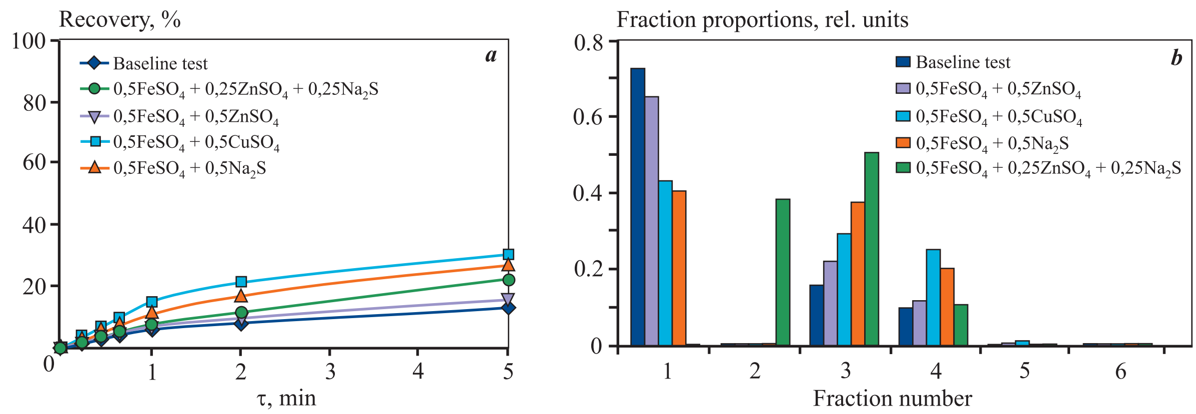

Fig. 6. Kinetics of zinc flotation into the copper-pyrite product (***a***) and its distribution by fractions (***b***) upon addition of reagent–modifier compositions into Flotation *II*

Рис. 6. Кинетика флотации цинка в медно-пиритный продукт (***a***) и его распределение по фракциям (***b***) при дозировании композиций реагентов-модификаторов во флотацию *II*

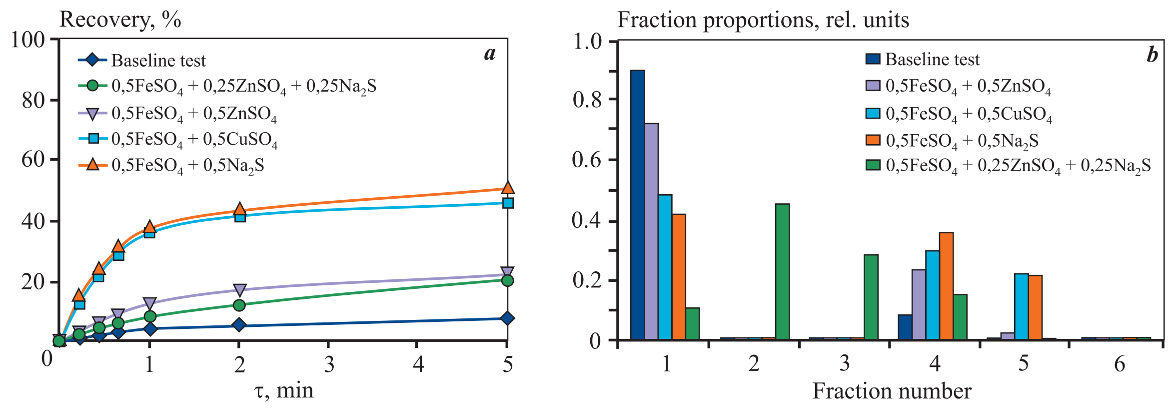

Fig. 7. Kinetics of iron flotation into the copper-pyrite product (***a***) and its distribution by fractions (***b***) when upon addition of reagent–modifier compositions into Flotation *II*

Рис. 7. Кинетика флотации железа в медно-пиритный продукт (***a***) и его распределение по фракциям (***b***) при добавлении композиций реагентов-модификаторов во флотацию *II* very reached 46.06 % over the same period. The smallest increase in iron flotation rate was observed when using $0.5FeSO_4$ + $0.5ZnSO_4$ and $0.5FeSO_4$ + $0.25ZnSO_4$ + + $0.25Na_2S$.

From the data in Fig. 7, *b*, it can be seen that in all experimental variants the proportion of poorly floatable fractions was high, especially in the baseline test (0.912 rel. units). However, upon addition of $0.5FeSO_4$ + + $0.5Na_2S$ and $0.5FeSO_4$ + $0.5CuSO_4$, this proportion decreased significantly to 0.428 and 0.48 rel. units, respectively. At the same time, the proportion of easily floatable fractions was the highest, reaching 0.221 and 0.211 rel. units. The most pronounced transfer of iron minerals into the moderately floatable fractions (3 and 4) was observed when the $0.5FeSO_4$ + $0.5Na_2S$ composition was used, whereas with $0.5FeSO_4$ + $0.5ZnSO_4$, the share of these fractions increased, but a high proportion of poorly floatable ones remained.

Thus, the analysis of selective flotation kinetics demonstrated that the introduction of reagent—modifier compositions into Flotation *II* allows for evaluating the floatability of copper, zinc, and iron minerals into the froth (copper—pyrite) product, which is critical for improving process selectivity. In this case, copper minerals are mainly represented by moderately and easily floatable fractions, sphalerite by poorly and moderately floatable fractions, and pyrite by poorly, moderately, and partially easily floatable fractions.

Conclusions

1. A selective reagent mode was developed for the bulk—selective flotation of a copper—zinc pyrite ore from a Ural deposit, based on applying compositions of metal-containing reagent modifiers with sodium sulfide in the bulk flotation circuit.

2. Analysis of the flotation results for the copper—zinc pyrite ore identified an optimal combination of modifier reagents — ferrous sulfate, copper sulfate, and zinc sulfate with sodium sulfide — which not only reduced pyrite recovery to the bulk concentrate but also increased the selectivity of separating copper and zinc minerals into individual concentrates.

3. Based on the flotation tests, among the reagent—modifier compositions studied, the combination of ferrous sulfate and sodium sulfide in an equal ratio ($0.5FeSO_4$ + $0.5Na_2S$) markedly enhanced the selectivity of separating copper and zinc minerals, increasing copper recovery to the froth (copper—pyrite) product while minimizing losses to the scavenger (zinc) product.

4. Using the proposed reagent composition yielded a copper—pyrite concentrate assaying 12 % Cu at a copper recovery of 75.45 %, as well as a zinc concentrate assaying 4.76 % Zn at a zinc recovery of 73.68 %. The mixture $0.5FeSO_4$ + $0.5CuSO_4$ also produced positive selective-flotation results; however, copper recovery in this case was lower than with the former reagent mixture.

5. Kinetic analysis showed that introducing the $0.5FeSO_4$ + $0.5Na_2S$ mixture in Flotation *II* had a pronounced effect on the copper flotation rate, ensuring a maximum copper recovery of 86.74 % to the froth product. Under these conditions, the share of poorly floatable fractions decreased to 0.009 rel. units, the moderately floatable fractions increased to 0.481, and the easily floatable fractions rose to 0.51. Sphalerite was virtually absent from the easily floatable fractions when this modifier composition was used.

6. The studies on the effect of sodium-sulfide-containing reagent—modifier compositions applied in the bulk flotation circuit of copper—zinc ores on the selectivity of copper and zinc sulfide separation support the feasibility of using an equal-dose mixture of ferrous sulfate and sodium sulfide (50 g/t + 50 g/t) in analogous flotation processes at concentrators treating copper—zinc pyrite ores from the same deposit.

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The article was submitted 02.09.2024, revised 10.03.2025, accepted for publication 12.03.2025

Статья поступила в редакцию 02.09.2024, доработана 10.03.2025, подписана в печать 12.03.2025