Processing of Mercury-Stibium Gold Bearing Ore of Djijikrut Deposit

Mahsuda Abdusalyamova¹, David Dreisinger², Zinaida Zinchenko¹, Petr Solozhenkin³, Safarali Gadoev¹, Ivan Tyumin¹ and Marhabo Ismailova¹

1. *V.I. Nikitin Institute of Chemistry, Academy of Science of the Republic Tajikistan, Dushanbe 734063, Tajikistan*

2. *Department of Materials Engineering, University of British Columbia, Vancouver 3096350, Canada*

3. *IPKON the RASci (Institute of Comprehensive Exploitation of Mineral Resources of the Russian Academy of Science), Moscow 111020, Russia*

Received: February 23, 2013 / Accepted: April 23, 2013 / Published: May 20, 2013.

Abstract: The mercury-antimony-gold ore was subjected of the flotation. As a result of enrichment of ore, the mercury-antimony concentrate with the contents of 51.5% Sb, 3.55% Hg and 6 g/t Au was received. Vacuum distillation of concentrate allocation in serial sublimates mercury and antimony was designed. The big amount gold passed to calcine after vacuum distillation. The gold in calcine roasting up to 900 °C was 30.6 g/t.

Key words: Ore, concentrate, flotation, cyanidation, roasting, mercury, antimony, gold.

1. Introduction

Tajikistan is rich in mercury-antimony deposits (Scalnoe, Djijikrut, etc.). The Republic takes the second place in the countries by having antimony resources and the fourth place in the world. Antimony-mercury ores of Djijikrut deposits are processed at Anzob (Tajikistan) by flotation to produce a mercury-antimony concentrate. However, for the lower horizons of the field, we can observe a sharp increase in the content of antimony, mercury, and gold which content is already more than 2 g/t in the ore. There are about 6.9 million tons of gold ore and 14 tons of gold in it.

Gold in these ores is presented in the form of nanoscale particles. Size of individual gold particles is about 2.8 μm.

All these predetermine the research on complex processing of the ores. The technology of complex processing of antimony-gold and gold-antimony ore must be based on the combined scheme in which the

elements of the “golden” and “antimony” technology are closely intertwined and complement each other, providing maximum extraction of both valuable components [1].

In the balance of the distribution of securities and associated components of the ore deposits Djijikrut, it was found that the same minerals are both hubs and carriers of certain elements. Thus, the hub and carrier of antimony is antimonite, mercury-cinnabar, hubs, and partly carriers of gold, silver, thallium, selenium, marcasite and pyrite. The main carrier of gold is non-metal raw materials of breccias.

Carbonaceous substance is widely spread in the deposit both in ore-bearing rocks and in the ores. There are two major associations, bearing gold: early carbonaceous-carbonate with sulfides (first sedimentary-digenetic type ores) and late gold-cinnabar-calcite (third type of supergene ore). Inclusions of early paragenesis gold are installed, mostly in clastic rocks.

A detailed study of mercury-antimony ores in polished sections showed that the inclusions of native gold are confined primarily to the remnants of clastic

Corresponding author: Mahsuda Abdusalyamova, professor, main research fields: pnictides rare earth elements. E-mail: amahsuda@mail.ru.

rocks, without showing the structural-textural subordination of antimony-quartz-dolomite breccia cement. The main quantity of gold in the ore falls along with the host breccias, which have undergone a phase of repetition changes till antimony period. This conclusion is consistent with the conclusion of the researchers on the scattering of 88.1% of non-metallic minerals, gold, and with the conclusion made by technologists about association of 48%-76% gold with non-metallic minerals.

The main hubs of gold are arsenopyrite and pyrite, forming a close intergrowths with marcasite. At an average content of pyrite and marcasite 1.4% in the ore in the balance of their share of gold is 12.5%. This is due to low concentrations of minerals in the ore (average 0.2%).

Thus, the main minerals of pyrite-marcasite-arsenopyrite, being the hub of gold, are not their carriers, i.e., they borrow it from the surrounding rocks. The role of quartz and dolomite in the distribution of gold is low in a very low content of gold in them.

2. Materials and Methods

Ore sample was investigated in which the metal content was: Sb—3.8%; Hg—0.4%; Au—2.5 g/t; Tl—29 g/t; Se—12 g/t; Te—14 g/t. Flotation was carried out in a mechanical flotation device with 250 g hinges. Nonionic low-toxicity surfactants were used as flotation reagents. Grade 4 toxicity and hazards, such as “Oksafor-1107” (O-1107), “Pemisol” and butyl xanthate, lead nitrate, frother-T-80, Formula O-1107 is shown on Fig. 1. “Pemisol” has the following composition: kumilfenol—1.5%-15%, the fraction of aromatic hydrocarbons with a boiling point of 135 °C-154 °C beginning and the end of the boil

Fig. 1 Formula of Oksafor-1107.

164 °C-270 °C-70%, non-ionic SAS-OP-10-15-28.5%.

Brand UV-visible spectrophotometer was used to record the spectrum of absorption. The characteristic band for xanthate is located in the field of 301 nm, O-1107 and “Pemisol”—289 nm. X-ray analysis was recorded on the device “Dron-3M”.

Cyanidation was carried out in bottles fitted with a tube for access to air, rotating continuously on the rolls, which made it possible to carry out the leaching process without using special oxidizing agents, such as barium peroxide, hydrogen peroxide, etc.. Weight of tailings in the experiments is 200 g. The ratio of S:L is 1:1.5. Leaching time is 24 h. During the leaching pH of the pulp was maintained in the range 10.5-11.5 with lime. Gold in solution was identified with atomic absorption spectroscopy.

3. Results and Discussion

3.1 Technological Researches of Ore

The technological scheme of flotation includes rougher, two scavenging and two cleaner flotation with extraction of antimony—87.5%, mercury—96.3%, thallium—62.4%, selenium—58.9%, tellurium—31.7%. Extraction of gold into mercury-antimony concentrate was 30%. The gold content in tailings of flotation is 1.7-1.8 g/t [2]. The distribution of metal in the products of enrichment is given in Table 1.

Table 1 The flotation of antimony ore in closed cycle.

Name of product	Yield (%)	Content (%)						Recovery (%)					
		Sb	Hg	Tl (g/t)	Se (g/t)	Te (g/t)	Au (g/t)	Sb	Hg	Tl	Se	Te	Au
Concentrate	5.96	40.7	4.11	256.6	108.6	57.1	12	87.45	96.3	62.4	58.9	31.7	30.9
Tailings	94.04	0.37	0.01	9.8	4.8	7.8	1.7	12.55	3.7	37.6	41.1	68.3	69.1
Ore	100.0	2.8	0.25	24.5	11.0	10.7	2.3	100	100	100	100	100	100

Typically, in industrial practice the xanthogenate is used as a collector to enrich antimony ores with pre-activation of antimonite by lead cations [3]. Cations of lead sorptioning on the mineral surface soil concentrate and prevent at metallurgical processing. In addition, lead nitrate is a highly toxic agent that negatively affects the environment. We proposed to use low-toxicity nonionic surface active agents which suggest excluding lead nitrate from the process. The test reagents are small toxic surface-active agents (Grade 4 toxicity and hazards), such as “Oksafor-1107” (O-1107), “Oksafor-43”, “Pemisol”. These reagents are widely used in the practice of flotation. O-1107 and “Pemisol” were studied in the flotation of the ore samples. Diphylic structure of molecules O-1107 predetermines surface activity of this reagent on the boundary surface of liquid-gas, that is demonstrated in its foaming capacity. Effective use of surfactants has been confirmed by us in the flotation of antimony-arsenic ore deposit “Scalnoe” [4]. Previously, we have installed a synergistic effect of studied SAS and butyl xanthate. The binary mixtures of these reagents have been studied on this basis. Table 2 shows the results of the flotation, which suggests that the studied mixtures of chemicals are effective and allow you to refuse from using toxic activator-lead nitrate.

It is known that at the flotation of antimonite by SAS, coordination compounds are formed on its surface. The electron source is the oxygen of ethylene oxide surfactants, and antimony acid, which is formed on the surface of antimonite that is able to attach electrons. Chernikh et al. [5], using electron paramagnetic resonance formation of complexes with charge transfer between the “Oksafor-1107” and strong acceptors benzoquinone is shown, indicating the donor ability of “Oksafor” and the formation of adducts between the xanthate and surface-active substances. To study the synergistic of these reagents the researches have been conducted on their sorption on the surface of antimonite. Sorption of xanthate

reduces with increasing concentration of surfactants, indicating the joint sorption of reagents on the surface of minerals (Fig. 2). Obtained results confirm the synergies of the mixtures of reagents.

3.2 Development of Technology for Extracting Gold from the Flotation Tailings

Further studies have been devoted to studying hydrometallurgical leaching of gold by cyanide. The apparatus was mounted for continuous ongoing process. The concentration of NaCN in the experiments was 0.05%-0.1%, pH was maintained at 10.3-10.6 by adding lime. 2, 4, 8, 24 h later after the start of the experiment stirring was stopped, to take the sampling aliquots of the slurry (20 mL) to determine the pH, the concentration of free NaCN and concentration of gold. Gold in solution was determined by atomic absorption spectroscopy. Tailings of

Table 2 The flotation of mercury-stibium ore by binary mixture of Xanthogenate (300 g/t) with “Oksafore-1107” and “Pemisol”.

Name of product	Yield (%)	Content (%)		Recovery (%)	
		Sb	Hg	Sb	Hg
Concentrate	7.46	45.46	3.64	94.82	93.91
Tailings	92.54	0.2	0.019	5.18	6.09
Ore	100.00	3.57	0.29	100.00	100.00
“Pemisol” —400 g/t					
Concentrate	7.65	42.6	3.32	90.28	94.18
Tailings	92.35	0.38	0.017	9.72	5.82
Ore	100.0	3.61	0.27	100.0	100.00

Fig. 2 Change of xanthogenate adsorption (Kx) on antimonite of reagents concentration: 1—Kx; 2—Kx + “O-1107”; 3—Kx + “Pemisol”.

flotation was established by cyanidation: 1) an increase in cyanide concentration of 0.1% has a positive effect on the extraction of gold, a further increase in concentration has no appreciable effect on the cyanidation process; (2) Regrinding flotation tailings fineness up to 94% of the class “-0.063 mm” makes it possible to reveal several gold, located in intergrowths, resulting in the increase of dissolution rate of gold. It was determined that an increase in the duration of the leaching of up to 48 h not only increases the extraction of Au to solution, but on the contrary, starting from 4 h after the start of the experiment, we can observe a decrease in the concentration of Au in solution, which can be explained by the presence of carbonaceous materials in the ore. Processing of carbonaceous ore has its own characteristics [6]. Carbonaceous or graphite sludge can adsorb dissolved gold and thereby impede the process of cyanidation. In order to reduce their impact, experiments were conducted with the change of the leach solution by the method of 3-stage-cyanide. As a result of the experiments, the total recovery of gold to the solution of the third stage of cyanide slightly increased and amounted to 48.5%. One way to reduce the negative impact of carbonaceous materials on the process of cyanide is pre-treatment of the material by kerosene. Treatment by kerosene prevented the sorption of gold by carbonaceous matter, but the recovery of gold remained low. In order to improve the cyanide process, the samples of flotation tailings were subjected to roasting at a temperature of 300 °C at different duration time. It is well known the low-temperature roasting of gold ores as a preparatory operation before cyanidation. Roasting is carried out at a temperature of 300-400 °C, which allows to increase gold extraction at subsequent cyanidation. This is due to better disclosure of the metal, due to cracking of the ore during roasting and due to the destruction of the integumentary structures on gold. Coatings on gold, which consisting of iron and manganese hydroxides, lost crystallization water and disintegrated after

calcination. From roasted tailings for 3 h, maximum gold extraction—41.5% was achieved. Considering the fact that gold is finely dispersed in the sample and related to host rock, tailings of the samples were regrinded to 95% of the class “-0.08 mm”, and then were subjected to roasting under conditions similar to the previous experiments. Fine grinding improves slightly opening of gold, but increases the rate of gold leaching. After 8 h of cyanidation and roasting for 2 h, 43.2% of gold goes to the solution. Increasing the roasting time up to 24 h does not affect the gold extraction. Oxidative roasting followed by cyanidation of calcine is one of the most common methods of processing of sulphide gold concentrates [7]. Gold-bearing sulfides were oxidized and converted into a porous oxides which well-permeable for cyanide solutions. Subsequent leaching of calcine allows to transfer the opening gold to cyanide solution. In ores of Djijikrut deposit, the main hubs of gold are arsenopyrite and pyrite, forming a close intergrowth with marcasite. Therefore, we carried out the oxidative roasting of tailings. The oxidation of pyrite begins at a temperature of 450-500 °C. The process proceeds with the formation of an intermediate product pyrrhotine, which is oxidized to magnetite and further to hematite:

At temperatures above 600 °C, oxidation of pyrite precedes its dissociation with the formation of pyrrhotine:

3.3 Cyanidation of Tailings

After 2 h roasting, samples of tailings were subjected to cyanidation. Cyanidation was carried out at a supported cyanide concentration—0.1%. Duration of cyanidation in all experiments was 24 h. Effect of roasting temperature on the leaching of gold from the tailings is shown in Table 3. As can be seen from the data, the optimum of tailings roasting temperature is

Table 3 Influence of roasting temperature on the results of tailings cyanidation.

T _{Roasting} (°C)	Time (h)	pH	Concentration of Au in solution (mg/L)	Recovery Au (%)
300	2	10.4	0.50	44.1
	4	10.3	0.54	47.6
	8	9.9	0.50	44.1
	24	9.8	0.47	41.5
400	2	10.0	0.52	45.9
	4	9.9	0.53	46.7
	8	10.0	0.53	46.7
	24	9.8	0.47	41.5
600	2	11.2	1.19	99.1
	4	11.4	1.19	99.1
	8	11.4	1.15	95.8
	24	11.4	1.01	84.2
750	2	12.0	0.96	84.7
	4	11.9	1.01	89.2
	8	11.9	1.00	88.2
	24	11.9	1.00	88.2

600 °C. In this case, almost all gold transfers to the solution. Temperature increase to 750 °C reduces the extraction of gold, which is consistent with literature data. It should be noted that after roasting sharply increases the rate of dissolution of gold for 2 h, gold recovery to solution is 99.1%.

3.4 The Leaching of Gold by Thiourea

According to its technological data, the leaching method which is presently applied to extract gold from ores is rather a complete process, however, it has negative feature as cyanic compounds used for this purpose are too toxic. Therefore, the objective of the present work is to test thiourea solution as a substitute for cyanic solvents of gold [8].

Gold dilution process in the solutions of thiourea runs according to the following reaction:

with instability constant $B = 3.2 \times 10^{-26}$, therefore, the gold standard potential decreases up to +0.38 V. This is how gold dissolution in aqueous acidic thiourea solutions contained Fe^{3+} ions as oxidant, which is explained.

As in the case with cyanidation process, the investigations have been made according to ordinary laboratory technique of bottle tests by stirring in open bottles. Tailings were placed in bottle and thiourea and sulfuric acid solutions were added, and in some cases oxidant was added, the stirring was done using rotating disks for the certain period of time. After 2, 4, 8 h after the initiation of the experience, rotating disks were stopped and aliquot selection (10 mL) was made to identify pH and gold concentration. Aliquot loss was compensated by adding the same volume of thiourea solution. The experiments have been carried out with flotation tailings which gold content is 1.8 g/t.

Thiourea concentration was 0.5%, sulphuric acid was 0.5%, oxidant (H_2O_2) was 0-0.06% in the solutions. Ratio of liquid to solid was 1.5%.

The results of leaching experiments carried out both in the presence of oxidant and without it were presented in Table 4. Gold extraction to the solution is unsatisfactory and does not exceed 21.3%. pH = 6, 4 leaching solution was after stirring with tailings was, that is not optimal condition to carry out thiourea

Table 4 Leaching of flotation tailings with thiourea in the presence of an oxidant.

Time (h)	Supplements					Solution		Extracting Au	
	Ore (g)	Water (mL)	$\text{CS}(\text{NH}_2)_2$ (g)	H_2SO_4 (conc.) (mL)	H_2O_2 -3% (mL)	pH	Au (mg/L)	g/t	%
Start	200	300	1.5	1.5	6				
2						6.5	0.21	0.315	17.5
4						6.4	0.215	0.322	17.8
8						6.4	0.208	0.312	17.3

pH of leaching solution = 1.5;

Original content of Au in the flotation tailings = 1.8 g/t.

Table 5 The results of thiourea leaching of gold from burnt tails after their processing with sulfuric acid.

Time (h)	Additives					Solution		Extracting Au	
	Ore (g)	Water (mL)	CS(NH ₂) ₂ (g)	H ₂ SO ₄ (conc.) (mL)	H ₂ O ₂ -3% (mL)	pH	Au (mg/L)	Au g/t	Au %
Start	200	300	1.5	1.5					
2						2.1	1.030	1.54	85.8
4			0.5			4.0	1.040	1.56	86.6
8						3.0	1.050	1.57	87.5

The oxidant-air.

leaching and that is presumably explained by the presence of acid-soluble minerals in tailings. This did not allow reducing pH medium to the required values. Therefore, ore was previously subjected to acidulation by sulphuric acid while stirring for 6 h.

Thus, in further experiments on thiourea leaching of gold the tailings flotation, the last acid treatment did not interact with sulfuric acid to be added to create an acidic environment, and pH during the experiments remained to be 2-4.

Thiourea leaching of gold from the flotation tailings discovered that the extraction of gold in solution did not exceed 39.5%. These results leaching were as presence of hydrogen peroxide, acting as an oxidant, and as without it. Therefore, in subsequent experiments, hydrogen peroxide has not been feed. Oxidant was atmospheric oxygen. The tails, baked for 2 h at temperatures of 400 °C and 600 °C.

The gold extracted in solution after baking at 600°C without acid 48.8%.

Only after acid processing and subsequently baking at 600 °C in 8 h 87.5% of gold was extracted into the solution (Table 5).

4. Physico-Chemical Basis of the Vacuum Distillation of Antimony-Mercury Gold Bearing Concentrates of Djijikrut Deposit

Antimony-mercury gold bearing concentrate contained 52.5% Sb, 4.1% Hg, 6 g/t Au and 259.6 g/t. Tl have been used X-ray analysis of the concentrate showed that main part of antimony is represented by

mineral antimonite (Sb₂S₃), and mercury-cinnabar (HgS). Mercury from concentrates is obtained mainly by pyrometallurgical method [9]. Pyrometallurgical process for the preparation of mercury is reduced to two methods: distillation roasting of mercury concentrate and subsequent condensation of mercury vapor and furnace gases. Despite the simplicity of the process, this method is dangerous for both staff and the environment.

In this regard, a method was developed for the separation of antimony and mercury by vacuum-thermal distillation of mercury. This method is based on the difference between saturated vapor pressure of mercury, cinnabar, and antimonite [10, 11]. Fig. 3 shows the experimental apparatus assembled for this purpose. Studies have been conducted on the effect of temperature, duration of the process, the residual pressure on the sublimation of mercury. The mercury volatilizes efficiently and selectively at 300 °C, residual pressure 0.79×10^{-4} Pa for 20-25 min (Table 6). Calcine of the distillation of mercury was analyzed by chemical, thermal, and X-ray phase analysis methods. Fig. 4 shows the resulting X-ray spectrum. As can be seen from Fig. 4, after vacuum sublimation of mercury the lines of cinnabar are absent in X-ray spectrum. Thermal analysis revealed a peak at 531 °C, which corresponds to the melting temperature of the mineral antimonite (melting Sb₂S₃—540 °C).

The resulting calcine contains 58.7% of antimony, 0.011% of thallium, 0.22% of arsenic, 5.8 g/t of gold.

5. Pyroselection of Calcine after a Preliminary Distillation of the Mercury

Gold-arsenic and gold-antimony concentrates are complex raw materials, and their processing by existing methods is related with technological difficulties and low yield. When antimony is melting, gold is distributed among all melting products [12]. In connection with this, the special scheme of antimony vacuum sublimation in the form of stibnite and arsenic compounds separately with concentration of gold in residues was developed [11]. Apparatus was collected using fractional condensation of antimonite and arsenic compounds by vacuum distillation (Fig. 5).

In order to study the fractional condensation of sulfides from the calcine the effect of temperature, duration, residual pressure on the separation of sulfide and gold concentrations in it have been studied. Influence of temperature on the degree of sublimation of the volatile components of the calcine was investigated at a residual pressure of 0.013×10^{-4} Pa for 1 h. Condensate deposited in the form of two rings of different colors after sublimation of calcine. Needle-shaped crystals of dark-silver, almost black in color, reminiscent of single crystals of antimony

Fig. 3 The experimental equipment for the sublimation of mercury from concentrate: 1—tube furnace; 2—thermocouple; 3—millivolt meter; 4—expansion chamber; 5—sewers; 6—filter; 7—manometer; 8—mercury manometer; 9—valve shut-off; 10—vacuum pump; 11—temperature control; 12—crucible; 13—exhaust valve.

Table 6 Content of antimony, mercury, thallium and arsenic in calcine after sublimation of mercury depending on residual pressure.

No.	Pressure (Pa·10 ⁻⁴)	Antimony (%)	Mercury (%)	Thallium (%)	Arsenic (%)
1	0.013	50.52	-	0.011	0.21
2	0.13	50.54	-	0.011	0.21
3	0.26	54.2	0.012	0.01	0.2
4	0.39	56.19	0.020	0.010	0.22
5	0.53	56.27	0.024	0.011	0.23
6	0.66	56.66	0.04	0.011	0.22
7	0.79	59.44	0.041	0.011	0.28

Fig. 4 X-ray calcine produced at 300 °C: 1—antimonite Sb_2S_3 ; 3—aurumpigmentum As_2S_3 ; 4—dolomite $\text{CaMg}(\text{CO}_3)_2$; 5—anorthite $\text{Ca} \{ \text{Al}_2\text{Si}_2\text{O}_5 \}$; 6—thallium natrolite $(\text{Ti, Na}) \{ \text{Al}_2\text{Si}_3\text{O}_{10} \} \cdot 2\text{H}_2\text{O}$.

Fig. 5 Equipment of vacuum selection of cinder: 1—TC (thermocouples); 2—tube furnace; 3—crucible; 4—a quartz tube; 5—glass stopper; 6—a trap for the exhaust gas; 7—temperature controller.

sulfide are deposited in the area of higher temperature. Mixture of arsenic sulfide, sulfur, and traces of stibnite was precipitated in the colder parts of the condensate of orange color. X-ray spectrum of condensation indicates the selective allocation of stibnite from the calcine. X-ray and silicate analysis of

the cinder at 900 °C showed the content of mostly silicates. Table 7 shows the material balance of sublimation process. The content of antimony in the residue decreased with increasing temperature from 700 °C to 900 °C from 14% to 2.63%, and the degree of sublimation of antimony, respectively, increased from 97% to 99.6%.

The sulfur content decreased from 8.93% to 8.67%, and the degree of sublimation of 94.4% to 95.8%. The distribution of Au, Ag in the cinder, depending on temperature, is shown in Table 8. The content of Au and Ag in the residue was determined by atomic absorption method. Thus, the vacuum distillation of antimony-mercury gold bearing concentrates of Djijikrut deposit and pyroselection of calcine after preliminary mercury sublimation allowed us to obtain commercial qualitative mercury and antimony products.

6. Investigation of Stibnite Oxidation Process in Furnace of Boiling Bed

Technological processes in furnaces of boiling bed

are often applied to process oxidizing ores and concentrates. Antimony sulfide oxidation study from stibnite was carried in the furnace of boiling bed.

Oxidation mechanism of sulfides of different metals up to present time can not be considered being investigated completely. Mainly, there are two theories that explain sulfide oxidation mechanism. According to one (sulphate) theory, sulfide oxidation runs through intermediate phase-sulphate formation, metal and sulfur oxides are obtained in the result of sulphate decomposition. Depending on temperature and condensate of sulfurous anhydride in gaseous phase the formed oxides can transfer into sulphates again. According to another, (oxide) theory, oxide formation happens directly from sulfides. Many scientists support combined theory. Many researchers have been dealing with antimony sulfide oxidation;

Table 8 Dependence of gold and silver content in cinder on distillation temperature.

Temp (°C)	700	800	900
Gold (g/t)	5.76	11.15	30.8
Silver (g/t)	660	1,040	1,020

Table 7 Material balance of sublimation process of gold containing antimony calcine.

No.	Name	Temp. (°C)	Weight (g)	% of calcine weight	Antimony			Sulphur		
					Content (%)	Quantity (g)	% of common quantity	Content (%)	Quantity (g)	% of common quantity
Charged:										
1	Calcine	-	7	100	65.97	4.893	100	17.6	1.232	100
Received:										
1	Black condensate (stibnite)	700	5.4004	77.1486	78.69	4.2524	86.908	19.2	1.037	84.16
2	Orange condensate		0.0855	1.2214	79.75	0.0682	1.393	18.9	0.0162	1.3116
3	Cinder in crucible		0.9117	13.024	14.09	0.1358	2.776	8.93	0.0814	6.608
			6.3972	91.494		4.4564	91.087		1.134	92.079
1	Black condensate (stibnite)	800	5.108	72.971	80	4.0864	83.515	19.2	0.9807	79.605
2	Orange condensate		0.3204	4.577	80.2	0.257	5.2524	19.1	0.0623	5.06
3	Cinder in crucible		0.7597	10.8529	11.32	0.08593	1.756	19.2	0.07597	6.176
			6.1881	88.4009		4.42933	90.523		1.0506	84.665
1	Black condensate (stibnite)	900	4.984	71.2	80.51	4.0126	82.006	20.8	1.037	84.145
2	Orange condensate		0.422	6.029	75.13	0.317	6.4786	18.2	0.0768	6.234
3	Cinder in crucible		0.584	8.34	2.63	0.01536	0.3139	8.67	0.0506	4.1098
			5.99	85.569		4.345	88.793		1.1644	94.4888

Table 9 Material balance of the oxidation roasting process in boiling layer.

No.	Roasting rate (°C/h)	Stibnite weight before and after roasting (g)	Sb content before and after roasting (%)	Sb amount before and after roasting (g)	Volatilization of antimony (%)	S content before and after roasting (%)	S amount before and after roasting (g)	Volatilization of sulfur (%)
1	350	4.0023	65.97	2.6403	9.1	17.6	0.7	35.7
		3.4682	69.14	2.3979		13.2	0.45	
2	400	4.0075	65.97	2.6437	7.13	17.6	0.7	84.8
		3.513	69.89	2.4552		3.04	0.10	
3	450	4.0029	65.97	2.6407	13.8	17.6	0.7	92.7
		3.2737	69.49	2.2748		1.57	0.51	
4	350	4.0023	65.97	2.64	10.6	17.6	0.7	90.85
		3.4101	69.49	2.36		1.9	0.064	
5	400	4.0034	65.97	2.64	9.8	17.6	0.7	82.42
		3.4313	69.38	2.38		3.6	0.123	
6	450	4.0026	65.97	2.64	17.045	17.6	0.7	99.27
		3.115	70.42	2.19		1.64	0.051	

however, there is no certain opinion on mechanism of this process [13].

The authors found that the antimonite oxidation begins at temperature at about 200 °C to 300 °C and runs comparatively slowly and depends on grain size. Shahov and Slobodskoy [14] showed that antimonite oxidation process is 180 °C. They think that up to 400 °C oxidation runs according to reaction:

However, besides this reaction antimony sulfide oxidation can run along with tetraoxide Sb_2O_4 formation, but Zhao [13] insist on the fact that oxidation up to tetraoxide runs after whole sulfide oxidation to trioxide. The best reaction conditions are:

temperature at 400 °C-500 °C and air surplus. Trioxide amount decreases and of tetraoxide increases depending on temperature growth. Besides, antimony forms Sb_2O_5 with oxygen, which is stable up to 357 °C, with further temperature increase it is decomposed with tetraoxide formation and then trioxide. Antimony tetraoxide decomposes into antimony trioxide and oxygen, when tetraoxide heating. Decomposition starts at 900 °C.

The authors studied oxidation process of the obtained stibnites after mercury sublimation in the furnace of boiling bed.

In the mounted apparatus the heating was electrical

and temperature did not exceed 450 °C. Since stibnite structure was very thin, stibnite was granulated. The coupling medium was aqueous solution of office glue.

Stibnite oxidation dependence on temperature, air rate and process duration have been investigated. Data on oxidation roasting at air rate 2.5 m/s and temperatures 300-450 °C are shown in Table 9. As it can be seen from the table, the sulfur has volatilized for 99% with temperature increase, and time of antimony sublimation was from 9% to 17%. In stibnite oxidation process non-volatile oxides Sb_2O_4 , Sb_2O_5 are formed.

These oxides are restored till volatile Sb_2O_3 at temperatures above 1,000 °C.

At that moment the following reactions are running in the furnace of boiling bed:

7. Conclusions

The technology of mercury-antimony gold bearing ore processing of Djijikrut deposit is offered. The technology of vacuum distillation of antimony-mercury gold bearing concentrates of Djijikrut deposit and pyroselection of calcine after a

preliminary distillation of mercury, which allowed us to obtain marketable products of mercury and antimony, is developed.

Acknowledgments

This work was supported by ISTC (International Science & Technology Center), # Project T-1598.

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