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TECHNOLOGY FOR PROCESSING LOW-SULFIDE GOLD-QUARTZ ORE

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The relevance *of the research is caused by the necessity to acquire new knowledge about the enrichment of gold-bearing ores and the need to develop the resource base of the Russian Federation, namely involve subsoil use objects in economic circulation.*
The purpose *of the study is to study the processing of low-sulfide gold-quartz ore using an integrated technology for the enrichment of mineral raw materials.*
Object: *gold ore raw materials with an average gold grade of 11,88 g/t. The silver content is negligible – 2,43 g/t. The main ore minerals in the sample are pyrite and pyrrhotite. The average content of these minerals in the ore, according to mineralogical and X-ray diffraction analysis, was about 6 % (in total). The main rock-forming minerals of the original ore are: quartz – 60,1 %; quartz-chlorite-mica aggregates – 3,8 %; carbonates – 7,1 %.*
Methods. *The research methodology was based on the theoretical foundations of mineral processing. The study of the material composition of the ore, and enrichment products, was carried out using chemical, assay, thermal, spectrometric methods, as well as atomic emission spectrometry and other methods. The study of washability was carried out by the methods of gravity and flotation enrichment, namely: GRG test, stage test of the Institute TOMS (Russia) (determination of the optimal size of ore grinding and the number of enrichment stages), modeling of gold extraction in the grinding cycle, determination of the optimal degree of grinding, study of the choice of flotation reagents, study of the kinetics of the process and the implementation of a complex of technological studies.*
Results. *It was found that the gold recovery when performing the GRG test was 72,75 % with a total concentrate yield of 1,34 % and a content of Au 664,78 g/t. At the same time, the gold content in the tailings was 3,29 g/t. A stage test showed that for ore processing only by gravity technology, it is advisable to use a two-stage scheme. The first stage is in the grinding cycle with the ore size of 60–70 % and the second stage – with the final size of the discharge of the classification 90 % size of –0,071 mm. Centrifugal separation, as a free gold recovery operation in a grinding cycle, works efficiently. A concentrate with a gold content of 2426 g/t was obtained with an output of 0,31 % and an extraction of 63,74 %. The enrichment of the first stage tailings crushed to 90 % size of –0,071 mm at the KC-CVD concentrator (modeling) made it possible to extract gold into a total gravity concentrate (KC-MD+KC-CVD) of 87,25 % with a concentrate yield of 22,63 %. The gold content in the tailings was 1,97 g/t. The results of gravity and flotation concentration of the original ore indicate the feasibility of using a combined gravity-flotation technological scheme. In a closed experiment of the initial ore beneficiation according to the gravity-flotation scheme at a natural pH of the pulp (without adding acid), the following products were obtained: gravity concentrate with a gold content of 2426 g/t at a yield of 0,31 % and an extraction of 64,06 %; flotation concentrate (after the II cleaning) with a gold content of 122 g/t with a yield of 2,90 % and recovery of 33,01 %; the total gold recovery in the gravity-flotation concentrate was 94,07 % with a yield of 3,21 % and Au content of 345,87 g/t, the gold content in the flotation tailings was 0,72 g/t.*

Key words:
Gold, enrichment, technological research, flotation, gravity, modeling, concentrate, tailings, extraction.

Introduction

Research and technology of the mining and processing industry has developed continuously as the amount of knowledge and experience expanded, and the analytical and instrumental base improved. Work in this area has been carried out since mining has existed. Despite the colossal number of research projects carried out both on the territory of the Russian Federation [1–4] and in foreign research centers in China [5], Korea [6], Brazil [7], the topic of the maximum possible extraction of valuable components from refractory ores remains a topical direction of development for the entire mining and processing and metallurgical complexes. Based on the analysis of the literature [8–11], the most demanded direction is the processing of gold-bearing ores with different content of substances that prevent the complete extraction of the valuable component.

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To achieve these goals and objectives, as a rule, various complex processing schemes are used: gravity-flotation, gravity-metallurgical, etc. The use of one or another scheme for the enrichment of mineral raw materials depends on many different factors, such as the shape of particles, geometric dimensions, weight, wettability, association with minerals, etc. For example, in [12], the effect of particle shape and roughness index on the flotation is shown. In [13], studies on the influence of the particle size of raw materials on the efficiency of enrichment in industrial conditions are presented. Due to its chemical inertness, almost all the gold in the earth's crust is in a native state, namely in the form of a metal with an insignificant content of impurities. In view of this fact and the physical properties of this metal, it becomes possible to use gravitational methods for separating minerals. However, the use of this technology, in the presence of finely dispersed raw materials, is often accompanied by high losses of a valuable component and a negative impact on environmental objects.

In this case, a promising direction is the use of an integrated processing scheme, with the use of highly efficient enrichment processes. For example, flotation is the most efficient and more widespread method of beneficiation of gold-bearing ores when sulfide minerals are present in it, even in small amounts [14]. Gravity methods will be effective for the extraction of larger particles of valuable minerals [15]. Gravity-flotation technologies for processing gold-bearing ores are a fairly common and effective way of enriching mineral raw materials [16–19].

This article is devoted to the processing of low-sulfide gold-quartz ore using an integrated technology for the enrichment of mineral raw materials.

The main objectives of the study are:

- analysis of qualitative and quantitative characteristics of processed raw materials;
- study of ore concentration using gravity methods;
- study of ore dressing using the flotation method;
- recommendations on the use of the proposed technology on analogous types of ore.

Object of research

The study was carried out using a sample that is characterized by a low-sulphide gold-quartz type of ore. The average gold content in the technological sample according to the sampling certificate is 8,5 g/t. The granulometric composition of the ore with a particle size of –2 +0 mm, prepared for tests, is presented in the Table.

Table. *Granulometric characteristics of crushed ore (size of –2 mm) with distribution of gold, silver, iron and sulfur by size classes*

Таблица. *Гранулометрические характеристики дробленой руды (класс крупности –2 мм) с распределением золота, серебра, железа и серы по классам крупности*

Product size class Класс продукта	Product yield, % Выход продукта %	Au		Fe		S	
		g/t (г/т)	%				
		content содержание	distribution распределение	content содержание	distribution распределение	content содержание	distribution распределение
–2+1	37,29	9,60	33,21	5,73	33,22	2,63	32,68
–1+0,5	12,97	17,50	21,24	5,66	11,24	2,77	12,35
–0,5+0,315	6,44	13,20	7,44	5,73	5,61	2,67	5,72
–0,315+0,2	7,21	9,40	5,81	5,77	7,18	2,65	6,05
–0,2+0,1	11,27	8,60	8,86	6,57	11,12	0,34	12,35
–0,1+0,071	6,24	9,50	6,01	7,83	6,94	4,14	7,99
–0,071+0,045	4,77	12,20	5,12	8,67	6,18	4,32	6,52
–0,045+0	13,81	11,20	12,81	10,07	18,51	4,04	16,17
Total/Итого	100,00	11,11	100,50	6,62	100,00	3,08	100,00

According to the sieve analysis of the ore, gold distribution by size classes is uneven. Its content in size classes ranges from 8,6 g/t (–0,2 + 0,1 mm) to 17,50 g/t (–1 + 0,5 mm). This distribution pattern is an indication of the presence of large and medium-sized gold particles in the crushed ore in free form and in rich intergrowths.

The results of optical emission analysis of the ore (analyzes ICP90, ICP40) showed that the content of harmful elements – arsenic and antimony, does not exceed 0,005 %, and they will not have a negative effect on ore processing. The SiO_2 content is 78,7 %. The content of heavy non-ferrous metals (Cu, Pb, Zn) is thousandths of a percent, and they will not have an industrial value for extraction. The only valuable component of the ore is gold.

The rocks of the deposit are represented by mineralized carbonate-quartz rocks with different-grained granoblastic quartz, ferruginous calcite or ankerite in the interstices of quartz grains, grouping into spotty, vein-like accumulations. Isometric siderite grains are present among carbonate minerals, the decomposition of which produces irregular magnetite grains. Among other nonmetallic minerals, thin sections contain flakes of chlorite, muscovite, and epidote aggregates [20].

The main rock-forming minerals of the original ore are: quartz – 60,1 %; quartz-chlorite-mica aggregates – 3,8 %; carbonates – 7,1 %. Slimes are represented by carbonated mica-quartz, with chlorite, weight – 22,8 %. Ore minerals (sulfides) are represented by pyrite – 2,2 %; arsenopyrite – signs; chalcopyrite and galena are rare signs. In terms of sulfide content, the ore of the deposit is classified as poor, in terms of the degree of oxidation to sulfide.

Phase analysis of the original ore showed that with a size of –2 mm, the share of free gold is 29,11 %. With an increase in the fineness of grinding to 60 % and further to 95 % size of –0,071 mm, the amount of free gold is 50,50 and 63,16 %, respectively. The share of cyaninated gold (free and in intergrowths) with a size of –2 mm is 57,59 %, with a size of 60 and 95 % size of –0,071 mm, respectively, 77,35 and 93,29 %.

It can be concluded that this ore is a favorable raw material for beneficiation by gravity and flotation methods. It is also possible to efficiently process the original ore and products of its concentration by hydrometallurgical methods [21].

Research methodology

Research technique for washability by gravity methods

The results of studying the material composition of the ore showed that when grinding to a size of 95 % size of –0,071 mm, more than half of the gold (63,16 %) is present in free form. Therefore, for processing this ore, it is advisable to use gravity concentration methods and, in particular, the following processes:

- centrifugal separation with low concentrate yield (up to 1,5 %);
- centrifugal separation with increased concentrate yield (more than 1,5 %).

In principle, the possibility of using centrifugal methods for extracting free gold from the ore of the deposit was established according to the results of a special international GRG test (the method of Knelson, Canada).

The final grinding size and the feasibility of dressing the intermediate size ore was determined according to the results of a stage dressing test on a centrifugal concentrator (TOMS method).

Evaluation of the level of free gold recovery at an intermediate ore size (60 % size of –0,071 mm) by gravity methods was determined according to the results of enlarged concentration tests on a jig and on a separator with periodic unloading.

The influence of the concentrate yield on gold recovery was determined by the results of modeling ore concentration on a separator with continuous concentrate discharge.

In total, at the stage of research on enrichment of ore of the deposit by gravitational methods, the following tests were performed:

- *GRG test (Knelson methodology, Canada)*

The purpose of the GRG test (Knelson, Canada) is to determine the amount of gold recovered by gravity and to evaluate the possibility of using centrifugal separators with a low concentrate yield for enriching the ore of the deposit.

The method of performing the GRG test provides for a three-stage ore beneficiation. The feed size is: at the first stage –2 (1,7) mm, at the second – approximately 80 % size of –250 microns and at the third – 80 % size of –75 microns [22–24] (Fig. 1).

- *TOMS stage test (determination of the optimal size of ore grinding and the number of beneficiation stages)*

The purpose of the experiment was to determine the optimal (final) size of ore grinding and the number of stages of gravity concentration. The determination of the optimal size of grinding was carried out according to the results of the stage-by-stage ore gravity concentration. The experiment was carried out on a centrifugal concentrator with a sequential reduction in the size of the ore at each stage. The material was investigated with a size of –2 mm in the range from 16,99 to 93,96 % and –0,071 mm. At each stage, four enrichment operations were performed (at the first stage, two operations). Based on the data obtained, a graph of gold recovery dependence on ore size was built (Fig. 2). Additionally, according to the calculation of the Hancock criterion, the enrichment efficiency curve was plotted on the graph. The full results of the staged test are shown in Fig. 3.

- *simulation of gold recovery in the grinding cycle (first stage of beneficiation)*

The purpose of this test was to assess the possibility of extracting free gold at an intermediate ore size (~60 % –0,071 mm) by gravity methods with a low concentrate yield. During the test, a single dressing was carried out on a centrifugal concentrator (KC-MD) of a sample of the original ore with a particle size of 60 % size of –0,071 mm. To assess the possibility of separating free gold into the «gold head», metallurgical refinement of the concentrate was performed.

- *modeling of the second stage of enrichment at KC-CVD with refinement of the concentrate*

The purpose of this test is a preliminary assessment of the applicability of the KC-CVD concentrator for the recovery of gold from the tailings of the first stage of concentration and to determine the effect of the yield of the final gravity concentrate on the concentration indicators. The preliminary assessment was carried out by modeling the process using the following equipment:

- concentration table (CT);
- centrifugal concentrator with periodic concentrate discharge (KC-MD).

The CT concentrate in combination with the KC-MD one characterizes the product of the KC-CVD concentrate refinement when obtaining the final gravity concentrate. This approach allows a preliminary assessment of the possibility of using CVD for ore beneficiation. If necessary, CVD performance indicators are specified on enlarged tests. The test was performed on stage 1 tailings from the KC-MD. Enrichment was carried out in several sequential stages on a centrifugal concentrator (KS MD3) and a concentration table (CT). Concentrate KS MD3 and CT of each stage were combined. On the tails, the enrichment operation was repeated at KS MD3 and RMS. A total of six stages have been completed. As a result, six concentrates and tailings were obtained. The resulting dependence of the efficiency of gold recovery on the yield of KC-CVD concentrate is shown in Fig. 3. In some cases, it is possible to develop statistical regression models that can be used to optimize the process [25].

Experimental procedure for washability by flotation methods

The study of the ore dressability of the technological sample was carried out on the original ore and on the tailings of gravity concentration. The purpose of the research was to work out the optimal flotation mode for obtaining sulfide gold-bearing concentrate.

The main objectives of the research were:

- study of granulometric characteristics;
- develop a reagent regime;
- study physical characteristics of the pulp;
- determine flotation time by operations and structure of the flotation circuit.

Determination of the optimal size of the initial ore grinding

To determine the optimal degree of grinding of the initial ore for flotation, a series of tests was carried out on material with a particle size of (70–95) % – 0,071 mm. Pine oil was used as a foaming agent at the initial stage of research. Medium regulators were not supplied. The flotation was carried out at a natural level of pH=8,04, formed due to the dissolution of ore minerals. Basic and control flotation operations were performed.

Determination of the optimum size for grinding gravity tailings

To determine the optimal degree of grinding of gravity tailings for flotation, a series of tests was carried out on material with a particle size of 70, 80, 85, 90 and 95 % – 0,071 mm. Pine oil (CM) was used as a foaming agent. Medium regulators were not supplied. Flotation was carried out at natural pH=7,64. The main and control flotation operations were carried out. To evaluate the results of flotation in each experiment, the value of the Hancock criterion was calculated, which characterizes the efficiency of the flotation, and a graph of this indicator dependence on grinding size of gravity tailings was plotted, shown in Fig. 4.

Flotation kinetics study

In order to select the type and consumption of the frother for flotation, three tests were delivered on material with a particle size of 90 % – 0,071 mm. Reagents pine oil, T-66 and T-80 (similar in properties to T-92), were used as foaming agents. Collector consumption and flotation time were the same for all experiments. The consumption of the foaming agent was selected empirically according to the type and loading of the foam layer. The baseline was the experience with pine oil. A similar approach to the experiment is presented in the works [26, 27].

Open experiments on flotation of gravity tailings at different pulp densities

In order to establish the flotation indicators at the optimal mode and to check the effect of the pulp density on them, three open experiments were performed on gravity tails, in which the solids content in the pulp was equal to 20, 27 and 35 % by weight. In each experiment, the main and control operations were performed. The reagent mode and flotation time were adopted based on the results of the kinetic tests. Tests have shown that the optimum solids content in the slurry is in the range of 27–30 %. An increase in the solids content from 27 to 35 % leads to an increase in the yield of the concentrate (from 11,90 to 12,92 %) with almost unchanged gold recovery in it. This leads to a decrease in the gold content in the concentrate from 31,78 to 29,31 g/t.

Open experience of flotation of gravity separation tailings according to the complete scheme

An open experiment on flotation of gravity separation tailings was set up at the optimal modes established at the prospecting stage of research in order to clarify the concentration indicators. The flotation was carried out at the natural pH of the pulp, which is formed due to the dissolution of ore minerals. The flotation scheme included the main, control and two cleaning operations.

Closed experiment on gravity-flotation concentration of the original ore at the natural pH value of the pulp

To establish the indicators of the initial ore beneficiation according to the gravity-flotation scheme, a test was put in which flotation was carried out with the return of middlings (five closed cycles) according to the regime worked out at the preliminary stage of research.

The scheme of the experiment is shown in Fig. 5. The original ore was crushed to a particle size of 60 % – 0.071 mm and enriched by centrifugal separation in order to extract free gold and rich intergrowths. The tails of centrifugal separation were re-crushed to a particle size of 90 % – 0,071 mm and fed to flotation, in which the main, control and two cleaning operations were provided. Flotation was carried out in a closed cycle with a turnover of industrial products using a circulating water supply according to a «short» scheme (through a tailings thickener).

To assess the technological efficiency of enrichment used in the work, the Luiken–Hencock formula was used:

$$E = \frac{\gamma(\beta-\alpha)}{\alpha(1-\alpha)},$$

where α is the content of the valuable component in the source material; β is the content of the valuable component in the fortified product; γ is the yield of the enriched product.

The process is considered highly efficient if E>75 %, effective if E>50 %, and ineffective if E<25 %.

Results and its discussion

Study of the deposit ore for dressability by gravity methods Evaluation of the possibility of using centrifugal methods (GRG test)

The total volume of gold extracted by gravity after three stages of enrichment was 72,75 % with a concentrate yield of 1,34 %. Including: at the first stage, the extraction by gravity was 24,80 %, at the second and third stages, respectively, 39,89 and 8,07 %. The estimated gold grade in the original ore for the GRG test was 11,91 g/t.

The results of the GRG test showed that the ore of the deposit is efficiently concentrated by centrifugal methods. The level of gold extraction by gravity during stage grinding of ore is high – 72,75 %. But for more complete gold recovery, the use of centrifugal concentrators in combined schemes (gravity-flotation, etc.) can be recommended [28, 29].

Staged centrifugal ore beneficiation

On the original ore size of –2 mm (16,99 % –0,071 mm), the extraction of gold was 36,37 %. When grinding to 49,10 % size of –0,071 mm, the extraction increased by 31,82 % and totaled 68,19 %. When grinding to 66,33 % size of –0,071 mm, the total gold recovery was 76,77 %. A further reduction in the size of the ore to 89,47 and 93,96 % made it possible to additionally extract 4,21 and 2,01 % of gold, respectively. The total gold recovery in the total concentrate is 87,44 % at a content of 71,56 g/t and yield of 13,70 %. Under these conditions, the gold content in the tailings was 1,63 g/t. The gold grade in the original ore was 11,21 g/t. The dependence of the gold recovery efficiency on the recovery size is shown in Fig. 2.

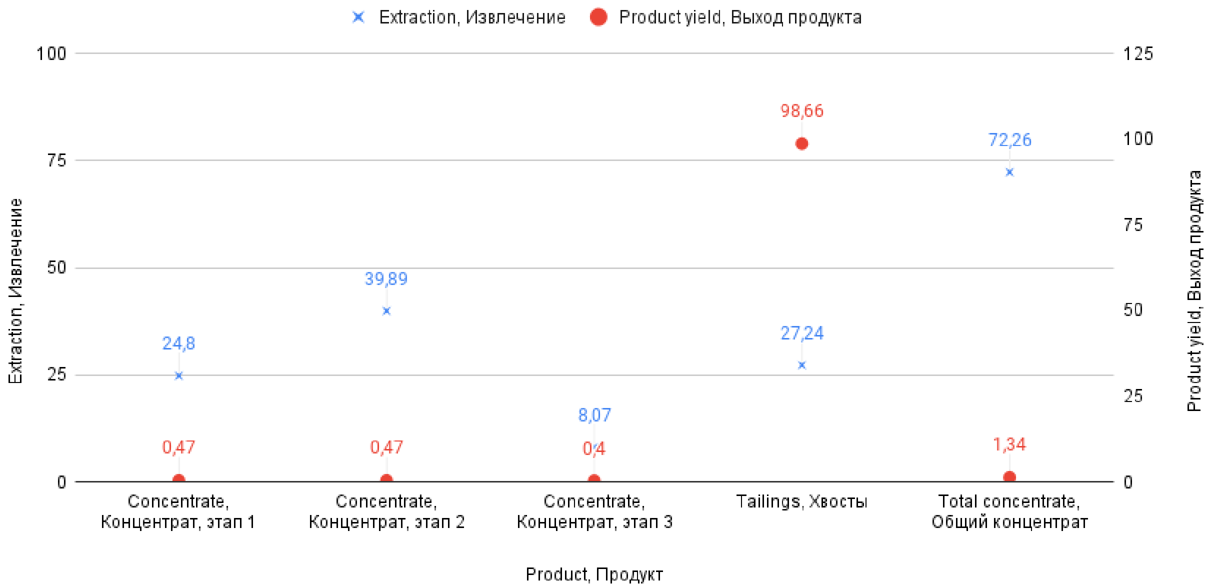

Fig. 1. *Results of the GRG test (gold recovery by stages)*
Рис. 1. *Результаты теста GRG (извлечение золота по стадиям)*

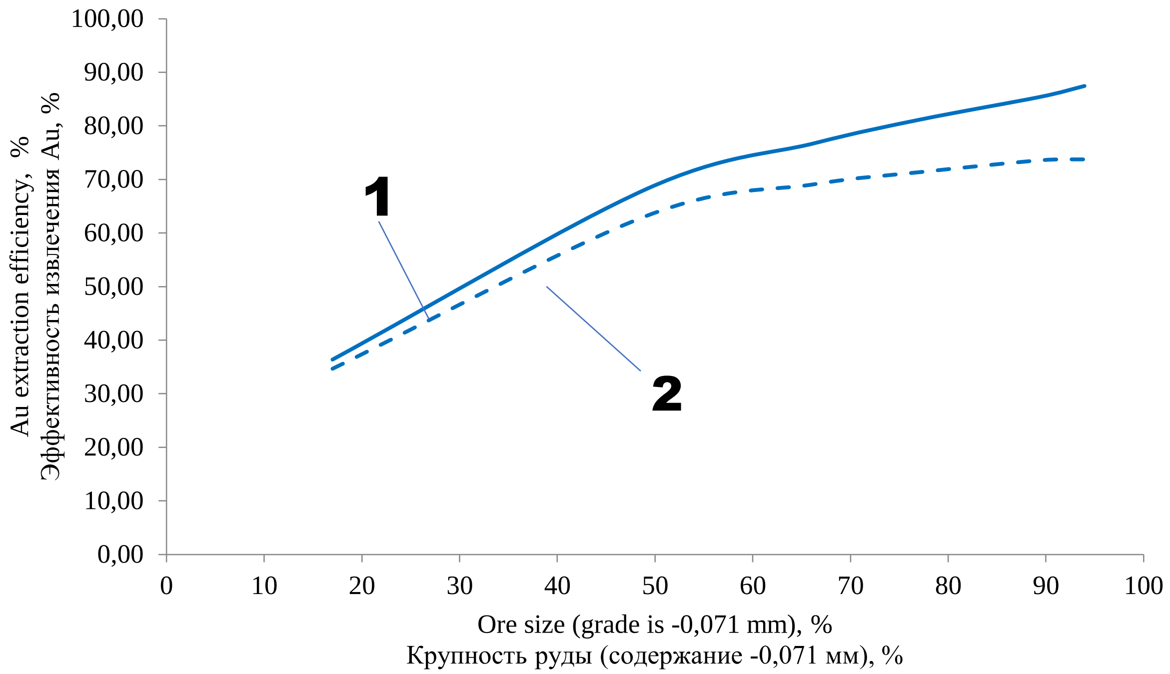

Fig. 2. *Dependence of gold extraction efficiency on ore grinding size: 1 – Au recovery, %; 2 – efficiency, %*
Рис. 2. *Зависимость эффективности извлечения золота от крупности измельчения руды: 1 – извлечение золота, %; 2 – КПД, %*

High gold recovery (76–81 %) at an intermediate size (66–77) % –0,071 mm indicates the advisability of using gravitational ore concentration in the grinding cycle. With a particle size of 90 % size of –0,071 mm, the rate of growth of the efficiency curve decreases (inflection point), and this coarseness can be taken as the final one for gravity concentration. It is recommended to use a two-stage enrichment scheme. The first stage in the grinding cycle with an intermediate ore size of 60–70 % size of –0,071 mm, the second stage at the drain of the classification unit with a final ore size of 90 % – 0,071 mm (for gravity concentration).

Simulation of gold recovery in a grinding cycle (first stage of beneficiation)

The results of the experiment showed that centrifugal separation in the grinding cycle at an intermediate size of 60 % –0,071 mm works effectively. A concentrate with a gold content of 2426,0 g/t was obtained with an output of 0,31 % and an extraction of 63,74 %. Therefore, this process can be recommended for use in an industrial technological scheme. Based on the results of metallurgical refinement of the KC-MD concentrate, the approximate level of gold extraction into a rich concentrate – «gold head», amounted to 59,60 % of the original ore [30].

Simulation of the second stage of beneficiation at KC-CVD with concentrate finishing

In the course of preliminary tests, it was found that the ore of the deposit is efficiently enriched by centrifugal separation with a low concentrate yield. But at the same time, the extraction of gold is not complete enough. One of the possible ways to increase gold recovery by gravity methods, including those associated with gangue minerals and sulfides, is the use of a centrifugal concentrator KC-CVD with continuous concentrate discharge at the second stage of enrichment.

Based on the test results, a graph (Fig. 3) of gold recovery dependence on the concentrate yield of the first and the second stages was built. Additionally, the enrichment efficiency curve is plotted on the graph, built according to the calculation of the Hancock criterion.

The graph shows that a significant amount of gold (65–70 %) is extracted with a concentrate yield of 0,5–1 %. Further, with an increase in the yield of the concentrate to 3,17 %, the growth of the efficiency curve slows down. At this stage, gold is extracted in intergrowths with rock and sulfides.

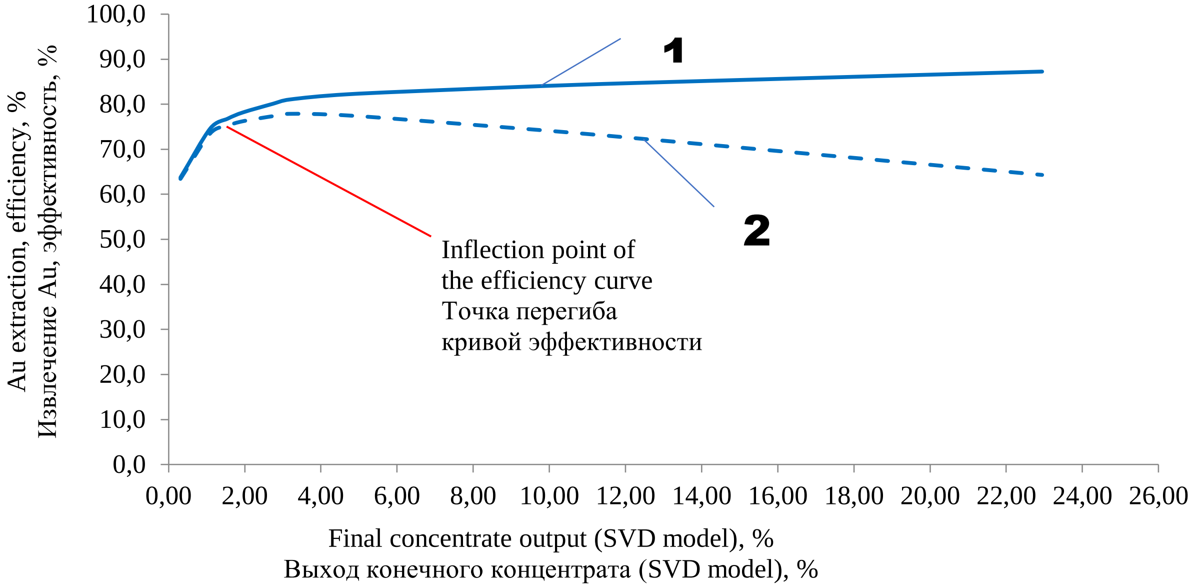

Fig. 3. *Dependence of gold recovery efficiency on the yield of KC-CVD concentrate: 1 – Au recovery, %; 2 – efficiency, %*

Рис. 3. *Зависимость эффективности извлечения золота от выхода концентрата KC-CVD: 1 – извлечение золота, %; 2 – эффективность, %*

With an increase in the concentrate yield over 3,17 %, the enrichment efficiency begins to decline. At this stage, gold recovery occurs with significant dilution of the concentrate by waste rock. The inflection point of the efficiency curve is the point that determines the minimum required concentrate yield. In this case, the yield of the total gravity concentrate must be at least 3,17 %, including the yield of CVD concentrate after finishing is 2,86 %. At a given yield of 3,17 % of the total concentrate (KC-MD and KC-CVD), the gold content in it will be 303,71 g/t with 81,04 % recovery, and the gold content in the tailings will be 2,33 g/t.

The test results showed that, with a high yield of KC-CVD concentrate (maximum – 22,63 %), gold recovery into the total gravity concentrate is high and amounts to 87,25 %. However, the content in the tailings remains significant – 1,97 g/t. After finishing the concentrate, the gold recovery decreases to 81,04 %, and, accordingly, the content in the tailings increases to 2,33 g/t. Consequently, the use of the KC-CVD concentrator at the second stage of concentration does not ensure the completeness of gold recovery. The potential for CVD can be determined after comparison with other alternative enrichment methods. In this connection, further studies on ore dressing by flotation methods have been initiated.

Investigation of the deposit ore dressability by flotation methods

Based on the test results, the following conclusions were made:

- the results of the balance calculation of the gold content in the original ore for each experiment showed a significant scatter in the value of this parameter (within 9,63–11,79 g/t). At the same time, gold extraction into the flotation concentrate of the main operation randomly fluctuated in the range of 64,14–70,95 % without any connection with the size of ore grinding, which did not allow establishing the optimal value of this parameter;
- the overall level of gold recovery into the flotation concentrate in the experiments remained at a relatively low level (64,14–70,95) %, which indicated an insufficiently high efficiency of flotation on the initial ore. This circumstance, most likely, is again explained by the presence of large free gold in the ore, which, due to the large mass of particles, cannot go into the foam product and remains in the chamber flotation product. Obviously, for the ore of the technological sample of the deposit, a combined enrichment scheme, including gravity processes for capturing free gold, will be more promising, and flotation to recover fine

free gold and gold associated with sulfide minerals. In relation to the above problems, it was decided to stop prospecting experiments on the original ore and continue research on flotation on gravity tailings [31].

Determination of the optimal size for grinding gravity tailings

The graph of the concentration efficiency dependence on the grinding size of gravity tailings before flotation is shown in Fig. 4.

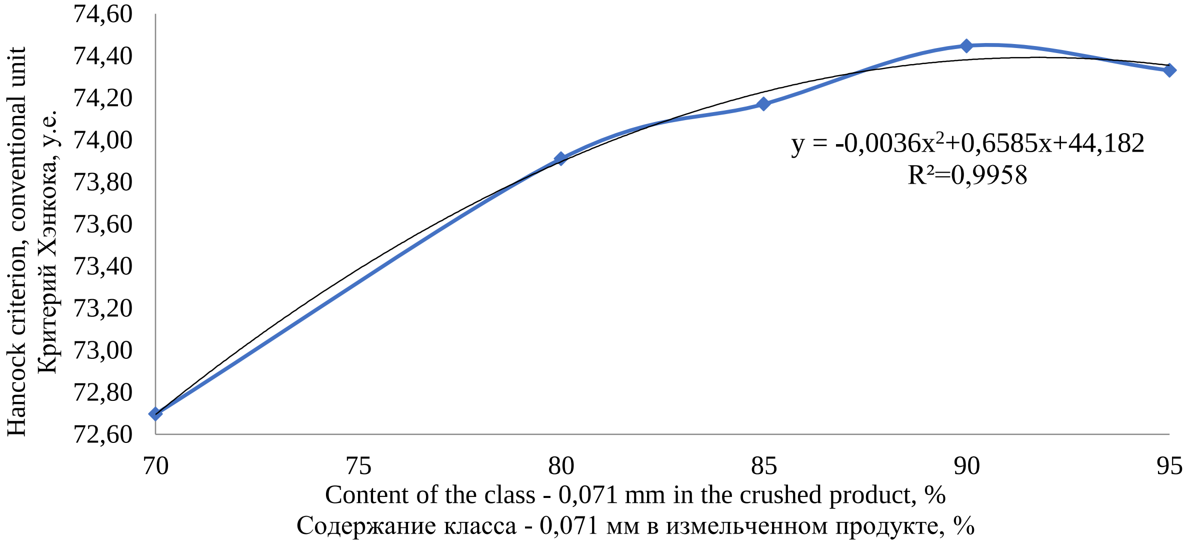

Fig. 4. *Dependence of the beneficiation efficiency on the size of crushing of gravity tailings before flotation*
Рис. 4. *Зависимость эффективности обогащения от крупности дробления хвостов гравитации перед флотацией*

It was found that the enrichment efficiency indicator increases to the size of ore grinding of 90 % size of –0,071 mm and then begins to decrease. Consequently, when the size of grinding of gravity tailings is greater than 90 % size of –0,071 mm, the fineness of ore grinding no longer contributes to the improvement of the concentration indicators. The recovery of gold into concentrate remains at approximately the same level. The gold content in tailings for a grinding size of 90 and 95 % size of –0,071 mm is practically the same (0,71 and 0,70 g/t). Therefore, the size of crushing of gravity tailings equal to 90 % size of –0,071 mm was taken as optimal for flotation.

Investigation of the kinetics of flotation

Based on the results of the tests on the study of the kinetics of flotation, the following conclusions were made:

1. Pine oil has the lowest flow rate for flotation. Normal foaming is ensured at a flow rate of 60 g/t (main + control operation). Reagents T-66 and T-80 require a flow rate of 100 g/t for the normal operation of the process.
2. When using oxal T-66, the concentrate yield decreases from 11,1 to 9,30 %, and with the concentration efficiency equal to that of pine oil, the extraction of gold into the flotation concentrate decreases.
3. The use of oxal T-80, which is close in foaming properties to T-92, allows one to obtain enrichment results as in pine oil and even slightly exceed them. Therefore, further experiments were performed using this reagent.

To determine the optimal flotation time for operations, two groups of tests were set to study the kinetics of the main, control and cleaning operations. In all tests, a fractional survey of concentrates was carried out, and the gold content was determined in each fraction. The tests were carried out on the optimal reagent mode with the ore grinding size 90 % size of –0,071 mm: butyl xanthate (PBX)=150+75 g/t, T-80=60+40 g/t, the pH value of the pulp is equal to the natural one (7,64).

Analysis of the results obtained made it possible to determine the optimal flotation time for operations, taking into account the following features:

1. The optimal time for the main operation is 6 minutes. In subsequent operations (7–9 and 10–12 minutes), the gold content decreases sharply, the increase in recovery becomes insignificant and does not exceed 1–1,5 %.
2. It is advisable to take the control flotation time equal to 18–20 minutes (6 minutes from the main flotation + 13 minutes from the control flotation).
3. It is advisable to take the time of the first cleaning operation equal to 4–6 minutes. An increase in this time will lead to dilution of the concentrate with a slight increase in gold recovery.
4. The high gold content in the first fraction of the cleaning operation, which is almost three times higher than the average content in this concentrate, indicates the advisability of including the second cleaning operation in the flotation scheme. Similar results were obtained in work [32]. The authors of this article also proposed an additional stage of flotation processing for maximum gold recovery.

Open experiments on flotation of gravity separation tailings at different pulp densities

A decrease in the content of solids in the pulp to 20 % leads to a decrease in the yield of the concentrate and a drop in gold extraction into it. At 20 % solids, gold recovery in the total flotation concentrate was 83,29 %, which is 4,27 % less than at 27 % solids. Consequently, a decrease in the pulp density from the optimal value adversely affects the results of flotation. An increase in the pulp density above the optimal value has a less significant effect, leading only to the dilution of the concentrate with waste rock.

Open experience of flotation of gravity separation tailings according to the complete scheme

The test showed that a concentrate with a gold content of 129 g/t at a yield of 2,50 % can be obtained from the gravity tailings after the second cleanup. The total recovery of gold in flotation products was 87,65 % with a yield of 12,5 %. The gold content in the tailings of the control flotation was 0,60 g/t. These results can be considered as satisfactory, and in the selected mode it is advisable to set up a closed experiment to establish the ore concentration indices according to the gravity-flotation scheme (Fig. 5).

Closed experiment on gravity-flotation concentration of the original ore at the natural pH of the pulp

As a result of enrichment, the following products were obtained:

1) gravity concentrate with a gold content of 2426 g/t with an output of 0,31 % and an extraction of 64,06 %;
2) flotation concentrate (after the II cleaning) with a gold content of 122 g/t with an output of 2,90 % and an extraction of 30,01 %.

The total gold recovery in the gravity-flotation concentrate is 94,07 % with a yield of 3,21 % and Au content of 345,87 g/t. The gold content in the flotation tailings was 0,72 g/t.

Based on the data obtained as a result of studying the granulometric composition of the beneficiation products, it was found that gold extraction from the ore increases with a decrease in the size of the ore fractions. The lowest gold content in the tailings and the highest in the concentrate is noted in the –0,045 + 0 mm fraction. As the size of the tailings fractions increases, the gold content increases, and the content in the flotation concentrate fractions decreases.

To test the effect of pulp alkalinity on the concentration of the original ore according to the gravity-flotation scheme, a closed experiment was set up, in which flotation was performed with the addition of a small amount of sulfuric acid to activate gold-bearing pyrite. In this case, the pH of the pulp decreased from 7,64 (natural value) to 7,06.

This experiment has shown that carrying out flotation with the addition of sulfuric acid (in comparison with the experiment without it) led to an increase in the yield of the flotation concentrate from 2,90 to 3,61 %, i. e. by 24,5 % (relative). At the same time, gold extraction into the flotation concentrate increased from 30,01 to 30,22 % (by 0,7 % relative). Consequently, carrying out flotation with the addition of acid almost does not increase gold recovery, but at the same time leads to a significant dilution of the flotation concentrate and a decrease in the gold content in it, and can also lead to an increase in the tailings hazard category. Therefore, the addition of sulfuric acid during the flotation of the ore of the deposit was recognized as inexpedient.

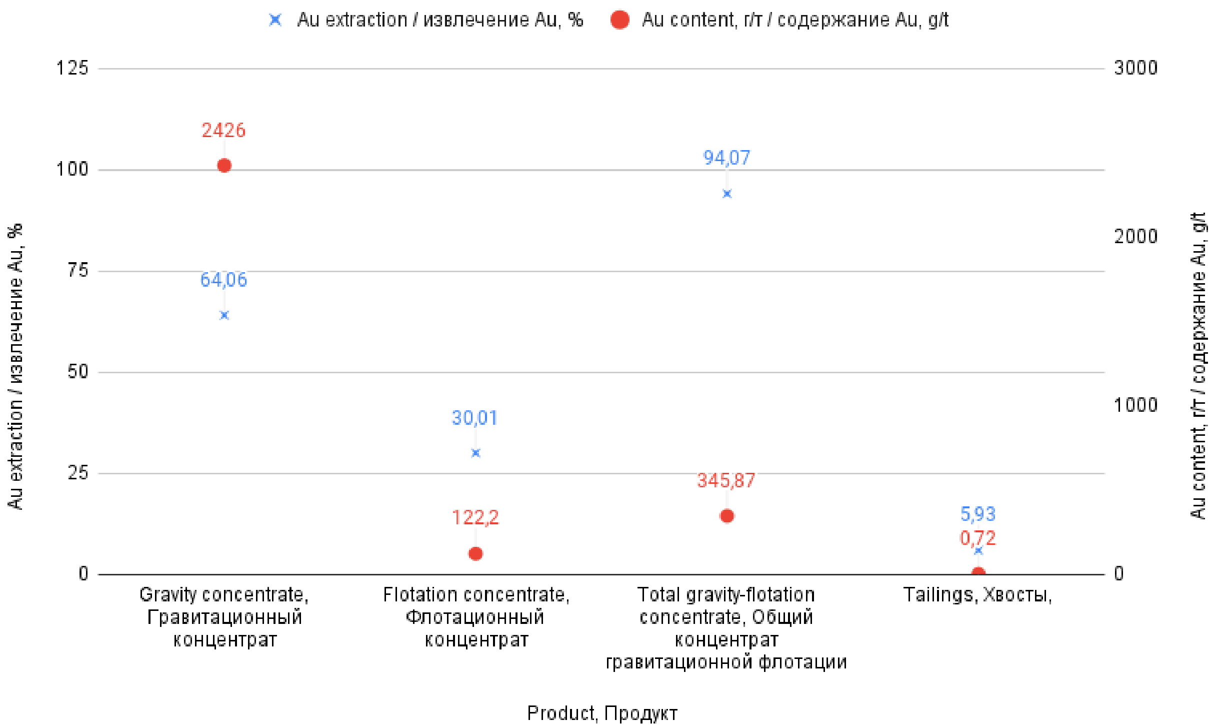

Fig. 5. *Results of ore dressing according to gravity-flotation scheme (closed experiment, natural value of pulp pH)*

Рис. 5. *Результаты обогащения руды по гравитационно-флотационной схеме (закрытый опыт, естественное значение pH пульпы)*

As in the previous experiment, the granulometric composition of the enrichment products shows that the ore fraction with a size of –0,045 +0 mm is best enriched. In this class, the minimum gold content in the tailings (0,46 g / t) and the maximum (111 g / t) in the flotation concentrate are noted.

Conclusion

As a result of a complex of studies, tests were carried out on gravity and flotation enrichment of gold-bearing ore: GRG test, stage test, modeling of gold in the grinding cycle was carried out, the optimal size of grinding of the initial ore and gravity tailings was established to define the technological parameters of flotation enrichment, studies on the kinetics of flotation, as well as experiments on ore enrichment according to the full scheme. As a result of the experiment, the feasibility of using the proposed gravity-flotation technological scheme was established and the following technological enrichment indicators were obtained: gravity concentrate with a gold content of 2426 g/t with a yield of 0,31 % and recovery of 64,06 %; flotation concentrate with a gold content of 122 g/t with a yield of 2,90 % and recovery of 33,01 %; total gold recovery in the gravity-flotation concentrate was 94,07 % with a yield of 3,21 % and Au content of 345,87 g/t; gold content in the flotation tailings was 0,72 g/t.

The data obtained and the proposed research methodology can be used to study the concentration of similar ores of various deposits. The proposed research methodology, taking into account the knowledge of the qualitative and quantitative characteristics of the ore, can significantly reduce the time of research by reducing the number of experiments.

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ТЕХНОЛОГИЯ ПЕРЕРАБОТКИ МАЛОСУЛЬФИДНОЙ ЗОЛОТО-КВАРЦЕВОЙ РУДЫ

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Актуальность *исследования обусловлена необходимостью получения новых знаний об обогатимости золотосодержащих руд и необходимостью развития ресурсной базы Российской Федерации, а именно вовлечения объектов недропользования в хозяйственный оборот.*

Цель: *исследование по переработке малосульфидной золото-кварцевой руды с применением комплексной технологии обогащения минерального сырья.*

Объект: *золотосодержащая руда со средним содержанием золота 11,88 г/т. Содержание серебра незначительное – 2,43 г/т. Основными рудными минералами в образце являются пирит и пирротин. Среднее содержание этих минералов в руде, по данным минералогического и рентгеноструктурного анализа, составило около 6 % (суммарно). Основными породообразующими минералами исходной руды являются: кварц – 60,1 %; кварц-хлорит-слюдяные агрегаты – 3,8 %; карбонаты – 7,1 %.*

Методы. *Методология исследования базировалась на теоретических основах обогащения полезных ископаемых. Изучение вещественного состава руды и продуктов обогащения было выполнено с использованием химического, пробирного, термического, спектрометрического методов, а также атомно-эмиссионной спектрометрии и других методов. Изучение обогатимости проводили методами гравитационного и флотационного обогащения, а именно: GRG тест, стадиальный тест института ТОМС (Россия) (определение оптимальной крупности измельчения руды и количества стадий обогащения), моделирование извлечения золота в цикле измельчения, определение оптимальной степени помола, исследование выбора флотационных реагентов, исследование кинетики процесса и проведение комплекса технологических исследований.*

Результаты. *Установлено, что извлечение золота при проведении GRG-теста составило 72,75 % при выходе общего концентрата 1,34 % и его содержании 664,78 г/т. При этом содержание золота в хвостах составило 3,29 г/т. Стадийные испытания показали, что при переработке руд только гравитационным способом целесообразно использовать двухстадийную схему. Первая стадия находится в цикле измельчения при крупности руды 60–70 % и вторая стадия – при конечной крупности насыпи классификации 90 % класса –0,071 мм. Центробежная сепарация как свободная операция извлечения золота в цикле измельчения работает эффективно. Получен концентрат с содержанием золота 2426 г/т с выходом 0,31 % и извлечением 63,74 %. Обогащение дробленых до 90 % хвостов первой очереди класса –0,071 мм на обогатительной фабрике KC-CVD (моделирование) позволило извлечь золото в общий гравиоконцентрат (KC-MD+KC-CVD) 87,25 % при содержании выхода фугата 22,63 %. Содержание золота в хвостах составило 1,97 г/т. Результаты гравитационно-флотационного обогащения исходной руды свидетельствуют о целесообразности использования комбинированной гравитационно-флотационной технологической схемы. В закрытом опыте первичного обогащения руды по гравитационно-флотационной схеме при естественном pH пульпы (без добавления кислоты) были получены следующие продукты: гравиоконцентрат с содержанием золота 2426 г/т при выходе 0,31 % и извлечении 64,06 %; флотоконцентрат (после II очистки) с содержанием золота 122 г/т с выходом 2,90 % и извлечением 33,01 %; общее извлечение золота в гравиофлотационный концентрат составило 94,07 % при выходе 3,21 % и содержании Au 345,87 г/т, содержание золота в хвостах флотации 0,72 г/т.*

Ключевые слова:
Золото, обогащение, технологические исследования, флотация, гравитация, моделирование, концентрат, хвосты, извлечение.

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