

ТОРАЙҒЫРОВ УНИВЕРСИТЕТІНІҢ
ҒЫЛЫМИ ЖУРНАЛЫ

НАУЧНЫЙ ЖУРНАЛ
ТОРАЙҒЫРОВ УНИВЕРСИТЕТА

**ҚАЗАҚСТАН ҒЫЛЫМЫ
МЕН ТЕХНИКАСЫ**

2001 ЖЫЛДАН БАСТАП ШЫҒАДЫ



**НАУКА И ТЕХНИКА
КАЗАХСТАНА**

ИЗДАЕТСЯ С 2001 ГОДА

ISSN 2788-8770

№ 3 (2025)

ПАВЛОДАР

**НАУЧНЫЙ ЖУРНАЛ
ТОРАЙГЫРОВ УНИВЕРСИТЕТ**
выходит 1 раз в квартал

СВИДЕТЕЛЬСТВО

о постановке на переучет периодического печатного издания,
информационного агентства и сетевого издания
№ KZ51VPY00036165

выдано

Министерством информации и общественного развития
Республики Казахстан

Тематическая направленность

Публикация научных исследований по широкому спектру проблем
в области металлургии, машиностроения, транспорта, строительства,
химической и нефтегазовой инженерии, производства продуктов питания

Подписной индекс – 76129

<https://doi.org/10.48081/QXJM4576>

Импакт-фактор РИНЦ – 0,216

Импакт-фактор КазБЦ – 0,406

Касенов Асылбек Жумабекович – к.т.н., профессор (главный редактор);
Сулейменов Ансаган Дюсембаевич – PhD (ответственный секретарь);
Шокубаева Зауреш Жанатовна – технический редактор.

Члены редакционной коллегии:

Богомоллов Алексей Витальевич – к.т.н., ассоц. профессор (Павлодар, Казахстан);
Калиакпаров Алтай Гиндуллинович – д.т.н., профессор (Нур-Султан, Казахстан);
Шеров Карибек Тагаевич – д.т.н., профессор (Караганда, Казахстан);
Кажипбаева Галия Тулеуевна – к.т.н., профессор (Павлодар, Казахстан);
Жунусов Аблай Каиртасович – к.т.н., асс. профессор (доцент) (Павлодар, Казахстан);
Зарубежные члены редакционной коллегии:
Baigang Sun – PhD, профессор (Пекин, Китай);
Gabriele Comodi – PhD, профессор (Анкона, Италия);
Jianhui Zhao – профессор (Харбин, Китай);
Khamid Mahkamov – д.т.н., профессор (Ньюкасл, Великобритания);
Magin Lapuerta – д.т.н., профессор (СьюДад Реал, Испания);
Mareks Mezitis – д.т.н., профессор (Рига, Латвия);
Petr Bouchner – PhD, профессор (Прага, Чехия);
Барзов Александр Александрович – д.т.н., профессор (Москва, Россия);
Витвицкий Евгений Евгеньевич – д.т.н., профессор (Омск, Россия);
Иванчина Эмилия Дмитриевна – д.т.н., профессор (Томск, Россия);
Лазарев Владислав Евгеньевич – д.т.н., профессор (Челябинск, Россия);
Мягков, Леонид Львович – д.т.н., профессор (Москва, Россия);
Янюшкин Александр Сергеевич – д.т.н., профессор (Чебоксары, Россия);
Ребезов Максим Борисович – д.с/х.н., профессор (Москва, Россия).
Чайкин Владимир Андреевич – д.т.н., доцент (Сафоново, Россия);

За достоверность материалов и рекламы ответственность несут авторы и рекламодатели
Редакция оставляет за собой право на отклонение материалов
При использовании материалов журнала ссылка на журнал «Наука и техника Казахстана» обязательна

© Торайгыров университет

D. B. Kassymova¹, *R. V. Sapinov², M. A. Adilkanova³

^{1,2,3}D. Serikbayev EKTU, Republic of Kazakhstan, Ust-Kamenogorsk;

²Toraighyrov University, Republic of Kazakhstan, Pavlodar;

¹VNIItsvetmet, Republic of Kazakhstan, Ust-Kamenogorsk.

¹ORCID: <https://orcid.org/0009-0005-0937-8477>

²ORCID: <https://orcid.org/0000-0001-9334-5806>

³ORCID: <https://orcid.org/0000-0002-8193-116X>

*e-mail: sapinov.r@teachers.tou.edu.kz

SELECTION OF OPTIMAL CYANIDE LEACHING MODES FOR GOLD FROM ORE DEPOSITS IN THE REPUBLIC OF KAZAKHSTAN

The processing of gold-copper ores with high copper and gold content embedded in sulfide minerals using cyanide solutions is associated with a number of difficulties. Copper, when reacting with cyanide, causes its increased consumption, and the embedding of gold in sulfide minerals causes gold losses with tailings. Therefore, selecting the optimal leaching conditions is necessary for the rational use of reagents, which leads to a reduction in the cost of hydrometallurgical processes for cyanide leaching of gold-copper ores and a reduction in environmental risks. The paper investigates the possibility of processing gold-copper ore with a high copper content and gold incorporated into sulfide minerals (pyrite, chalcopyrite). The aim of the work is to establish the optimal parameters for cyanide leaching by conducting tests in bottle agitators. As a result of the experiments, it was found that as the degree of ore grinding increases, gold extraction increases and cyanide consumption decreases slightly. The addition of coal during leaching significantly increases gold recovery, up to 7 %. It was noted that high sodium cyanide consumption cannot be reduced without a significant decrease in gold recovery. Au \approx 20 % reduction in sodium cyanide consumption also reduces gold recovery by \approx 20 %. A laboratory-industrial experiment showed that after 16 hours of ore leaching, 85.8% of the gold is transferred into solution, and after 18 hours, 86.7 %. Further prolongation of the process does not increase the gold recovery rate.

Keywords: gold-copper ore, optimal parameters, cyanide leaching, gold, copper.

Introduction

The selection of optimal leaching modes is necessary for the rational use of resources, which leads to a reduction in the cost of hydrometallurgical processes [1;2] and a decrease in environmental risks [3;4]. Increased cyanide consumption is known to be mainly associated with the presence of impurities in the raw material. For example, the presence of copper in the raw material leads to cyanide overspending [5] due to reactions between cyanide and copper [6] according to reaction (1).



There are various approaches to the economic optimization of cyanide leaching. Some researchers suggest precipitating copper in order to obtain an additional commercial product [7;8] with subsequent regeneration of the solution [9]. One of the most difficult types of raw materials in terms of gold extraction by cyanidation is raw materials in which gold is embedded in copper-containing minerals. Gold particles are included in sulfide ore (pyrite, arsenopyrite, and chalcopyrite) or embedded in it. As a result, gold is difficult to extract even after fine grinding [10]. This article explores the possibility of processing gold-copper ore with a high copper content and gold embedded in sulfide minerals (pyrite, chalcopyrite).

The aim of the work is:

– to establish the optimal parameters for cyanide leaching of gold-copper ore by conducting tests in bottle agitators in an open cycle on a sample with a total weight of 30 kg;

– laboratory-industrial leaching of ore at the VNIItsvetmet pilot plant to verify and refine process parameters and obtain the required volumes of intermediate products from an industrial sample with a total weight of 2070 kg.

Materials and methods

All experiments were carried out in D. Serikbayev “and ‘Eastern Research Mining and Metallurgical Institute of Nonferrous Metals ’VNIItsvetmet”, located in the Republic of Kazakhstan, in the city of Ust-Kamenogorsk. The object of the study - gold-copper ores of the deposit of Kazakhstan (2070 kg). Mass-spectrometer ICP-MS 7500cx of “Agilent technologies” (USA) was used to study the chemical composition of raw materials. To study the material composition of the material under study, X-ray diffractometer BRUKERD8 ADVANCE was also used to analyze the X-ray diffractometer, which is a common method for determining the qualitative composition of the material. In order to achieve the objectives of the study, the chemical and phase chemical compositions of the incoming sample were determined. Further to study the material composition of the investigated material, XRD was also carried out. Particle size analysis was performed using a FRITCH ANALYSETTE 22 laser analyzer. Tap water was used in the studies, the chemical composition of which is shown in Table 1.

Table 1 – Chemical composition of the source water used in the studies

Indicator	Contents
Na, mg/dm ³	13
K, mg/dm ³	<10
Al, mg/dm ³	0.0097
Sulfates, mmol/dm ³	25.28
chloride ion, mmol/dm ³	24.68
Ca, mg/dm ³	28
Mg, mg/dm ³	7.88
Insoluble residue, mg/dm ³	0.017

Total alkalinity, mg/dm ³	2.5
Overall stiffness, mg-eq/dm ³	2.12
pH	7.25
Dry residue, mg/dm ³	0.2782
Nitrites, mg/dm ³	<0.002
Ammonium chloride, mg/dm ³	<0.05

Testing methodology in bottle agitators. The ore was ground for leaching in a batch ball mill. The ball mill was loaded with 50 kg of 40 mm and 25 mm diameter balls and 150 kg of 20 mm diameter balls. The sample weight was 30 kg. S/L = 1/1. The tests were conducted on samples ground to a size of 80 %, 85 %, and 90 % of the minus 0.071 mm class, with a sodium cyanide concentration of 0.05 % in the leaching solution, with and without the addition of coal to the process. The crushed ore was leached for 24 hours. The main parameters of the bottle leaching process were: pulp density – 40 %; pulp pH 10.0-10.5; sodium cyanide concentration in the initial solution 0.05%. Ore leaching tests were carried out with and without activated carbon. Activated carbon (NORIT) was used at a rate of 10 % of the weight of the initial sample. Before the start of the tests, the carbon was washed to remove sludge. Samples of the product solutions were taken every 2, 4, 6, and 24 hours after the start of the tests. The pH and sodium cyanide concentration were determined in each solution sample. The pH of the product solution was maintained at 10.5. When the sodium cyanide concentration in the leaching solution fell below 50 % of the initial content, sodium cyanide was added to the pulp in the form of a 10 % solution, the consumption of which was recorded. After the tests were completed, the pulp was filtered. The final volume of the solutions was measured, and samples were taken to determine the pH, gold, silver, and copper content, and sodium cyanide concentration. The coal and solid residues from the leaching process were washed with water, dried, and subjected to chemical analysis for gold, silver, and copper content.

Methodology for laboratory-industrial ore leaching at the VNIItsvetmet pilot plant. The grinding stage is similar to that described above for tests conducted in bottle agitators. After unloading the mill into an intermediate collection tank, the pulp was pumped into a 1.5 m³ agitator tank, where it was adjusted to the required density by adding water. The agitator tank, in which the cyanide leaching of the crushed ore from the Yubileynoye deposit was carried out at the pilot plant, is equipped with dispersers for air supply and a mixer for stirring. The initial concentration of sodium cyanide in the leaching solution was 0.05 %, and the solution was replenished to the initial concentration of sodium cyanide 2, 4, 6, and 8 hours after the start of the process. The total leaching time was 24 hours. Activated carbon was not added to the process. To determine the optimal leaching time, samples were taken to determine the gold, silver, and copper content in the cake after 16, 18, 20, and 24 hours. At the end of leaching, the leaching pulp was fed to a Nutsch filter for filtration. Samples were taken to determine the content of gold, silver, copper, and sodium cyanide. The solid residues from leaching

were washed with water, dried, and subjected to chemical analysis for gold, silver, and copper content.

Results and discussion

1 Study of the composition and physical properties of ore

The particle size distribution of the ore at the selected optimal grinding time (60 min) for leaching is shown in Table 2.

Table 2 – Results of sieve analysis of crushed ore

Size class, mm	Class exit, %
+0.140	0.8
-0.140+0.100	3.6
-0.100+0.071	5.1
-0.071	90.5
Total:	100.0

The summary results of the particle size analysis of the initial ore sample of minus 0.071 mm are presented in Table 3.

Table 3 – The summary results of the particle size analysis of the initial ore sample of minus 0.071 mm

Size class, mm	Class exit, %
+0.140	0.8
-0.140+0.100	3.6
-0.100+0.071	5.1
-0.071+0.045	5.6
-0.045+0.038	5.3
-0.038+0.020	31.3
-0.020+0.010	22.9
-0.010+0.005	8.4
-0.005	17.0
Total:	100.0

Content of gold in the sample amounted to – 1.2 g/t. The content of silver also amounted to – 1.2g/t. The total amount of copper amounted to 0.15 %. Iron content amounted to 5.10 %. The total amount of sulfur was 0.34 %. The main ore mineral in the sample is magnetite. Secondary ore minerals are pyrite, chalcocopyrite, single grains of bornite, chalcocine, and covellite. The main ore mineral in the sample is magnetite. Secondary ore minerals include pyrite, chalcocopyrite, and isolated grains of bornite, chalcocite, and covellite. Magnetite is represented by irregularly shaped grains. It accounts for 53.3 % of the sample (of 100 % of all ore minerals) and forms aggregates with pyrite and non-metallic substances. Chalcocopyrite is represented by amoeboid, irregularly shaped grains. It accounts for 10.2 % of the sample. It forms aggregates with pyrite, bornite, and chalcocite. Pyrite is represented by irregularly shaped grains. It accounts for 27.9% of the sample. It forms aggregates with chalcocopyrite and magnetite. The composition of the ore is described in more detail in the work [11]. The results of

studying the material composition of the material under study using X-ray structural analysis showed the presence of the following non-metallic minerals: quartz, albite, magnetite, clinocllore, orthoclase and calcite.

2 Results of tests conducted in bottle agitators

The main summary results of each of the bottle cyanide leaching tests performed are presented in Table 4. The data presented in Table 4 were previously reported in a study on the sacred problem of copper precipitation from cyanide solution [8, p.4].

Table 4 – Main results of leaching a sample of gold-bearing ore in bottle shakers [8, p.4]

№	Ore Size	Coal	Composition of the Cake			Leaching Rate, %			NaCN Consumption, kg/t Ore
			Au, g/t	Ag, g/t	Cu, %	Au	Ag	Cu	
1	P ₈₀ =0.071 mm	-	0.26	0.75	0.1088	78.3	37.5	27.5	1.98
2		+	0.18	0.44	0.1138	85.0	63.3	24.1	2.40
3	P ₈₀ =0.067 mm	-	0.19	0.60	0.1086	84.2	50.0	27.6	2.16
4		+	0.17	0.48	0.1102	85.8	60.0	26.5	2.53
5	P ₈₀ =0.057 mm	-	0.20	0.72	0.1096	83.3	40.0	26.9	2.44
6		+	0.16	0.46	0.1084	86.7	61.7	27.7	2.57
7		+*	0.15	0.49	0.1104	87.5	59.2	26.4	2.45

Note: * denotes that in experiment No. 7, coal entered the process 6 h after leaching began.

In experiment №. 6, the distribution of gold by size class was determined. The results are shown in Table 5.

Table 5 – Distribution of gold by size class in the residues from leaching of gold-bearing ore (test 6, Table 2)

Size class, mm	Class exit, %	Au content in cake, g/t	Au distribution, %
+0.071	5.3	0.24	7.8
-0.071; +0.045	18.8	0.23	26.7
-0.045	75.9	0.14	65.5
Total:	100.0	0.16	100.0

Table 6 shows the chemical composition of the final solutions obtained by leaching ore with a particle size of P₉₀ = 0.071 mm with and without the addition of coal in the bottle cyanide leaching process (cakes from tests 5 and 6 shown in Table 4).

Table 6– Gold, silver, and impurity content in final solutions of bottle cyanide leaching of gold-bearing ore sample with a particle size of P90 = 0.071 mm

Solution from exp. (Table 2)	Content, g/dm ³										
	Au, mg/dm ³	Ag, mg/dm ³	Cu	Fe	S	As	Ca	Mg	Al	Mn	Zn
№5	1.1	0.20	0.18	0.08	0.2	<0.001	<0.001	0.0015	<0.001	<0.001	<0.001
№6	0.016	<0.05	0.17	0.01	0.2	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001

A series of experiments was also conducted to determine the dependence of gold extraction into solution on sodium cyanide consumption. The tests were carried out without the addition of activated carbon, at a pulp density of 40%, a sodium cyanide concentration in the initial solution of 0.05%, and a leaching time of 20 hours. The test results are presented in Table 7. The data presented in Table 7 were previously reported in a study on the sacred problem of copper precipitation from cyanide solution [8, p.5].

Table 7 – Ore leaching results [8, p.5]

№	Supply NaCN, kg/t					NaCN content in the final solution, %	Leaching content in cake			Extraction into solution, %		
	0 h	2 h	4 h	6 h	Total		Au, g/t	Ag, g/t	Cu, %	Au	Ag	Cu
1	0.75	-	-	-	0.75	<0.01	1	1.2	0.12	16.7	-	20.0
2	0.75	0.67	-	-	1.42	<0.01	0.7	1.0	0.12	45.0	16.7	20.0
3	0.75	0.67	0.59	-	2.01	<0.01	0.4	0.8	0.11	66.7	36.7	26.7
4	0.75	0.67	0.59	0.56	2.57	0.01	0.2	0.7	0.11	83.3	40.0	26.7

3 Conducting laboratory-industrial ore leaching at the VNIItsvetmet pilot industrial plant

The results of the study of the dependence of the degree of extraction of gold, silver, and copper into the leaching solution on the duration are shown in Table 8.

Table 8 – Dependence of the degree of extraction of gold, silver, and copper into the leaching solution on duration

Duration, h	Leaching content in cake			Extraction into solution, %		
	Au, g/t	Ag, g/t	Cu, %	Au	Ag	Cu
16	0.17	0.9	0.12	85.8	25.0	20.0
18	0.16	0.8	0.13	86.7	33.3	13.3
20	0.17	0.8	0.11	85.8	33.3	26.7
24	0.26	0.9	0.11	78.3	25.0	26.7

The results of the particle size distribution of the second cycle leaching cake, performed using sieve analysis and analysis on a laser particle size analyzer, are shown in Table 9.

Table 9 – Summary results of particle size analysis of the leaching cake from the ore sample

Size class, mm	Class exit, %
+0.140	0.2
-0.140+0.100	1.9
-0.100+0.071	3.0
-0.071+0.045	23.0
-0.045+0.038	9.0
-0.038+0.020	30.4
-0.020+0.010	15.6
-0.010+0.005	5.3

-0.005	11.6
Total:	100.0

The main results of ore leaching at the pilot plant are shown in Table 10.

Table 10 – Results of ore leaching at the VNIItsvetmet pilot plant

Loaded ore, kg	Pulp density, %	Contents in the cake			Extraction into solution by cake, %			Content in solution			NaCN consumption, kg/t of ore
		Au, g/t	Ag, g/t	Cu, %	Au	Ag	Cu	Au, mg/dm ³	Ag, mg/dm ³	Cu, g/dm ³	
720	39.8	0.19	0.92	0.101	84.2	23.3	27.7	0.65	0.40	0.28	2.72
870	39.8	0.26	0.90	0.101	78.3	25.0	27.5	0.70	0.49	0.34	2.40
480	41.0	0.21	0.94	0.101	80.0	21.7	27.6	0.78	0.24	0.59	2.49

Conclusions

As the degree of ore grinding increases, gold recovery increases: at an ore size of $P_{80}=0.071$ mm, gold recovery in solution is 85.0 %; at a size of $P_{80}=0.067$ mm (85 % class minus 0.071 mm) – 85.8 %; with a particle size of $P_{80}=0.057$ mm (90 % class minus 0.071 mm), gold extraction into the solution is 86.7%. Cyanide consumption during ore leaching depends to a lesser extent on the degree of grinding and is 2.4; 2.53 and 2.57 kg/t for ore sizes of 80, 85, and 90 % class minus 0.071 mm, respectively. The difference in gold extraction during ore leaching without and with the addition of angle was 6.7; 1.6 and 3.4 % for ore sizes of 80, 85, and 90 % minus 0.071 mm, respectively. Silver extraction also depends on the size of the ore and the presence of coal in the leaching pulp and varied depending on the leaching conditions from 37.5 to 61.7 %. Copper recovery is practically independent of ore size under the studied conditions and ranges from 24.1 % to 27.7 %. The distribution of gold by size class in the leaching cake of ore with a size of 90 % of the minus 0.071 mm class showed an increased gold content in the large classes (in the plus 0.071 mm class, the gold content was 0.24 g/t, with an average gold content in the cake of 0.16 g/t), which indicates the possibility of increasing gold recovery with even finer grinding. The leaching solutions showed elevated copper (0.17–0.18 g/dm³), iron (0.010–0.084 g/dm³), and sulfur (0.2 g/dm³) content, which may have a negative impact on closed-cycle ore processing. The high consumption of sodium cyanide cannot be reduced without a significant decrease in gold extraction: when the consumption of sodium cyanide is reduced from 2.5 to 2.01 kg/t, gold extraction decreases from 83.3 to 66.7 %, while the residual content of sodium cyanide does not exceed 0.01 %.

During ore leaching tests conducted at the VNIItsvetmet pilot plant, the following results were achieved:

– When studying the dependence of gold extraction into solution on duration, it was noted that after 16 hours of ore leaching, 85.8 % of the gold had been transferred into solution, and after 18 hours, 86.7 %. Further increases in the duration of the process did not result in any increase in gold extraction.

– The maximum gold extraction into the solution was 84.2 % with a sodium cyanide consumption of 2.72 kg/t of ore.

– Copper extraction into the solution in three cyanide leaching operations was 27.5–27.7 %.

During the tests, the required number of samples of grinding and leaching pulps were selected. The entire volume of leaching solutions was preserved for further sorption studies.

REFERENCES

1 **Han, J., Dai, S., Deng, J., Que, S., Zhou, Y.** Technology for Aiding the Cyanide Leaching of Gold Ores // *Separations*. – 2024. – № 11. – P. 228.

2 **Kameda, M.** Fundamental Studies on Dissolution of Gold in Cyanide Solutions. II: On Equations of Reactions and Effects of Cyanide Strength and Other Variables on Dissolution Rate : Ph.D. Thesis, Tohoku University. – Miyagi, Japan, 1949. – P. 223–230.

3 **Donato, D. B., Nichols, O., Possingham, H., Moore, M., Ricci, P. F., Noller, B. N.** A critical review of the effects of gold cyanide-bearing tailings solutions on wildlife // *Environ. Int.* – 2007. – № 33. – P. 974–984.

4 **Ya, X., Weishi, L., Qifei, H., Yuqiang, L., Jingcai, L., Li, L., Dahai, Y.** Long-term degradation characteristics of cyanide in closed monofills and its effects on the environment and human health: Evidence from nine landfill sites in northern China // *Science of The Total Environment*. – 2022. – № 839. – P. 156269

5 **Yilmaz, E., Yazici, E.Y., Ahlatci, F., Celep, O., Deveci, H.** Precipitation of copper from cyanide leach solutions using sodium dimethyldithiocarbamate (SDDC) // *Hydrometallurgy*. – 2021. – № 202. – P.105610.

6 **Dai, X., Simons, A., Breuer, P.** A review of copper cyanide recovery technologies for the cyanidation of copper containing gold ores // *Minerals Engineering*. – 2012. – № 25. – P. 1–13.

7 **Estay, H.** Designing the SART process – A review // *Hydrometallurgy*. – 2018. – № 176. – P.147–165

8 **Kassymova, D., Sapinov, R., Kushakova, L., Kulenova, N., Shoshay, Z., Adylkanova, M.** Optimization of Copper Recovery from Cyanide Leaching Solutions Used in Gold–Copper Ore Processing Using Probabilistic–Deterministic Experimental Design // *Processes*. – 2025. – № 13. – P.61.

9 **Estaya, H., Gim-Krumma, M., Serichea, G., Quilaqueoa, M., Barrosa L., Ruby-Figueroa R., Romeroc J., Troncoso E.** Optimizing the SART process: A critical assessment of its design criteria // *Minerals Engineering*. – 2020. – № 146. – P.106116.

10 **Lin, Y., Hu, X., Zi, F., Chen, Y., Chen, S., Li, X., Li, J., Jiang, Y., Zhang, Y.** Rapid gold cyanidation from a sulfur-high and arsenic-high micro-fine concentrate via facile two-stage roasting pre-treatment // *Minerals Engineering*. – 2022. – № 190. – P. 107938.

11 Kassymova, D.B., Sapinov, R.V., Kulenova, N.A., Adilkanova, M. A. Study of the material composition of gold-copper ores from kazakhstani deposits // Science and technology of Kazakhstan. – 2025– № 1. – P. 248–258.

Received 28.07.25.

Received in revised form 31.07.25.

Accepted for publication 13.08.25.

Д. Б. Касымова^{1,*}, Р. В. Сапинов², М. А. Адильканова³

^{1,3,4}Д. Серікбаев атындағы Шығыс Қазақстан техникалық университеті,
Қазақстан Республикасы, Өскемен қ.;

²Торайғыров университеті, Қазақстан Республикасы, Павлодар қ.;

³ШҒЗТКМИТүсмет, Қазақстан Республикасы, Өскемен қ.

28.07.25 ж. баспаға түсті.

31.07.25 ж. түзетулерімен түсті.

13.08.25 ж. басып шығаруға қабылданды.

ҚАЗАҚСТАН РЕСПУБЛИКАСЫНЫҢ КЕНІНЕН, КЕН ОРНЫНАН ЦИАНИДПЕН АЛТЫНДЫ ШАЙМАЛАУДЫҢ ОҢТАЙЛЫ РЕЖИМДЕРІН ТАҢДАУ

Цианид ерітінділерімен сульфидті минералдарға шашыраған мыс пен алтынның көп мөлшері бар алтын мыс кендерін өңдеу бірқатар қиындықтармен байланысты. Сонымен, мыс цианидпен әрекеттесіп, оның шығынын жоғарылатады, ал алтынның сульфидті минералдарға қосылуы алтынның құйрығымен жоғалуына әкеледі. Осылайша, оңтайлы шаймалау режимдерін таңдау реагенттерді ұтымды пайдалану үшін қажет, бұл алтын мыс кендерін цианидті шаймалаудың гидрометаллургиялық процестерінің арзандауына және экологиялық қауіптердің төмендеуіне әкеледі. Жұмыста құрамында мыс мөлшері жоғары алтын кенін және сульфидті минералдарға (пирит, халькопирит) ендірілген алтынды қайта өңдеу мүмкіндігі зерттеледі. Жұмыстың мақсаты-бөтелке агитаторларында сынақтар жүргізу арқылы цианидті шаймалаудың оңтайлы параметрлерін анықтау. Жүргізілген тәжірибелер нәтижесінде кенді ұнтақтау дәрежесі жоғарылаған сайын алтын алу артып, цианидтің шығыны біршама төмендейтіні анықталды. Шаймалау кезінде көмір қосу арқылы алтын алу айтарлықтай артады, 7% дейін. Натрий цианидінің жоғары шығынын алтын өндірудің айтарлықтай төмендеуінсіз төмендетуге болмайтындығы атап өтілді. Натрий цианидінің шығынын $\approx 20\%$ -ға төмендету алтынның өндірілуін $\approx 20\%$ -ға төмендетеді. Зертханалық-өнеркәсіптік тәжірибе көрсеткендей, 16 сағат кенді шаймалағаннан кейін алтынның ерітіндіге ауысуы 85,8%-ға, 18 сағаттан кейін-86,7%-ға жетеді. Процесс ұзақтығының одан әрі ұлғаюымен ерітіндіге алтын алудың ұлғаюы байқалмайды.

Кілтті сөздер: алтын мыс кендері, оңтайлы параметрлер, цианидті шаймалау, алтын, мыс.

Д. Б. Касымова¹, *Р. В. Сапинов², М. А. Адильканова³

^{1,3,4}ВКТУ имени Д. Серикбаева, Республика Казахстан, г. Усть-Каменогорск;

²Торайгыров университет, Республика Казахстан, г. Павлодар;

³«ВНИИцветмет», Республика Казахстан, г. Усть-Каменогорск.

Поступило в редакцию 28.07.25.

Поступило с исправлениями 31.07.25.

Принято в печать 13.08.25.

ПОДБОР ОПТИМАЛЬНЫХ РЕЖИМОВ ЦИАНИДНОГО ВЫЩЕЛАЧИВАНИЯ ЗОЛОТА ИЗ РУДЫ МЕСТОРОЖДЕНИЯ РЕСПУБЛИКИ КАЗАХСТАН

Переработка золотомедных руд с повышенным содержанием меди и золота вкрапленного в сульфидные минералы цианидными растворами, связана с рядом трудностей. Так медь, вступая в реакцию с цианидом вызывает его повышенный расход, а вкрапление золота в сульфидные минералы вызывает потери золота с хвостами. Таким образом подбор оптимальных режимов выщелачивания необходим для рационального использования реагентов, что ведет к удешевлению гидрометаллургических процессов цианидного выщелачивания золотомедных руд и уменьшению экологических рисков. В работе исследуется возможность переработки золотомедной руды с высоким содержанием меди, и золота внедренного в сульфидные минералы (пирит, халькопирит). Целью работы является установление оптимальных параметров цианидного выщелачивания путем проведения тестов в бутылочных агитаторах. В результате проведенных экспериментов было установлено что при увеличении степени измельчения руды извлечение золота возрастает, а расход цианида несколько снижается. Добавлением угля при выщелачивании существенно увеличивает извлечение золота, до 7%. Отмечено что высокий расход цианида натрия не может быть снижен без существенного снижения извлечения золота. Снижение расхода цианида натрия на $\approx 20\%$ также снижает извлечение золота на $\approx 20\%$. Лабораторно-промышленный эксперимент показал что уже после 16 часов выщелачивания руды достигается переход золота в раствор на 85,8%, через 18 часов – на 86,7%. При дальнейшем увеличении продолжительности процесса повышения извлечения золота не происходит.

Ключевые слова: золотомедные руды, оптимальные параметры, цианидное выщелачивание, золото, медь.

Теруге 27.08.25 ж. жіберілді. Басуға 30.09.25 ж. қол қойылды.

Электрондық баспа

5,07 Mb RAM

Шартты баспа табағы 1,09 Таралымы 300 дана. Бағасы келісім бойынша.

Компьютерде беттеген: Е. Е. Калихан

Корректорлар: А. Р. Омарова, Д. А. Кожас

Тапсырыс № 4434

«Toraighyrov University» баспасынан басылып шығарылған

Торайғыров университеті

140008, Павлодар қ., Ломов көш., 64, 137 каб.

«Toraighyrov University» баспасы

Торайғыров университеті

140008, Павлодар қ., Ломов к., 64, 137 каб.

67-36-69

e-mail: kereku@tou.edu.kz

nitk.tou.edu.kz