Research into the processing of gold-containing technogenic waste

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Abstract. This study investigates the processing of technogenic waste generated by the gold extraction plant at the Navoi Mining and Metallurgical Combine. The research focuses on gold-bearing technogenic waste, with its quality assessed based on the content of valuable components, additives, accompanying elements, particle size, and moisture content. The chemical and mineralogical composition, as well as the concentrations of valuable components in the waste, were determined. Various enrichment methods were applied, including flotation using different reagents and gravity separation with centrifugal concentrators and shaking tables, to process the mixed-composition technogenic waste. The optimal flotation reagent and regime for enriching gold-bearing technogenic waste were established. These findings contribute to the development of efficient processing methods for technogenic waste from gold extraction operations.

1 Introduction

The aim of the research work was to extract valuable components by processing technogenic waste. To extract valuable components, a flotation enrichment method was carried out using various flotation reagents in various reagent modes, using a 3-stage flotation process, as well as gravity enrichment methods using a KNELSON centrifugal concentrator and a concentration table.

Flotation enrichment (flotation) is a mineral enrichment process based on the selective attachment of mineral particles at the boundary of two phases. Depending on the phases involved in the process, the flotation process is divided into foam, film, oil, impermeable solid surface and oil surface types. The most widely used flotation method today is the foam flotation process, which can enrich almost all types of minerals. The flotation process is carried out in an enrichment device known as a flotation machine [1].

Flotation agents are substances that provide high selectivity, stability, efficiency, and acceleration of the flotation process when sorting mineral particles by flotation.

Different ores flotate differently. Sulfide minerals can be easily separated from non-sulfide minerals by flotation. Oxide ores, which are formed by oxidation and selective melting of sulfide ores, have poor flotation properties and cannot be flotated without first being sulfidized [2].

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In flotation, the size of the initial product must be such that the loose rock minerals to which the valuable component particles are attached are completely freed, and the size of the flotation particles must be compatible with the buoyancy force of the air bubbles [3].

Typically, the particle size in the flotation process is between 0.02-0.5 mm. The maximum size of the flottable mineral particles depends on their hydrophobicity and shape. When crushing the ore before flotation, it is necessary to ensure that the initial flotation does not contain large particles that cannot be flotated, as well as sludge with a size of less than 0.02 mm, which sharply increases the separation and increases the consumption of reagents [4].

When the density of the slurry is high, its degree of bubble saturation decreases, flotation of large mineral particles deteriorates, and the quality of the concentrate decreases. When it is required to obtain a high-quality concentrate, flotation is carried out in a more liquid slurry. An increase in temperature in many cases has a positive effect on the flotation process. In this case, the solubility of a number of reagents (in particular, fatty acids and soaps) increases, and their consumption decreases. At the same time, this is not observed when xanthates are used as a collector, and it is advisable to heat the plant only in winter [5].

2 Materials and methods

This research work aims to extract valuable components by processing gold-bearing technogenic waste. In order to extract valuable components, the flotation and gravitation enrichment method was carried out.

Flotation studies were conducted on laboratory flotation machines. For this purpose, the Ginsvetmet machine with mechanical foam removal and air supply regulation device and the Essa pneumomechanical flotation machine are convenient, the chamber volume of the machine is 3.6 and 3l, respectively, which can be used for flotation of 1-2 kg of ore samples. For re-cleaning operations, as well as for small samples, machines made of less volumetric plexiglass or another material with chamber volumes 1 were used; 0.5 and 0.2 l.

Before loading, the machine was washed with water, the electric motor was started, the air duct was closed, the slurry was silted in the receiver, and then poured into the machine compartment where mixing was carried out. If a layer of settled material remains at the bottom of the receiver, then a small amount of water is added, and the settled sediment is suspended by a stream of water.

Water is poured into the machine in such a quantity that the pulp freely circulates from the mixing chamber to the pyramidal chamber. It is not advisable to add excess water, because when the air intake opens, the volume of pulp in the machine increases and conditions are created for pulp to flow out of the machine.

Then reagents are added in a specific order, stirred for the required time, and air is introduced. If the slurry level turns out to be low, the machine is filled with water in such a quantity that the foam-removing oar touches the slurry level. During the removal of foam, a large amount of water is removed with it, and the slurry level decreases. To maintain a constant level, water with approximately the same alkalinity as the slurry at the beginning of flotation is added. During the experiment, it is necessary to wash the foam adhering to the walls and other parts of the chamber several times.

The completion of the flotation process is determined by the change in the color of the foam and the results of examining the foam samples on a watch glass. At the beginning of flotation, the foam acquires the color of the flotation mineral. After the completion of flotation of this material, the foam becomes loose, white, or acquires the next color in terms of floatability.

The experiment on the concentration table was set up in a closed cycle, from four ten-kilogram samples. During the initial grinding, the sample was fed to a laboratory jigging machine of the MOD 0.02 SKL brand, the jigging concentrate was transferred to the concentration table of the 51KTs brand (deck area 0.41 m²), the table tails with the subsequent sample were sent for jigging.

For comparison, experiments were conducted on a Knelson centrifugal concentrator. The experiments were carried out with staged and direct enrichment of ore (1st experiment - one concentration with initial grinding, 2nd experiment - one concentration with grinding of 80% of the -0.074+0 mm class, and 3rd experiment staged enrichment with threefold concentration - 1st at 65% and 2nd at 80% of the finished class; 3rd control).

3 Results and discussion

The object of the research is the technogenic waste of the gold extraction workshop of the Navoi Mining and Metallurgical Combinat. Four flotation enrichment experiments were conducted to process GES waste. In the experimental work, 3-stage flotation was carried out in different reagent modes. Before carrying out the experimental work, the mineralogical and chemical composition of the initial product was studied. The results obtained are shown in Tables 1 and 2.

Table 1. Results of the chemical composition analysis of the sample.

Content, %								
Au, g/t	Ag, g/t	S_{common}	Ss	Fe_{common}	C_{org}	CO_2	As	Sb
2.57	0.63	1.7	0.49	3.6	0.12	0.98	<0.089	0.018

Table 2. Results of the complete chemical composition analysis of the sample.

Elementary content, %									
SiO_2	Al_2O_3	TiO_2	FeO	CaO	MgO	Na_2O	K_2O	P_2O_5	MnO
61.7	14.4	0.57	3.8	1.3	1.3	1.3	2.1	0.14	0.03

The results show that the average gold content in the GES tailings, i.e. the initial product, is 2.57g/t and the silver content is 0.63g/t (Table 1). In the first pilot study, a flotation enrichment process was carried out with the initial product size being 65% -0.074+0 mm.

In the second experimental study, a flotation enrichment process was carried out using a single-stage crushing process on the initial product (80% -0.074+0 mm class).

In the third experimental study, a crushing process was carried out, and a flotation process was carried out with the addition of soda-1000g/t and I-20 -60g/t flotation reagents during the crushing process.

In the fourth experimental study, a crushing process was carried out, and a flotation process was carried out with the addition of soda-1000g/t and I-20 -60g/t during the crushing process, and CuSO4 was added to the control flotation.

The experiments on flotation enrichment were carried out in an open cycle under the following reagent regime (experiment 1):

- I-main flotation – 10 minutes; $CuSO_4$ – 100 g/t, Kst – 140 g/t, T-92 – 80 g/t;
- II-main flotation – 20 minutes; Kst – 70 g/t, T-92 – 30 g/t;
- Control flotation – 15 minutes; Kst – 30 g/t, T-92 – 10 g/t;

The results of all experiments are presented in Table 3.

Table 3. Results of flotation enrichment.

Product name	Product output, %	Composition, %				Extraction rate, %			
		Au, g/t	Ag, g/t	S_s	C_{org}	Au	Ag	S_s	C_{org}
Experiment 1. Initial product size 65% -0.074+0mm class									
Enriched product 1	3.74	9.6	1.9	1.1	0.9	13.86	13.5	7.91	14.23
Enriched product 2	5.91	8.5	1.2	1.6	0.6	19.38	13.46	18.16	14.98
Enriched product 3	4.63	4.2	0.9	0.9	0.47	7.50	7.91	8.00	9.19
Flotation process enriched product combined	14.27	7.39	1.29	1.24	0.64	40.75	34.87	34.08	38.4
Waste	85.73	1.79	0.4	0.4	0.17	59.25	65.13	65.92	61.6
Ore	100.00	2.59	0.53	0.52	0.24	100.0	100.0	100.0	100.0
Experiment 2. The size of the crushed product is 80% in the -0.074+0mm class									
Enriched product 1	4.5	9.5	2.8	1.4	0.92	16.42	22.75	10.61	16.74
Enriched product 2	9.1	6.1	0.8	1.3	0.62	21.32	13.15	19.93	22.81
Enriched product 3	4.7	4.6	0.6	0.95	0.4	8.3	5.09	7.52	7.6
Flotation process enriched product combined	18.3	6.55	1.24	1.23	0.64	46.04	40.99	38.06	47.15
Waste	81.7	1.72	0.4	0.45	0.16	53.96	59.01	61.94	52.85
Ore	100.0	2.6	0.55	0.59	0.25	100.0	100.0	100.0	100.0
Experiment 3. Soda ash - 1000g/t, I-20 -60g/t were added to the crushing process									
Enriched product 1	4.52	11.1	1.9	1.7	0.86	19.06	16.06	16.04	16.2
Enriched product 2	8.94	6.2	1.0	1.3	0.65	21.07	16.73	24.28	24.23
Enriched product 3	4.33	5.6	0.7	0.9	0.45	9.22	5.67	8.14	8.12
Flotation process enriched product combined	17.78	7.3	1.16	1.3	0.65	49.35	38.45	48.46	48.55
Waste	82.22	1.62	0.4	0.3	0.15	50.65	61.55	51.54	51.45
Ore	100.0	2.63	0.53	0.48	0.24	100.0	100.0	100.0	100.0
Experiment 4. Soda ash-1000g/t, I-20-60g/t were added to the crushing process and CuSO4 was added to the control flotation									
Enriched product 1	9.6	1.8	1.9	0.81	15.9	14.53	16.61	15.09	9.6
Enriched product 2	6.0	1.0	1.5	0.57	21.4	17.45	28.37	22.97	6.0
Enriched product 3	5.4	0.7	0.5	0.41	9.96	6.31	4.88	8.53	5.4

Flotation process enriched product combined	18.29	6.68	1.11	1.33	0.58	47.26	38.29	49.86	46.58
Waste	81.71	1.67	0.4	0.3	0.15	52.74	61.71	50.14	53.42
Ore	100.0	2.59	0.53	0.49	0.23	100.0	100.0	100.0	100.0

The technological scheme of the research work carried out is shown in Figure 1.

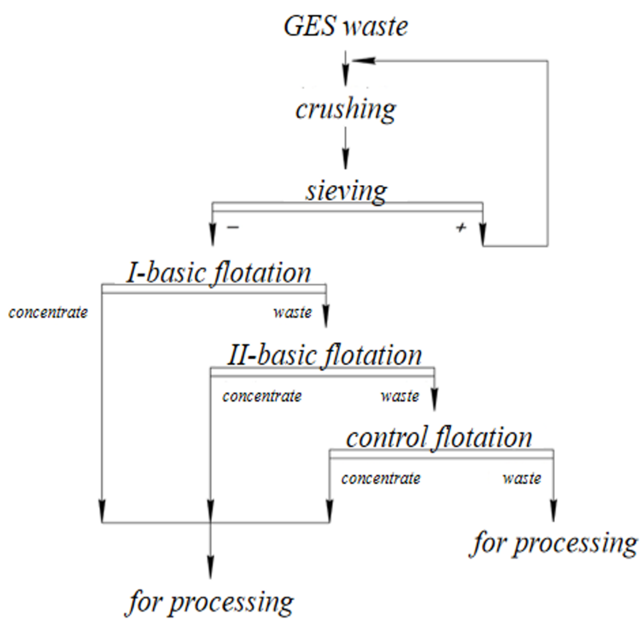

Fig. 1. Technological scheme of the flotation enrichment process.

As can be seen from Table 3, in the first study, when we enriched the initial size product, the gold recovery rate was 40.75%, and for silver, it was 34.87%. The gold content in the flotation waste was 1.79 g/t, and for silver, it was 0.4 g/t.

In the second study, the gold recovery rate increased by 46.04%, and for silver, it was 40.99%. The gold content in the flotation waste decreased by 1.72 g/t, and for silver, it was 0.4 g/t.

In the third study, when the crushed sample was flotated by adding soda and I-20 reagents to the crushing process, the gold recovery rate increased by 49.35%, and for silver, this indicator was 38.29%. The amount of gold in the waste was 1.62 g/t.

In the fourth study, the flotation process was carried out with the addition of soda ash-1000g/t, I-20 -60g/t during the crushing of the initial product, and flotation was carried out with the addition of $CuSO_4$ reagent to the control flotation. In this study, the recovery rate for gold was 47.26%, and for silver it was 38.29%. The amount of gold in the waste was 1.67 g/t, and the amount of silver was 0.4 g/t.

Table 4 shows the results of the gravity enrichment experiment using the jigging-table scheme. The recovery into the table concentrate was 3.51%, with the quality of the concentrator being 4.20 g/t, and into the gravity concentrate tailings 2.15%. The gold content in the gravity enrichment tailings was 2.54 g/t.

Table 4. Results of tailings enrichment using the jigging table scheme.

Product name	Product yield, %	Content	Extraction
		Au, g/t	Au, %
With initial grinding of 65% cl.-0.074+0mm			
Concentration table concentrate	2.15	4.20	3.51
Concentration table waste	51.31	2.48	49.46
Sedimentation oncentrate	53.46	2.55	52.97
Sedimentation waste	46.54	2.6	47.03
Waste of gravity	97.85	2.54	96.49
Total initial sample	100.00	2.57	100.00

For comparison, experiments were conducted on a Knelson centrifugal concentrator. The experiments were carried out with staged and direct enrichment of ore (1st experiment – one concentration with initial grinding, 2nd experiment – one concentration with grinding of 80% of the -0.074+0 mm class, and 3rd experiment staged enrichment with threefold concentration – 1st at 65% and 2nd at 80% of the finished class; 3rd control). The results of the experiments are presented in Table 5.

Table 5. Results of enrichment experiments on the Knelson centrifugal concentrator.

Product name	Product yield, %	Content, %				Extraction, %			
		Au	Ag	S_s	C_{org}	Au	Ag	S_s	C_{org}
1-experiment 65% grinding class -0.074+0mm									
Graviconsentrat	0.56	7.7	1.5	6.8	0.32	1.64	1.39	7.8	1.8
Waste	99.44	2.6	0.6	0.45	0.1	98.36	98.61	92.2	98.2
Ore	100.0	2.63	0.61	0.49	0.1	100.0	100.0	100.0	100.0
2-experiment 80% grinding class -0.074+0mm									
Graviconsentrat	0.73	6.8	1.0	4.6	0.17	1.89	1.32	7.0	1.2
Waste	99.27	2.6	0.6	0.45	0.1	98.11	98.68	93.0	98.8
Ore	100.0	2.63	0.6	0.48	0.1	100.0	100.0	100.0	100.0
3-stage concentration experiment									
Concentrate 1	0.56	5.2	1.2	2.8	0.17	1.11	1.23	3.32	0.94
Concentrate 2	0.52	5.3	7.1	4.0	0.19	1.05	6.75	4.41	0.98
Concentrate 3	0.57	5.5	2.1	3.9	0.17	1.19	2.19	4.71	0.96
Combined concentrate	1.65	5.34	3.37	3.56	0.18	3.35	10.16	12.44	2.87
Waste	98.35	2.58	0.5	0.42	0.1	96.65	89.84	87.56	97.13
Ore	100.0	2.63	0.55	0.47	0.1	100.0	100.0	100.0	100.0

From the obtained data, it can be seen that gold extraction is low, the content in gravity tails is practically at the initial level.

4 Conclusion

The obtained results on gravity enrichment show that gravity enrichment for this sample of the old tailing dump gold extraction workshop is not advisable. As noted above, the best results for gold extraction were obtained by sorption cyanidation of finely ground material (90% class -0.020+0 mm).

The results obtained on gravitational enrichment show that gravitational enrichment is not appropriate for this model of gold extraction from industrial waste, since the gold in the industrial waste is distributed in the form of very small particles, so enrichment by the gravitational method is ineffective.

In flotation enrichment, the best result was obtained when the sample was finely ground to 80% grade -0.074+0 mm and dosed with soda and I-20 for grinding. At a flotation time of 45 minutes, gold extraction was 49.35%, and the metal content in the tailings was 1.62 g/t.

As can be seen from the results obtained, the best result was observed in the third research study, when flotation was carried out by adding soda and I-20 reagents to the crushing process, the gold recovery rate increased significantly, to 49.35%. This indicator also has an effect on the subsequent processing of the obtained concentrate, that is, it serves as the basis for a higher overall level of recovery of the valuable component from the GES waste.

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