

Article

Assessment of Silicon and Rhenium Recovery Efficiency from Copper-Containing Tailings of Processing Plants

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Abstract: In the face of the global depletion of natural resources and increasing demand for sustainable development, processing industrial waste, such as tailings from processing plants, is becoming essential. This study focuses on combined processing technologies, including flotation concentration and concentrate processing, allowing the efficient recovery of valuable components. This study aims to investigate the possibility of thermochemical enrichment and the opening of low-grade copper tailings of processing plants with the transfer of silicon and rhenium in the form of silicate-ions and perrhenate-ions into a solution with the output of a multifactor multiplicative model and obtaining tabular nomograms. Multifactor experiments on the thermochemical enrichment of rough copper concentrates made it possible to construct partial dependences of silicon and rhenium extraction into a solution and to obtain multiplicative Protodyakonov–Malyshev models of these processes and multifactor nomograms over a wide range of temperatures, durations, and alkali-to-concentrate ratios to determine the maximum recovery rates. The developed multifactor models made it possible to establish the optimal intervals of changes in the concentrate sintering parameters, providing high recovery rates (over 85% of silicon and 98% of rhenium) during subsequent water leaching. Optimal sintering conditions (temperature of 350 °C, the duration of 90 min, and the ratio of NaOH to concentrate = 1:2) ensured a recovery of up to 85% of silicon and 98% of rhenium from the concentrate into the solution. This recovery rate reduces the need for primary raw materials and positively affects the production’s environmental performance because it minimizes the amount of industrial waste disposal.

Keywords: final tailings; concentrate; sintering; recovery; mathematical models; silicon; rhenium; multiple correlation analysis



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1. Introduction

Environmental pollution caused by metallurgical industry waste is a major modern problem. Tailings are one of the large-scale and underutilized types of mining waste [1,2]. Tailings occupy large territories, fine waste particles fall into water bodies, and wind blows them off to considerable distances.

However, the tailings generated from ore processing may have promising applications, including the production of permeable bricks and their use as industrial additives. Therefore, future studies should focus on developing technologies to extract valuable components, such as silicon and nonferrous metals from tailings, thereby increasing their economic attractiveness [3,4].

2. Literature Review

Modern studies have shown that SiO_2 recovery from secondary sources expands the raw material base and has a significant potential for industrial applications. As noted in [5], processing industrial waste for silica recovery helps solve environmental problems and opens up opportunities for developing a green economy. Environmental sustainability is a key factor that has stimulated research in this area. A previous study [6] provided evidence of significant reduction in waste disposal by recycling waste and biomass to produce SiO_2 . These processes can be integrated into the existing production cycles to reduce harmful environmental impacts. Thus, the tailings of processing plants are technogenic deposits with significant potential for processing and extracting beneficial components that could significantly replenish the national mineral resource base.

Thermal treatment is one of the technologies used for processing silicon-containing waste, which makes it possible to obtain pure silicon dioxide with minimal carbon dioxide emissions. Thus, in studies [7–12], silicon dioxide was precipitated after processing mineral high-silicon ore and phosphorus production slags [9]. The main disadvantages of these processes are the low recovery of silicon into a solution, the lack of information on the recovery of rhenium, which is often an associated metal, and the complexity of processing owing to the need for an additional leaching operation.

Hydrometallurgical processing methods are also under development to extract silicon from the ash and waste generated during silicon production [13,14]. To decompose high-silica bauxite, a sequential combined hydrochemical alkaline method for alumina production, the Bayer process, has been developed [15,16].

Papers [17,18] have described a thermochemical enrichment method for high-silica alumina-containing raw materials. The preliminary chemical enrichment technique proposed in [19] is of particular interest for hydroalkali treatments. Chemical enrichment consists of autoclave treatment with a caustic sodium solution of sulfide copper ore of the Konyrat deposit (Balkhash, Karaganda region, Kazakhstan) with a composition of Cu—0.32 wt. %, SiO_2 —68.35 wt. %, Al_2O_3 —10.88 wt. %, Fe—3.1 wt. %, S—2.35 wt. %, and Re—1.0 g/t. To reduce the concentration of SiO_2 , the concentrate underwent autoclave desiliconization at an alkali concentration of 160 g/L, id, a solid ratio of 5:1, and a temperature of 230 °C. The disadvantages of this method are the low degree of desiliconization of concentrates (57.10% SiO_2) and the lack of complex recovery of other nonferrous metals, such as rhenium [19].

In addition, methods for processing copper concentrates, including sintering the concentrate with alkali and producing white soot, have been previously developed [20–22], but these methods did not include the extraction of the associated metals.

The analysis of the literature shows that given the assessment of the efficiency of various options for processing alumina-containing materials (sintering, acid treatment, and hydro-alkali treatment), preference should be given to the alkaline hydrochemical method of decomposition as it is more economical, excluding high-temperature sintering, and less energy-intensive. In contrast to similar studies, the processing of raw materials with the complex extraction of silicon and rhenium in a solution is proposed.

Thus, the development of an efficient technology for ore processing with the associated extraction of rhenium is urgently required. The basis for solving the urgent problem of creating a scientific basis for the complex processing technology of sub-standard concentrates is the process of thermochemical decomposition-sintering, which makes it possible to reveal and transfer valuable components (Si and Re) into water-soluble compounds in one process.

This study aims to investigate the possibility of thermochemical enrichment and the opening of low-grade copper tailings of processing plants with the transfer of silicon and

rhenium in the form of silicate- and perrhenate-ions into a solution with the output of a multifactor multiplicative model and tabular nomograms.

In contrast to similar methods, this study proposes the extraction of silicon and rhenium from a rough concentrate obtained after the flotation enrichment of waste copper-containing tailings using low-temperature sintering and sodium hydroxide as a reagent that forms a chemical compound with silicon dioxide with further water leaching. A feature of this direction is the possibility of complex processing owing to the optimal combination of flotation and chemical enrichment, which provides the maximum interdependent recovery of metals (Si and Re). The essential characteristic features are the sequence of technological operations, conditions of rough concentrate sintering, and silicon and rhenium recovery, with further enrichment of the copper cake (through the removal of waste rock) after water leaching. This study uses process operations with multifactor dependencies based on tabular nomograms.

3. Materials and Methods

Rough copper concentrate obtained by flotation from waste copper-containing tailings was used for the experiments. The enrichment of mature tailings via flotation has been investigated in previous studies [21–23]. Figure 1 shows the technological scheme and the conditions of the laboratory experiments. The primary components in the copper-containing tailings were copper—0.14 wt. %, rhenium—1.51 g/t, sulfur—0.13 wt. %, and total iron—2.10 wt. %. The rock components included silicon dioxide—59.12 wt. %, aluminum oxides—11.15 wt. %, calcium—4.84 wt., and magnesium—2.01 wt. %.

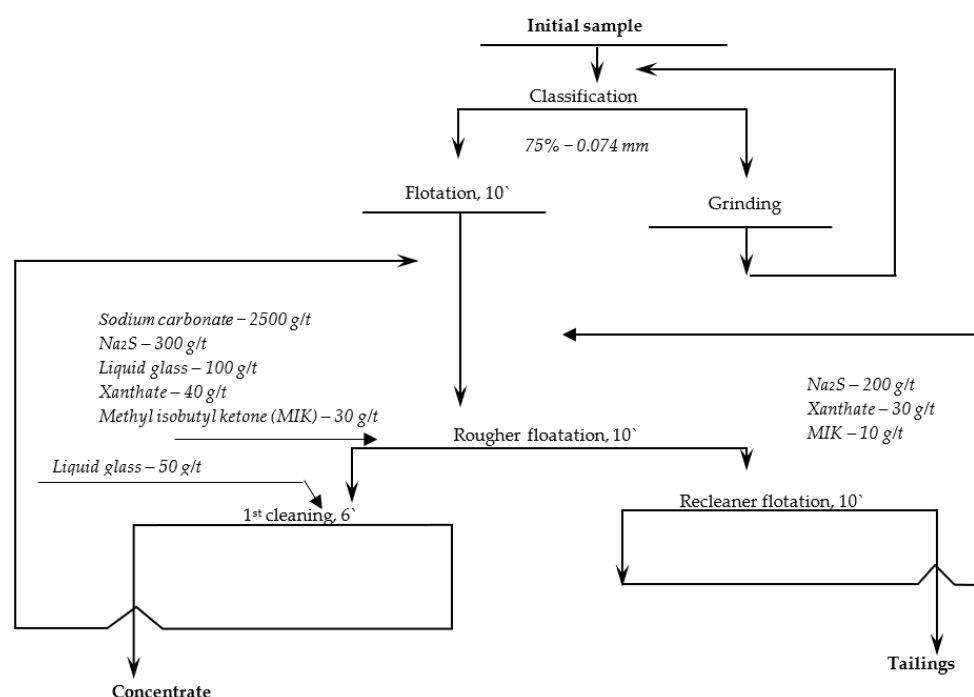


Figure 1. Design of the closed experiment for concentrate production (developed by the authors).

Ore mineralization, mainly represented by iron hydroxides and oxides, was insignificant. Much smaller amounts of copper oxide were formed in the malachite form; copper sulfides in the form of chalcopyrite, chalcosine + digenite + covellite, bornite, and pyrite were even rarer.

Ore mineralization was as follows.

- Major: iron hydroxides and oxides (goethite, hematite).

- Rarely occurring (in descending order): malachite, chalcopyrite, chalcosine + digenite, covellite, bornite, pyrite, magnetite, and covellite.
- All ore minerals were almost 100% represented in concretions containing rock minerals or as inclusions. Separate ore mineral grains are extremely rare.
- The size of copper minerals varied from 0.001 to 0.07 mm:
 - malachite—0.01 mm to 0.05 mm;
 - sulfide copper minerals—from 0.001 mm to 0.07 mm.

During the closed experiment, based on the developed technological scheme and optimized reagent mode, copper concentrate with a yield of 1.53% was obtained and used for further studies. Tables 1 and 2 list the chemical and phase compositions of the concentrate samples, respectively.

Table 1. Content of main components in rough copper concentrate.

Content of Components, %		Content of Components, %	
Cu	4.857	Al	5.399
Fe	5.388	Ag, g/t	123.7
Zn	0.293	Si	25.98
Pb	0.314	Re, g/t	4.776
S	3.484	-	-

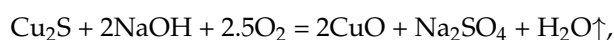
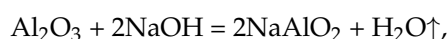
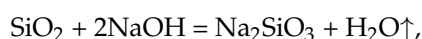
Table 2. Phase composition of copper in concentrate.

Occurrence Form	Content of Cu, % (abs.)	Distribution of Cu, % (rel.)
Cu of sulfide minerals	4.291	88.347
Cu of oxidized minerals	0.566	11.653
Total	4.857	100

The concentrate contained 88.347% Cu of sulfide minerals and 11.653% Cu of oxidized minerals.

Using a Bruker D2 Phaser diffractometer (step size of 1 s, scanning speed of 120° / min), the phase composition of the concentrate was determined: quartz (SiO₂)—39.4%; albite (K[AlSi₃O₈])—28.1%; chalcopyrite (CuFeS₂)—8.7%; anorthite (Ca[Al₂Si₂O₈])—20.0%; and calcite (CaCO₃)—3.7%.

Studies on concentrate sintering with alkali occurred in the temperature range t from 300 °C to 500 °C, and sintering duration τ was from 30 min to 120 min; the ratio of caustic alkali NaOH to concentrate was $\gamma_{\text{NaOH}} = 0.5\text{--}2$ units at a constant layer height in the crucible $h = 0.03$ m; NaNO₂ was added at 0.1% of the concentrate mass to improve oxidation processes in the intervals of the reaction of the interaction of caustic alkali with base minerals, as well as based on the stoichiometry of the reaction [24]. The following interactions express the main sintering reactions.



The main experiment followed these conditions: the sample mass was 100 g (with a class yield of $-0.071 + 0\text{--}80\%$), t was 350 °C, τ was 90 min, and γ_{NaOH} was 2:1.

Water leaching of the sinter occurred at a temperature of 60 °C, liquid–solid ratio of 3:1, and duration of 60 min. Under these conditions, copper is extracted (by 5%) into an aqueous solution. However, in our case, when the precipitant reagent was added, copper did not enter the solution. Further studies should include experiments in this regard. The obtained silicate solution was used to produce commercial products [22,24]. Technical reagents were used in all experiments.

Design of experiments for the sequential study of the acting factors involved using the method [25] and obtaining a multiplicative multifactor Protodyakonov–Malyshev model. The model generalized partial functions through their normalization by a general value and combined them into a multifactor dependence of the Protodyakonov equation type. For a more economical and uniform representation of the generalized function in the entire multifactor space within the actual limits of the change and influence of each factor, further adaptation of the method was expressed in such an organization of the experiment when the normalization of partial dependencies occurred according to the results of the main experiment $\bar{y}_{me}$, which entered each partial dependence. In this case, the total number of experiments is reduced by subtracting one factor. After obtaining this equation, it can be used for the construction of multifactor tabular nomograms with the selection of areas of acceptable and unacceptable combinations of factor levels, and thus, for process control [24].

$$y = \bar{y}_{me} \prod_{i=1}^{i=n} \frac{y_i}{\bar{y}_{i,ei}}$$

When deriving the equation, the nonlinear multiple correlation coefficient R and its significance t_R expressed by the known formulas [26] were used to test its adequacy:

$$R = \sqrt{1 - \frac{\sum_{i=1}^n (y_{e,i} - y_{c,i})^2}{\sum_{i=1}^n (y_{e,i} - \bar{y}_{e,av})^2}} \quad (1)$$

$$t_R = \frac{R\sqrt{n-k-1}}{1-R^2} > 2 \quad (2)$$

where $y_{e,i}$ —experimental value, $y_{c,i}$ —calculated value, $y_{e,av}$ —average experimental value, n —the number of independent (not repeated) experimental data, k —the number of acting factors, and $(n - k - 1)$ —the number of degrees of freedom of the adequacy variance.

Our research highlights the main problems of silicon and rhenium recovery from final copper-containing tailings. After selecting suitable research methods, we performed concentrate thermochemical enrichment and water leaching while obtaining multiplicative multifactor Protodyakonov–Malyshev models for Si and Re. As a result, we recovered silicon up to 85% and rhenium up to 98% into a solution and obtained multifactor tabular nomograms, which included the complete number of combinations of all factors and levels among themselves. The effectiveness of the selected methods of thermochemical treatment was confirmed, and the scientific novelty and practical significance of the study were established.

4. Results and Discussion

To recover the target components and obtain marketable products, tests on the thermochemical enrichment of rough concentrates with alkali, followed by water leaching, were conducted. The concentrate was mixed with sodium hydroxide at a given ratio (according to the reaction stoichiometry), after which the material under study was placed in a furnace and heated to a given temperature and duration. Figure 2 shows dot plots with the approximation of partial dependencies. Table 3 presents the experimental results

of the thermochemical enrichment of the concentrate with the alkali. According to the X-ray phase analysis performed using a D2 Phaser, the concentrate after thermochemical enrichment contained Na_4SiO_4 —sodium silicate and NaAlSiO_4 —sodium aluminosilicate.

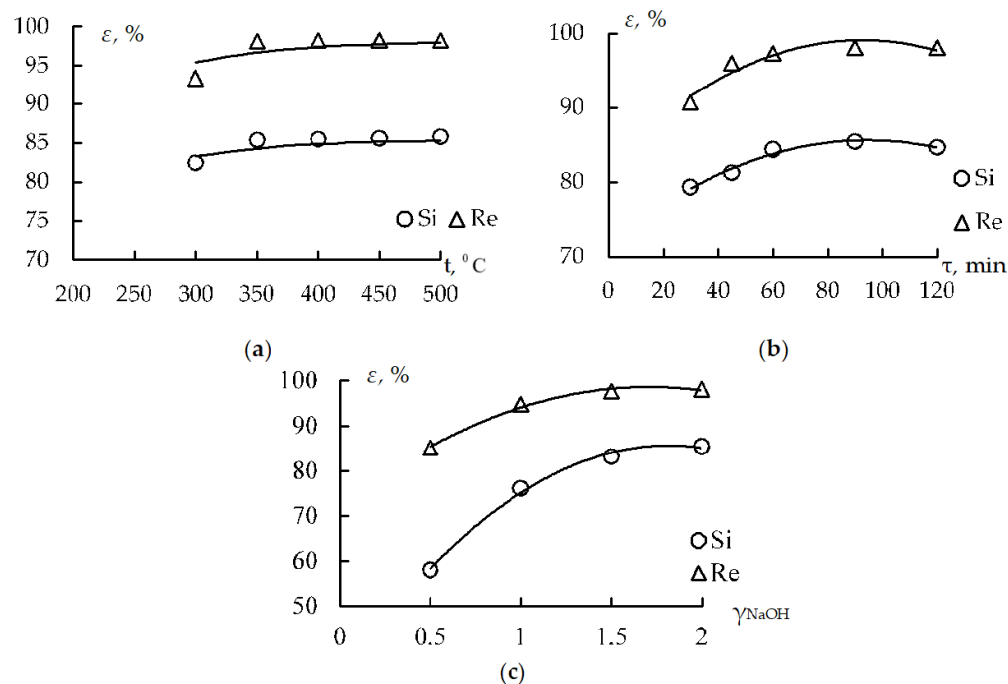


Figure 2. Effect of various factors on silicon and rhenium recovery into the solution: (a) temperature, °C; (b) duration, min; (c) alkali consumption. $\circ \Delta$ —experimental data for Si and Re; lines, according to the equations in Table 4.

Table 3. Conditions and results of the effect of different factors on silicon extraction into the solution.

Factor Under Study, Experimental Conditions		Cake Output, %	Content in Cake		Experimental Data on Extraction into the Solution, %, (ϵ)		Extraction According to Generalized Equations, %	
			Si, %	Re, g/t	Si	Re	Si by (3)	Re by (4)
t , °C ($\tau = 90$ min, $\gamma_{\text{NaOH}} = 2:1$)	300	21.18	21.58	1.74	82.41	93.30	82.26	96.48
	350	21.54	17.63	0.43	85.38	98.06	83.28	97.73
	400	20.73	18.25	0.42	85.44	98.18	83.84	98.42
	450	20.00	18.79	0.43	85.54	98.18	84.14	98.80
	500	19.73	18.71	0.43	85.79	98.23	84.31	99.01
τ , min ($\gamma_{\text{NaOH}} = 2:1$, $t = 350$ °C)	30	27.0	19.90	1.65	79.32	90.68	78.63	90.08
	45	24.54	19.85	0.78	81.25	95.99	80.32	93.25
	60	25.0	16.24	0.51	84.37	97.33	81.53	95.59
	90	21.54	17.63	0.43	85.38	98.06	83.28	97.73
	120	22.45	17.72	0.41	84.68	98.07	84.54	96.51
γ_{NaOH} ($\tau = 90$ min, $t = 350$ °C)	0.5	40.54	26.91	1.75	57.99	85.16	57.07	85.25
	1.0	27.91	22.32	0.91	76.02	94.69	74.83	93.96
	1.5	21.27	20.61	0.53	83.12	97.64	83.56	98.12
	2.0	21.54	17.63	0.43	85.38	98.06	84.58	97.73

The stage of the preliminary desiliconization of the rough concentrate yielded up to 85% of silica and 98% of rhenium in the solution. The silicate solution obtained under optimal conditions had the following composition: $\text{Na}_2\text{O} = 126.5$ g/L, $\text{SiO}_2 = 112.8$ g/L, and $\text{Al}_2\text{O}_3 = 4.0$ g/L.

Table 4. Partial functions of silicon and rhenium extraction into the solution with the determination of nonlinear multiple correlation coefficient R and its significance t_R according to (1) and (2).

Functions	R	$t_R > 2$
$\varepsilon_{Si} = 85.792(1 - \exp(-0.0121 \cdot t))$	0.8189	4.3059
$\varepsilon_{Si} = (66.905 \cdot \tau^{0.0523})$	0.9053	8.6904
$\varepsilon_{Si} = (-15.774 \cdot \gamma_{NaOH}^2 + 57.285 \cdot \gamma_{NaOH} + 33.601)$	0.9980	353.20
$\varepsilon_{Re} = 98.18 \cdot (1 - \exp(-0.0119 \cdot t))$	0.7783	3.4193
$\varepsilon_{Re} = 0.358 \cdot \tau - 0.0019 \cdot \tau^2 + 82.556$	0.9478	16.146
$\varepsilon_{Re} = 31.105 \cdot \gamma_{NaOH} - 9.11 \cdot \gamma_{NaOH}^2 + 72.087$	0.9962	185.73

After water leaching, the solution was used to obtain marketable products such as white soot and ammonium perrhenate. By increasing the leaching time from 30 min to 120 min, the extraction of silicon and rhenium into the solution increased from 79.32% to 85.37%, and 90.68% to 98.06%, respectively. According to the results of a series of experiments, the optimum sintering time was 90 min, because a further increase in time did not significantly affect the process.

Table 4 shows the partial equations for silicon and rhenium recovery into the solution, whose adequacy was determined using Equations (1) and (2).

The multifactor equation for silicon and rhenium extraction into the solution includes significant partial functions for temperature, duration, and alkali-to-concentrate ratio normalized to the central calculated value. As the partial functions (Table 4) in the multifactor equation are combined as a product, the overall mean value should be calculated as the geometric mean. The generalized equations for silicon and rhenium extraction into solution are as follows:

$$\varepsilon_{Si} = 1.37 \cdot 10^{-4} (85.792(1 - \exp(-0.0121 \cdot t)) \cdot (66.905 \cdot \tau^{0.0523})) \cdot (57.285 \cdot \gamma_{NaOH} - 15.774 \cdot \gamma_{NaOH}^2 + 33.601). \quad (3)$$

$$R = 0.9732. \quad t_R = 63.888 > 2$$

$$\varepsilon_{Re} = 1.04 \cdot 10^{-4} (98.18 \cdot (1 - \exp(-0.0119 \cdot t)) \cdot (0.358 \cdot \tau - 0.0019 \cdot \tau^2 + 82.556)) \cdot (31.105 \cdot \gamma_{NaOH} - 9.11 \cdot \gamma_{NaOH}^2 + 72.087). \quad (4)$$

$$R = 0.9304. \quad t_R = 23.985 > 2$$

The obtained values of the nonlinear multiple correlation coefficients and their significance, according to Equations (1) and (2), indicate the high adequacy of the multifactor equation. This makes it possible to use these equations to calculate the multifactor space of indicators of silicon and rhenium extraction into the solution with the variation in technologically possible limits of change of each significant factor and by selecting, in these space areas, acceptable and unacceptable values of indicators; in this case, 70–80% (for acceptable values) and below 70% and above 80% (for unacceptable values). Moreover, such a space can be represented on the plane in tabular numerical form for any number of factors. Table 5 presents a three-factor numerical space for Si and Re recovery rates. Combinations of factor levels that provide an acceptable recovery of 80% (green) are highlighted (based on the data obtained in [20]). This table can be used as a flowchart for the recovery of the joint silicon and rhenium.

Table 5 shows that the thermochemical enrichment is inexpedient at a sintering duration of 30 min, a temperature of 300 °C, and a ratio of alkali to the concentrate in the range from 0.5 to 2. Simultaneously, the high values are in the area of increased sintering temperatures and extended duration, which may not be economically efficient because of excessive heating costs. Therefore, such a flow chart helps find an acceptable control solution in the sufficiently extensive zones of permissible and optimum indicators.

Table 5. Dependence of silicon and rhenium extraction into the solution (%) on various factors. (Combinations with extraction rates above 80% are highlighted in green).

$t, ^\circ\text{C}$	τ, min	30		45		60		90		120	
	γ_{NaOH}	Si	Re	Si	Re	Si	Re	Si	Re	Si	Re
300	0.5	53.15	77.57	54.29	80.3	55.11	82.31	56.29	84.16	57.15	83.11
	1.0	69.66	85.49	71.16	88.51	72.24	90.72	73.78	92.76	74.9	91.60
	1.5	77.8	89.28	79.47	92.43	80.67	94.74	82.4	96.87	83.65	95.66
	2.0	78.75	88.92	80.43	92.06	81.65	94.36	83.4	96.48	84.67	95.28
350	0.5	53.81	78.58	54.96	81.35	55.8	83.38	56.99	85.26	57.86	84.19
	1.0	70.53	86.61	72.04	89.66	73.13	91.90	74.7	93.97	75.83	92.79
	1.5	78.76	90.44	80.45	93.63	81.67	95.97	83.42	98.13	84.69	96.90
	2.0	79.72	90.08	81.43	93.26	82.67	95.59	84.44	97.74	85.72	96.52
400	0.5	54.17	79.13	55.33	81.92	56.17	83.97	57.37	85.86	58.24	84.79
	1.0	71	87.22	72.52	90.29	73.62	92.55	75.2	94.63	76.34	93.45
	1.5	79.29	91.08	80.99	94.29	82.22	96.65	83.98	98.82	85.25	97.59
	2.0	80.26	90.72	81.98	93.92	83.22	96.27	85.00	98.43	86.29	97.20
450	0.5	54.37	79.44	55.53	82.24	56.37	84.3	57.58	86.19	58.45	85.12
	1.0	71.26	87.56	72.78	90.64	73.89	92.91	75.47	95.00	76.61	93.81
	1.5	79.58	91.43	81.28	94.66	82.52	97.03	84.29	99.2	85.56	97.97
	2.0	80.55	91.07	82.27	94.28	83.52	96.64	85.31	98.81	86.6	97.58
500	0.5	54.47	79.61	55.64	82.42	56.49	84.48	57.70	86.38	58.57	85.30
	1.0	71.4	87.74	72.93	90.84	74.03	93.11	75.62	95.20	76.77	94.01
	1.5	79.74	91.63	81.45	94.86	82.68	97.23	84.45	99.41	85.73	98.17
	2.0	80.71	91.26	82.44	94.48	83.69	96.85	85.48	99.02	86.78	97.78

During the stage of preliminary desiliconization of the rough concentrate, up to 85% of silicon and 98% of rhenium passed into the solution. Under optimal conditions, the silicate solution had the following composition: $\text{Na}_2\text{O} = 126.5 \text{ g/L}$, $\text{SiO}_2 = 112.8 \text{ g/L}$, $\text{Al}_2\text{O}_3 = 4.0 \text{ g/L}$. The solution was then passed to obtain white soot and for the sorption extraction of rhenium [27–29]. After water leaching, the cake is extracted from the target components. As shown in Table 3, after water leaching, the cake yield decreased an average of 20%, thereby increasing the content of valuable components; for example, copper, from 4.857%, as shown in Table 1, to 6.5%, as shown in Table 6.

Table 6. Chemical composition of cake (residue) after water leaching.

Component	Cu, %	Fe, %	Re, g/t	Si, %	Ag, g/t	Zn, %	Cake Output, %
Content	6.50	7.36	0.43	17.63	132.2	0.45	21.54

The studies on thermochemical enrichment applied to the concentrate from the final tailings with sodium hydroxide at a temperature of 300–350 °C and subsequent water leaching with the extraction of silicon and rhenium in the solution made it possible to extract valuable components without using high-temperature pyrometallurgical processes.

The introduction of an integrated processing of copper-containing tailings, including sintering followed by water leaching, opens up new opportunities for the complete use of resources and minimization of harmful environmental impacts. Further studies are required

to optimize these technologies to improve their efficiency and expand their applications, including the production of silicon, which is in high demand in various industries.

5. Conclusions

The optimal sintering conditions (temperature 350 °C, duration 90 min) promoted the extraction of 84.44% of silicon and 97.44% of rhenium from the rough concentrate into the solution. After water leaching, the cake was subjected to the further extraction of target components. This degree of extraction reduces the need for fresh raw materials and positively affects the production environmental performance because it minimizes the volume of industrial waste disposal.

The partial dependencies of silicon and rhenium extraction into the solution were constructed based on multifactor experiments on the thermochemical enrichment of rough copper concentrates. In this study, mathematical Protodyakonov–Malyshev models of these processes and multifactor nomograms over a wide range of duration temperatures and the ratio of alkali to the concentrate with the determination of maximum recovery rates were developed. The developed multifactor models make it possible to establish optimal intervals of changes in the concentrate sintering parameters, providing high recovery rates (up to 85% of silicon and 98% of rhenium) in the subsequent water leaching.

5.1. Theoretical Significance

The obtained multiplicative multifactor Protodyakonov–Malyshev models can be used to calculate the multifactor space of indicators of silicon and rhenium extraction into the solution with the variation in technologically possible limits of change of each significant factor and with the selection of areas of acceptable and unacceptable values in this space.

The multifactor tabular representation of the summarizing function is the most illustrative and easily formalized representation of computer execution. Therefore, it is worthwhile to conduct independent theoretical and practical studies.

5.2. Practical Significance

Copper concentrator tailings currently occupy a large area owing to the absence of an efficient processing technology. The combined method, which includes flotation and thermochemical processes, allows for the recovery of valuable components, reduces the overall processing costs, and improves the economic profitability of the process. Industrialization of this technology has the potential to be an essential step in sustainable waste management and the provision of secondary raw materials for various industries, including electronics and construction.

The residue obtained after water leaching of the concentrate was sent for further extraction of the target components.

5.3. Strengths and Limitations of the Study

Strengths. This method of processing copper-containing final tailings aims to improve the environment by eliminating dumps in processing plants.

Limitations. The sintering of the rough copper concentrate should occur at a temperature not lower than 300 °C because of the crystallization of NaOH. After thermochemical enrichment, water leaching should occur at temperatures no higher than 60 °C because the solution coagulates.

5.4. Recommendations and Further Actions

Further studies will involve the multifactor analysis of technological operations by identifying zones of optimal process regimes and optimizing leaching technology with

further theoretical justification of the processes to increase their efficiency and expand their application, including the production of silicon demanded in various industries.

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