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Metallurgy



Hydrometallurgical studies on the leaching of copper from man-made mineral formations

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ABSTRACT

The article presents the study results for the processing of industrial waste from copper production at copper smelter Kazakhstan. The samples taken were analysed with the help of X-ray fluorescence and phase analyses which showed that the composition of the studied raw materials was determined as silicate, oxidized, and copper was in a mixed form with a total content of 0.481%, including some in the form of sulfates and sulfides. Mineralogical analysis showed the presence of magnetite, hematite and martite, while copper was present in various mineralogical formations - from magnetic fractions mainly with very fine dusty micron dissemination to native copper and copper minerals. Beneficiation studies performed included flotation and gravity methods. As a result, a concentrate with a copper content of 9.35% was obtained during gravity beneficiation, and a concentrate with a copper content of up to 46% was obtained during flotation. Copper was extracted from beneficiated raw materials with a sulfuric acid leaching method in agitation mode. The solid residue analysis conducted after (cake) leaching also showed the content of the noble metal - gold at the level of 0.47 g/t enabling us to consider its extraction in the future as an additional valuable component.

Keywords: copper, mineralogical analysis, technogenic raw materials, enrichment, leaching.

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Introduction

Technogenic waste from metallurgical production is a promising secondary raw material for obtaining valuable components. Effective processing of mineral raw materials and adjusted technology intended to extract valuable components will enable to involve poor, off-balance raw materials into the production cycle and increase production profitability. Slag is the main waste of copper pyrometallurgical production. Thus, 2–4 tons

of slag from smelting, converting and refining processes are released at copper smelters during production of a ton of copper. And hydrometallurgical, pyrometallurgical and flotation methods are used to process it. The choice of processing method is determined by the composition of the feedstock, fluxes and the technologies used at the enterprises. However, to date scientifically based technologies intended to process slag and other copper production waste have not been developed [[1], [2], [3], [4], [5]].

An analysis of the existing technologies used for processing of waste slag from copper production shows that pyrometallurgical processing methods using reduction smelting are not always suitable for this product. This is due to the presence of various forms of oxides in the slag often impossible to be reduced with carbon during the smelting process. Hydrometallurgical processing of copper slag most often involves sulfuric acid leaching, with the practice to use additional methods to intensify metal extraction. The following slag processing designs are being studied at the moment - leaching of slag with solutions of sulfuric acid and iron sulfate; autoclave ammonia-carbonate leaching of slag in an oxidizing environment; autoclave sulfuric acid leaching. Autoclave nitrate leaching is mainly used for reverberatory and shaft melting slags [[6], [7], [8], [9], [10], [11]].

Bio-leaching studies conducted for reverberatory smelting slag and its beneficiation products showed that the copper extraction into the solution increased up to 82% when bacterial oxidation was used.

The composition and characteristics of raw materials and the use of hydrometallurgical processing methods and processes were studied to involve slag in the technological cycle, and to

develop effective technological solutions [[12], [13], [14], [15], [16]].

Experimental part

Samples were taken from the dumps of man-made mineral formations (MMMMF) of the Kazakhstan smelter to perform the work. The technological sample is composed of vitreous slags which are mainly black, less often brown and dark green. The detailed elemental composition of the initial MMMF sample presented in Table 1 was determined with the help of X-ray fluorescence analysis enabling to capture spectra of elements from oxygen to uranium [[17], [18], [19], [20]]. To involve slag in the technological cycle and develop effective technological solutions, studies of the composition and characteristics of raw materials and the use of methods and methods of hydrometallurgical processing were carried out.

From Table 1 it can be seen that the raw material, under study, slag, belong to complex chemical systems, where the predominant metal is iron 9.796%, significant calcium content is 6.762%, copper content is 0.481%.

Table 1 - Results of X-ray fluorescence analysis of Karsakpay slag sample

Element	Content, %	Element	Content, %	Element	Content, %	Element	Content, %
O	36.386	S	0.093	Cu	0.481	P	0.055
Na	0.926	K	0.305	Zn	0.335	Fe	9.796
Mg	0.517	Ca	6.762	Rb	0.004	Pb	0.202
Al	3.231	Ti	0.271	Zr	0.013	Co	0.020
Si	19.084	Mn	0.247	Sr	0.021	Cl	0.012
						Ba	0.107

Table 2 - Phase composition of the initial slag sample

Component name	Formula	Amount, %
Gedenbergite, sodium	$(Ca_{0,615}Na_{0,385})Fe(Si_2O_6)$	27.3%
Magnetite	Fe_3O_4	15.6%
Aluminum augite	$Ca(Mg,Fe,Al)(Si,Al)_2O_6$	14.5%
Quartz	SiO_2	11.7%
Almandine	$Fe_3(Al_2Si_3O_{12})$	9.1%
Sodium mica, dehydroxylated paragonite, Sodium aluminosilicate	$NaAl_3Si_3O_{11}$	8.2%
Calcium silicate	$CaSiO_3$	6.9%
Calcium-aluminum silicide	$CaAlSi$	6.8%

The phase composition of the slag was determined using a D8 Advance (Bruker), α -Cu apparatus, with a tube voltage of 40 kV and a current of 40 mA. Processing of the obtained diffraction pattern data and calculation of interplanar distances were carried out using EVA software. Sample interpretation and phase search were carried out using the Search/match program using the PDF-2 Powder Diffractometric Database. The results of X-ray phase and chemical analysis are presented in Tables 2, 3.

The data of Tables 1 and 2 show the MMMF specific composition, the noticeable predominance of oxidized forms of phases containing significant amounts of iron, silicon, aluminum and calcium. It has been established that the predominant phases have a silicate form, as well as in the form of oxidized iron using the example of magnetite.

Table 3 - Results of chemical phase analysis of the MMMF sample

No.	Name of components and types of compounds	Content, in %
1	Copper sulfate	0.59
	Copper of oxidized compounds (except for sulfate)	0.44
	Metallic copper	-
	Copper sulfide	0.18
	Total copper content	0.75
2	Iron (II) oxide (ferrous oxide)	3.68
	Iron (III) oxide (iron oxide)	12.08
	Iron metallic	-
	Iron sulfide	1.04
	Total iron	14.06

Mineralogical analyses. A polished thin section ($\varnothing = 25$ mm, m-suspension = 10-15 grams) formed from this material was studied under the Axio Scope.A1 optical microscope. Magnetite, hematite and martite are present in the ore according to literature data. Two samples with the coarseness up to 5-7 centimeters were received for the study of slag - crushed sample up to 200 mesh and gravity concentrate with the coarseness -2,0 mm. The bulk sample material was separated into 4 fractions using a four-pole Sochnev magnet: No. 1. Strongly

magnetic fraction (magnetite), No.2 Medium magnetic fraction (hematite), No.3. Weakly magnetic fraction (glassy slag fragments), iron oxides and hydroxides), No.4 Non-magnetic fraction (amorphous non-metallic glass). A polished thin section was made from the sample, and an artificial polished thin section (briquette) was made from the gravity concentrate. All products were studied under a Leica DM 2500 P microscope. The crushed material of the sample was studied in immersion liquids under a microscope.

The sample is black, with a glassy sheen on chips, fine-grained, with layers of dark gray color of a fine-grained structure. In the immersion preparation, the bulk material of the sample is represented mainly by glass fragments of a light brownish tint, isotropic in crossed nicols (amorphous) (Figure 1, a).

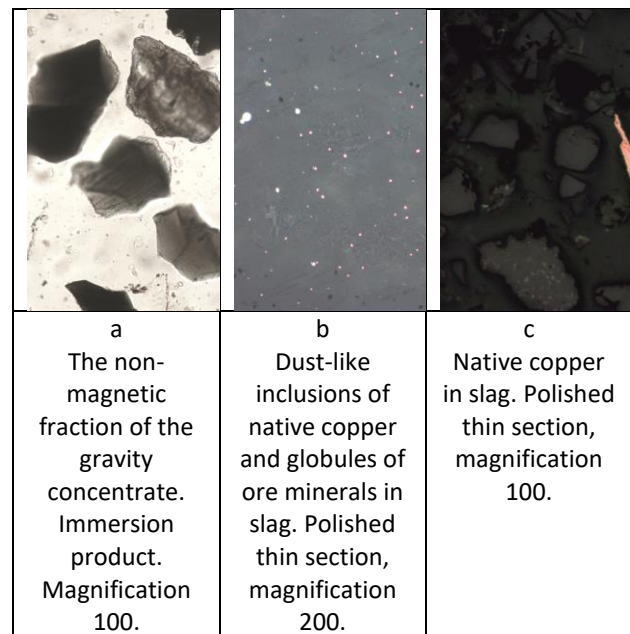


Figure 1 - Polished thin section of MMMF

The sample is represented by fragments of slag (glassy mass) with very fine dust-like micron inclusions of native copper and copper minerals with the size of hundredths and thousandths of mm and perfect rounded globular grains (chalcocine) in the polished thin section in the reflected light. These grains sometimes contain very small inclusions of native copper. The main body of the rock also contains point inclusions of globular-shaped mineral with the size of hundredth and thousandth fractions of mm (Fig. 1, b) along with copper phenocrysts. Copper crystals up to 0.2 mm in size are present in the polished thin section and briquettes of the gravity concentrate (Figure 1, c).

The gravity concentrate consists of glass fragments often with small inclusions of copper minerals and globules of white and light gray mineral. Light minerals are in the form of rounded crystals up to 0.13mm in size. Globular crystals sometimes have heterogeneous taxite coloration with fragments of light gray and beige-pink color (Figure 2, a). Inclusions of native copper are sometimes found in chalcosine crystals (Figure 2, b).

There are crystals of chalcopyrite with a tinge of orange color, characteristic of bornite in the briquette from gravity concentrate (Figure 2, c). Probably they have an intermediate composition between these two minerals (solid solution).

Determination under the microscope needs to be clarified and confirmed on the electron-probe analyzer to clarify the composition of minerals.

main 10-minute flotation mode included the following components: butyl xanthogenate - 150 g/t, foaming agent C7 - 150 g/t, sulfidizer Na_2S - 1000 g/t. The 5-minute control flotation mode provided the following consumption of reagents - butyl xanthogenate - 100 g/t, foaming agent C7 - 50 g/t. Caustic soda was added additionally at the rate of 5 kg/t to the slag slurry for the variant with alkaline environment, and 2 kg/t of sulfuric acid was added for the variant with acidic environment.

Three products of each pH variant were also obtained as a result of flotation beneficiation - main and control concentrates, tailings of flotation beneficiation. When the samples of each product were dried and weighed, they were analyzed for elemental composition presented in Tables 4-7.

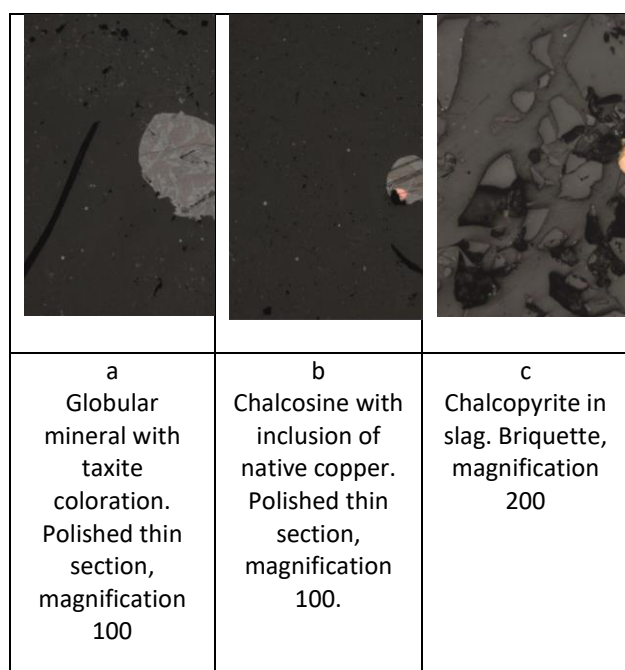


Figure 2 - Mineralogical studies of the copper mineral

Flotation beneficiation of the initial slag sample.

Initial slag was prepared for flotation beneficiation in the same way as for gravity beneficiation. Several crushing and grinding stages were envisaged followed by flotation beneficiation of the initial MMMF, including the main and control flotation.

Besides, three variants of pH of the medium were provided with flotation of initial MMMF. The

Table 4 - Elemental composition of flotation beneficiation products in the initial MMMF sample at pH = 8.4

Element	Element content, %		
	Main concentrate	Control concentrate	Flotation tailings
O	39.14	43.092	36.034
Na	1.036	1.067	0.915
Mg	0.635	0.664	0.501
Al	3.465	3.756	2.949
Si	20.095	22.15	17.565
P	0.057	0.056	0.045
S	0.189	0.122	0.055
K	0.447	0.494	0.417
Ca	8.96	9.546	8.218
Ti	0.457	0.54	0.437
Mn	0.235	0.228	0.214
Fe	11.369	11.119	10.34
Co	0.026	0.0	0.029
Cu	1.277	0.525	0.329
Zn	0.432	0.363	0.317
Sr	0.021	0.022	0.015
Zr	0.015	0.012	0.008
Pb	0.293	0.197	0.187

Table 5 - Elemental composition of flotation beneficiation products of initial slag sample at pH = 10.5

Element	Element content, %		
	Main concentrate	Control concentrate	Flotation tailings
O	41.739	41.806	35.116
Na	1.142	1.115	0.952
Mg	0.647	0.66	0.487
Al	3.653	3.634	2.748
Si	21.49	21.32	16.407
P	0.054	0.054	0.046
S	0.302	0.161	0.055
K	0.44	0.459	0.369
Ca	8.971	8.835	7.916
Ti	0.441	0.466	0.414
Mn	0.215	0.224	0.206
Fe	10.415	10.441	10.234
Co	0.019	0.0	0.02
Cu	1.937	0.769	0.37
Zn	0.393	0.36	0.332
Sr	0.018	0.022	0.017
Zr	0.011	0.011	0.01
Pb	0.274	0.207	0.224

Table 6 - Elemental composition of flotation beneficiation products of initial slag sample at pH = 6.0

Element	Element content, %		
	Main concentrate	Control concentrate	Flotation tailings
O	44.443	44.623	37.795
Na	1.147	1.075	0.986
Mg	0.634	0.609	0.535
Al	3.823	3.865	3.189
Si	22.067	22.137	18.689
P	0.068	0.057	0.047
S	0.214	0.209	0.092
K	0.492	0.454	0.404
Ca	8.922	9.228	8.47
Ti	0.491	0.47	0.453
Mn	0.243	0.216	0.216
Fe	10.535	10.732	10.457
Co	0.0	0.022	0.023
Cu	1.18	1.12	0.476
Zn	0.321	0.334	0.348
Sr	0.017	0.021	0.014
Zr	0.009	0.011	0.009
Pb	0.226	0.203	0.176

Table 7 - Results of flotation beneficiation of initial slag at different parameters of the medium pH

Experiment 1 (initial slag)					
Conditions	Product	weight, g	wt. yield, %	Cu, %	E Cu, %
Main flot. t-10 min, butyl xanth - 150 g/t, C7 -150 g/t, ; Control flot. t-5 min, b.xanth - 100 g/t, C7 - 50.; initial pH - 8,4.	Main concentrate	340.3	17.015	1.3	43.83
	Control concentrate	55	2.75	0.525	2.91
	Combined concentrate	395.3	19.765	1.2	46.75
	Tailings	1604.7	80.235	0.33	53.25
	Initial	2000.0	100.0	0.496	100.0
Experiment 2 (initial slag)					
Conditions	Product	weight, g	weight yield, %	Cu, %	E Cu, %
Main flot. t-10 min, butyl xanth - 150 g/t, C7 -150 g/t, ; Control flot. t-5 min, b.xanth - 100 g/t, C7 - 50.; pH - 10,5.	Main concentrate	128.0	6.4	1.9	26.08
	Control concentrate	24.8	1.24	0.769	2.01
	Combined concentrate	152.8	7.64	1.7	28.09
	Tailings	1847.2	92.36	0.37	71.91
	Initial	2000.0	100.0	0.475	100.0
Experiment 3 (initial slag)					
Conditions	Product	weight, g	wt. yield, %	Cu, %	E Cu, %
Main flot. t-10 min, butyl xanth - 150 g/t, C7 -150 g/t, ; Control flot. t-5 min, b.xanth - 100 g/t, C7 - 50.; pH - 6,0.	Main concentrate	79.7	3.985	1.1	8.61
	Control concentrate	50.2	2.51	1.12	5.43
	Combined concentrate	129.9	6.495	1.1	14.04
	Tailings	1870.1	93.505	0.48	85.97
	Initial	2000.0	100.0	0.518	100.0

The highest copper contents are observed in the main concentrates obtained at alkaline (10.5) and initial (8.4) pH in contrast to the gravity tailings beneficiation products. Copper content reaches 1.937 % in the main concentrate obtained by flotation beneficiation at pH = 10.5, and 0.769 % - in the control concentrate. The main concentrate with copper content of 1.277 % and control concentrate with 0.525 % were obtained at flotation in the initial medium pH = 8.4. Flotation at pH = 6.0 allowed to obtain the main concentrate with copper content of 1.18 % and control concentrate with 1.12 %. The main and control concentrates of all variants differed significantly in mass yield similarly to flotation of gravity concentration tailings. Thus, the highest mass yield (total 19.765 %) was also observed in the variant with initial pH = 8.4. Metal balance and copper recovery were calculated based on the obtained results and masses of beneficiation products as presented in Table 7.

The highest copper recovery of 46.75% was recorded in the experiment with initial pH = 8.4 in

the case of beneficiation of initial slag as in the case with flotation beneficiation of gravity tailings. However, the beneficiation concentrate is inferior only to the concentrate obtained by flotation in alkaline medium in terms of quality in this case. Discrepancy of analysis (Cu 0,481 %) and balance for the initial slag sample is 3,0 % in Experiment 1; 1,25 % in Experiment 2; 7,14 % in Experiment 3. Discrepancies of the calculated balance for the beneficiation products and the analysis results are within the acceptable range for copper raw materials.

Experiments intended to leach initial slag and beneficiation products by agitation method.

Samples of initial slag, concentrate of gravity concentration, main concentrates of flotation concentration of initial slag and gravity tailings were leached with sulfuric acid leaching.

Agitation leaching was performed under the following conditions - 80 % grade - 0.074 mm; weight of the sample 100 g; pH of the solution during

leaching 1.6-2.0; concentration of H_2SO_4 - 2.5 %; leaching duration - 10 hours.

The concentration of sulfuric acid and pH of the medium were monitored in the leaching process, and reagents were added if necessary. Experiments intended to leach initial samples were conducted with the use of agitator stirrers with a rotation speed of 150-200 rpm. Thus, there was a constant decrease in the concentration of sulfuric acid in the initial slag sample up to 0 within the first 4 hours. Initial slag was added to the variant with three times throughout the experiment. Acid addition was required only in the first two hours of leaching in the experiments with concentrate leaching, and further decrease in acidity was not observed.

The solution was separated by filtration and analyzed for copper content at the end of leaching. The cake was analyzed to consider the possible increase in gold content during the removal of impurities by sulfuric acid leaching. The copper recoveries presented in Table 8 were calculated based on the amounts of productive solutions obtained and their corresponding copper concentrations.

The results of agitation leaching experiments showed that the amount of recoverable copper from the initial slag and products of its beneficiation are

at the level of 23.08 and 30.89 %. The highest recovery is observed in the flotation concentrate sample obtained after beneficiation of gravity tailings - 30.83 % but the solution quality is at the level of copper concentration of 0.503 g/l taking into account the relatively low copper content in this concentrate. The maximum concentration of copper is noted in the solution after leaching of flotation concentrate of initial slag - 1.035 g/l resulted in recovery of 27.87 %. Copper content was found at 0.508 g/l, and the recovery was 24.36 % in the solution after leaching of gravity concentrate. The lowest copper concentration was recorded in the solution after leaching of the initial slag at 0.37 g/L, with a final recovery of 23.08 %.

The variant of biochemical leaching included pretreatment of mineral raw materials with a bacterial culture of *A. Ferrooxidans* at pH 2.2-2.4. Subsequently, leaching was carried out with sulfuric acid in a similar mode as with the standard method. The results of leaching after bacterial treatment are presented in Table 9.

Solid sediment of each sample (cake) was analyzed for possible increase in gold content, taking into account all stages of beneficiation and removal of part of acid-soluble impurities in the leaching process. However, an increase in gold content of more than 0.47 g/t was not observed in any variants.

Table 8 - Results of agitation leaching

Sample	Initial Cu, %	Leaching solution		Weight, g	Solution amount, ml	Cu, g/l	E Cu, %
		H_2SO_4 , g/l	S:L				
Initial slag	0.481	25.0	1:4	100	300	0.37	23.08
Gr. conc.slrag	0.73	25.0	1:4	100	350	0.508	24.36
Con. Flot. Gr. tailings	0.57	25.0	1:4	100	350	0.503	30.89
Con. Flot. and slag	1.3	25.0	1:4	100	350	1.035	27.87

Table 9 - Results of agitation leaching after biooxidation

Sample	Initial Cu, %	Leaching solution		Weight, g	Solution amount, ml	Cu, g/l	E Cu, %
		H_2SO_4 , g/l	S:L				
Initial slag	0.481	25.0	1:4	100	310	0.748	48.23
Gr. conc.slrag	0.73	25.0	1:4	100	330	1.3	58.77
Con. Flot. Gr. tailings	0.57	25.0	1:4	100	335	1.15	67.59
Con. flot.	1.3	25.0	1:4	100	330	3.24	82.27

Discussion of the results

The slag sample studies included the study of the slag chemical composition and phase composition, identification of potentially valuable components and search for methods of their recovery, including flotation and gravity concentration, agitation and percolation leaching.

The study of the material composition of slag by fluorescent and X-ray phase analysis confirmed the structure characteristic of the main mass of slag in full. Thus, the analysis results showed elements and phases mainly characteristic of $\text{Si}_x\text{O}_y\text{-Fe}_x\text{O}_y\text{-Al}_x\text{O}_y\text{-Ca}_x\text{O}_y$ -slag systems. The phase analysis found numerous silicates, aluminosilicates, iron oxides, quartz, etc.

Only copper was found in the slag sample in the analysis process for the search of potentially valuable components with the concentration of 0.4-0.5 % in the average. This copper content is acceptable for minerals involved in industrial processing (heap leaching, beneficiation). The contents of other potentially valuable elements are rather low, and it does not enable to consider their possible industrial recovery. Additional analysis for gold content also showed relatively low results at about 0.4 g/t. Gravity concentration experiments showed the possibility to recover only 9.35 % of copper in the concentrate due to the absence of clearly pronounced copper-containing fragments with high density in the slag composition relative to other components of the slag. The maximum recovery of copper in the concentrate reached the level of about 46%, while a significant part of the metal remained in the flotation tailings at flotation beneficiation of both initial samples and gravity tailings. This factor is due to the absence of an acceptable amount of floatable mineral forms of copper in the slag composition.

Agitation leaching from the initial slag sample recovered 23.08 %, while there was a significant absorption of acid and leaching solution that was also characteristic for percolation of crushed slag. The maximum copper recovery in solution during agitation leaching was observed in the variant with flotation concentrate from gravity concentration 30.83 %. However, the solution that was best in terms of concentration quality - 1.035 g/l with recovery of 27.87 %, was obtained by leaching of flotation concentrate from initial slag. The obtained recovery rates of beneficiation and agitation leaching processes, taking into account the

necessary stages of raw material preparation (crushing, grinding) and reagent regimes (xanthogenate, foaming agent, sulfidizer, etc.) indicate the technical and economic inexpediency to process this slag by the above methods. Thus, methods involving costly grinding and beneficiation operations are excluded for effective and expedient processing of this MMMF to obtain finished copper cathode.

The use of the preliminary method of oxidation of mineral samples of enrichment products, as well as the initial slag, allows to significantly increase the degree of copper extraction into the solution in the future. Thus, the concentration of copper during biochemical leaching of the initial slag increases from 0.37 g/l to 0.748 g /l, which gives a final extraction of 48.23%. An increase in the concentration and extraction of copper was more than doubled during the leaching of gravity concentrate – from 24.36 to 58.77%, as well as the concentrate of flotation enrichment of gravity tails – from 30.89 to 67.59%. The maximum increase in the concentration of copper – 3.24 g/l, was detected during biochemical leaching of the flotation concentrate of the initial slag, and the increase in extraction showed a threefold increase from 27.87% to 82.27%.

Conclusions

High efficiency of preliminary bacterial treatment of samples with *A. Ferrooxidans* culture was established during agitation leaching experiments with initial slag samples and products of its beneficiation by standard and biochemical method. Thus, preliminary oxidative treatment of the initial slag sample with bacteria allows increasing the final recovery rate from 23.08 to 48.23 %. Similar bioleaching efficiency was also observed for the leaching of gravity beneficiation slag sample, where the recovery rate was 58.77 % at copper concentration of 1.3 g/L. The use of bacterial oxidation factor for flotation concentrates of slag gravity tailings increased the total recovery from 30.89 % to 67.59 %. Application of bio-oxidation with *A. Ferrooxidans* bacterial culture on initial slag flotation concentrate contributed to a threefold increase in copper concentration and final recovery in solution that amounted to 82.27 %.

Thus, it has been established that the chemical and mineralogical composition of this slag makes it possible to effectively apply the biochemical method

of leaching both on the initial slag and on its enrichment products. In particular, during the leaching of the flotation concentrate, a sufficiently rich solution is observed, which can subsequently be used both for cementation deposition of copper and for extraction extraction by the organic phase. From an economic point of view, a less energy-consuming method of heap leaching with preliminary bacterial treatment may be acceptable for this type of slag, since this technology eliminates the need for preliminary crushing and grinding of mineral raw materials. However, if it is possible to implement effective enrichment methods and the prospect of extracting additional valuable components, a

combined processing technology can be used for these slags of the Karsakpai deposit, including the stages of enrichment, bio-oxidation and leaching.

Conflict of interest. Если Вы (корреспондент автор) согласны, Вы не должны удалять это предложение: On behalf of all the authors, the correspondent author declares that there is no conflict of interest.

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Техногендік минералды түзілімдерден мыс алу бойынша жүргізілген гидрометаллургиялық зерттеулер

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ТҮЙІНДЕМЕ

Мақалада Қарсақпай кен орнының қазақстандық мыс балқыту зауытындағы мыс өндірісінің техногендік қалдықтарын қайта өңдеу бойынша зерттеулердің нәтижелері келтірілген. Сынама үлгілері рентгенофлуоресцентті және фазалық талдаумен талданды. Зерттелетін шикізаттың құрамы силикатты, тотыққан, мыс сульфаттар, сульфидтер түрінде, жалпы мөлшері 0,481% болатындығы анықталды. Минералогиялық талдау магнетит, гематит және мартиттің болатынын көрсетті. Мыс магниттік фракцияларда, негізінен табиғи және мыс минералдарының өте жұқа шаң тәрізді микрондық сеппелілікте болады. Флотация және гравитация әдістерімен байыту бойынша зерттеулер жүргізілді, гравитациялық байыту кезінде құрамында 9,35% мыс бар концентрат, флотация кезінде құрамында 46% дейін мыс бар концентрат алынды. Байытылған шикізаттан мыс алу күкірт қышқылын шаймалау әдісімен, агитациялық режимінде, 30,83% дейін шығарылды. Қоқымның құрамындағы алтынға жүргізілген талдау оның мөлшері 0,47 г/т-дан аспайтынын көрсетті.

Түйін сөздер: мыс, минералогиялық талдау, техногендік шикізат, байыту, шаймалау.

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Гидрометаллургические исследования по извлечению меди из техногенных минеральных образований

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	АННОТАЦИЯ В статье представлены результаты исследований по переработке техногенных отходов медного производства Казахстанского медиплавильного завода, месторождения Карсакпай. Образцы проб были проанализированы рентгенофлуоресцентным, и фазовым анализом, выявлено, что состав исследуемого сырья определяется как силикатный, окисленный, медь находится в форме сульфатов, сульфидов, с общим содержанием 0,481%. Минералогический анализ показал присутствие магнетита, гематита и мартита, медь присутствует в различных магнитных фракциях, преимущественно в очень тонкой пылевидной микронной вкрапленностью меди самородной и медных минералов. Проведены исследования по обогащению методами флотации и гравитации. При гравитационном обогащении получен концентрат с содержанием меди 9,35 %, при флотации получен концентрат с содержанием меди до 46 %. Извлечение меди из обогащенного сырья проводили методом серноокислотного выщелачивания, в агитационном режиме, с извлечением до 30,83 %. Проведенный анализ кека на содержание золота показал содержание не более 0,47 г/т. Ключевые слова: медь, минералогический анализ, техногенное сырье, обогащение, выщелачивание.
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