# Distribution of the complex sulfide raw stuff components on roasting with soda, aqueous and acid leaching of calcine

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The roasting thermodynamics of sulfides of molybdenum, rhenium and other components of complex concentrates in the presence of sodium carbonate and oxygen is calculated. In the presence of sodium carbonate, sulfides and oxides of these metals can form sodium molybdate, sodium perrhenate, sodium arsenate and sodium antimoniate. Aqueous solubility of sodium molybdate and sodium perrhenate assumes a possibility of rare metals as well as sulfur separation from non-ferrous metals, iron and waste rock. Part of silica and arsenic can be transferred into aqueous solutions.

In a sulfide-soda-sulfate technology, the pelletized batch roasting is proposed, and indicators of the major metals (Cu, Zn, Pb, Fe) extraction from the process flowsheet products are given. In this work, experiments on roasting in the conditions of an oscillating pulsating layer at temperatures of 550, 600, 650 °C were carried out with a mixture of concentrates in order to study the possibility of eliminating the batch pelletizing operations and the need for subsequent drying, as well as evaluating the distribution indices of valuable components. The roasting burden composition was 10 g of the weighed concentrates mix samples and 22 g of soda ash. The soda consumption has been chosen taking into account the sulfur total binding and adding 10% of excess on the impurities binding. Air consumption has ranged from 17 to 34 l/h, the burning duration was 90 minutes. Resulting from the analysis of the roasting process, aqueous and acid leaching of calcine, the summary is calculated and extraction of components into a sodium sulfate aqueous solution, metal sulfates acid solution and sulphate lead cake is determined.

Key words: metal sulfides, copper, iron, zinc, lead, molybdenum, rhenium, sodium sulfate, sodium carbonate, roasting, sulphatization, oxidation, leaching.

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### Introduction

S ulfide ores containing noble and rare metals are the raw materials for the copper, zinc, lead and other metals manufacturing.

In connection with the reserve depletion of deposits, more lean complex ores are being involved into processing. On enrichment according to the bulk differential flotation schemes, the indicators of recovery of metals into a bulk concentrate make up 85–93%, and at a stage of obtaining the bulk (copper, zinc and lead) concentrates, which satisfy the requirements of metallurgical processing, end-to-end recovery of non-ferrous metals into concentrates decreases to 72–75% of the ore [1].

In the production of heavy non-ferrous metals, hydrometallurgical (mainly pressure) [2–3] and pyrometallurgical methods [4] methods of processing raw materials, dust and industrial products are used; they ensure the processing of the raw materials relatively rich in the major metal. Currently, some progressive pyrometallurgical methods are used — the process of Vanyukov A. V. [5], flash smelting [6], processes with bubbling the melts with gases — Ausmelt, ISASMELT [7–8], but hydrometallurgical technologies nevertheless exceed the pyrometallurgical ones in terms of the raw materials use all-inclusiveness and the environmental indicators.

In general, the problem of the low-quality sulfide raw materials processing has not yet been solved. New technologies are needed to reduce costs at the enrichment stage, to increase the end-to-end metal recovery and all-inclusiveness on reducing costs. In the technology [9] it is proposed to use the oxidizing charge roasting with soda ash followed by aqueous and acid leaching of calcined products. A functional diagram of the technology is shown in Fig. 1. Previously, the behavior of the main components of concentrates at the main operations of the diagram has been studied [10-11], but it is also necessary to study the distribution of accompanying impurities (rhenium, molybdenum, etc.) to pass on to processing research.

It is known that a large amount of sulfur dioxide is formed in the process of the sulfide raw materials treatment. Sulfuric acid workshops process the gases with  $SO_2$  content higher than 1.5–4.5%. The total sulfur recovery

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Fig. 1. Functional diagram of the complex sulphide concentrates processing

from waste gases into sulfuric acid is 75-85%, which is not enough in the view of environmental protection.

In the technology under development [1], the sulfide burden roasting with sodium carbonate allows to bind practically all the sulfur of raw materials into a solid sodium sulfate and to separate it from the metal-containing cake-1 at the subsequent aqueous leaching. During aqueous leaching of sinter, it is possible to remove the sodium compounds of molybdenum, rhenium, arsenic, and a part of silicon into an aqueous solution. Further, when processing cakes by sulfuric acid leaching, it is desirable to transfer copper and zinc into a solution to the maximum, and to concentrate lead and noble metals in a small amount of cake-2, the processing of which is possible by melting into a crude metal.

For leaching processes, in the case of the metallic copper and sulfide residues presence in cake-2, it is advisable to use oxidants: oxygen, ozone, hydrogen peroxide, etc. In the development of hydrometallurgical technology, the authors of [12] have studied the use of ozone and improvement of the oxidation process using a microwave system, which can create favorable conditions for the sulfide minerals oxidation in aqueous media. It is noted that oxidative ozone leaching is appropriate for copper and iron sulfides as well as sulfides containing gold and silver.

The behavior and distribution of such impurities as arsenic, antimony and bismuth, contained in sulfide concentrates, in the processes of metallurgical processing is considered in [13]. The authors point out that during heat treatment, a small portion of these elements will remain an undesirable impurity in the obtained non-ferrous metal, and available methods for removing or extracting these impurities usually include hydrometallurgical methods. In our case, the distribution of these components is also determined by the roasting temperatures and the calcine leaching conditions.

Processing of sulfide concentrates using alkali element salts is associated with a quite high cost of reagents, which can be recovered by crystallization of commercial sodium sulfate or by obtaining sulfuric acid and NaOH alkali solution by electrodialysis methods. Some of the acid can be used at a stage of the cake acid leaching with extraction into solutions of copper and zinc, and the alkali solution may be treated with firing gases containing  $CO_2$  followed by the release of Na<sub>2</sub>CO<sub>3</sub>. The results of research on the roasting of the batch pel-

letized with soda and leaching of calcine were published in the paper [11].

Unfortunately, it is not possible to provide a technical and economic assessment of the proposed technology at the stage of laboratory studies, but already at the stage of the compound complex ores flotation, the exclusion of selection and cleaner stages will lead to a significant reduction of costs. At that, almost complete sulfur utilization into sodium sulfate provides environmental advantages.

### **Experimental**

Sodium carbonate of grade (p) according to GOST 83–79 was used in the work. To study the distribution of the main components of complex raw materials, a charge has been prepared as a blend of concentrates, the composition of which is given in Table 1. The ratio of concentrates in the blend was chosen in order to obtain the composition feasible in bulk flotation of complex raw materials. When processing ores of a particular deposit, the composition of concentrates may be different; this will require extra investigations.

Table 1. Composition of the concentrate blend

		Content, %									
Material name	vveignt, g	Cu	Pb	Zn	Fe	Мо	S	Re	SiO <sub>2</sub>	CaO	Al <sub>2</sub> O <sub>3</sub>
Balkhash concentrate	50	14.85	3.25	0.65	31	-	29.3	-	15.2	1.65	3.4
Pb – Zn Shalkiya deposit concentrate	40	3.99	40.61	22.27	2.4	-	26.94	-	2.7	0.3	0.8
Maleev Pb – Zn – Cu concentrate	50	5.99	10.58	27.29	17.30	-	35.96	-	0.24	0.005	0.1
Balkhash Mo lean concentrate	50	2.28	0.30	0.102	12.59	5.09	16.26	0.43	37.35	3.03	7.79
Calculated blend composition	190	6.92	12.24	12.06	16.53	1.34	27.11	0.11	13.87	1.24	3.02

Analysis of the reactor waste gases on  $CO_2$  was carried out on a Gazohrom-1 chromatograph, the check analysis on  $CO_2$ , CO,  $O_2$ ,  $SO_2$  was implemented on a MAG-6-T multi-channel gas-analyzer, and the original charge and solid products X-ray analysis after each experiment was executed on a SPARK-1 spectrophotometer.

The furnace temperature was measured and maintained at a predetermined level by a CK thermocouple and an IRT-5301 technological meter-regulator. The reaction zone temperature was measured with an additional thermocouple and a DT-838 digital device. The weighed material samples were weighed on a VLT-510-P electronic scale.

The roasting unit is assembled on a base of the module with furnaces from the Q-1000 derivatograph. The use of two furnaces on the rotary console allows to carry out 2-3 experiments per day. The power supply of the furnaces is regulated by a thyristor block. Photos of the unit and reaction crucible are demonstrated in Fig. 2.

A crucible with a charge of a sulphide and soda mixture is attached to an electromagnetic vibrator through a rodtube that feeds the blast. The weighed material sample in the crucible was 10–15 g. The crucible with the weighed material samples was covered with an alundum crucible with a diameter of 70 mm and a height of 134 mm. The volume of gases taken from the reactor for analysis through a tube in the reactor bottom was 1-2 l/h, and the volume of the blast fed into the reactor was from 17 to



Fig. 2. Unit for roasting in an oscillating pulsating layer:
a – general view; b – reaction crucible;
1 – electric furnace; 2 – outer crucible; 3 – crucible with weighed sample; 4 – bottom; 5 – control unit

34 l/h; the excess gases have been removed from the reactor through the leaks between the outer crucible and the base. The charge layer has received the blast through a ceramic sprayer installed in the center of the crucible bottom.

The air consumption was set at the beginning of the experiment with the help of rotameters with valves. The vibrator power supply voltage was regulated by a LATR laboratory autoformer, which allowed to vary the amplitude of pulsations in the working area from 0.1 to 1.0 mm.

The vibrator power of 25-30 W and the vibration amplitude of 0.1-0.3 mm provide mixing the entire volume of the fine-dyspersated burden layer weighing 10-15 g.

## **Results and Discussion**

# Thermodynamics of oxidation reactions of impurities in concentrates

In [10–11], the results of thermodynamic evaluation of the copper, lead, zinc and iron sulfides reactions with sodium carbonate during roasting in the presence of oxygen were presented, as well as the start temperature and the sulfides oxidation sequence was determined, the sulfides roasting and the cinders leaching process variables were estimated.

The thermodynamic characteristics of reactions of sulfides of molybdenum, rhenium and other components of complex concentrates in the presence of sodium carbonate and oxygen are calculated taking into account

the compounds given in the bases of HSC Chemistry 8 (HSC ver: 8.1.5, License: KazNTU) [14] and the reference edition of the Joint Institute for High Temperatures of the Russian Academy of Sciences (Russia) [15]. Calculated are the changes in the Gibbs energy of reactions, which allows estimating the chance of the sodium molybdate, sodium perrhenate, sodium arsenate and sodium antimoniate formation from sulfides. The results of the calculations are presented in Table 2.

Calculations of the Gibbs energy changes show that sulfides of the accompanying components of complex concentrates during oxidizing roasting in the presence of soda can react with formation of the water-soluble compounds. It follows from the data of Table 2 that sodium compounds can also be formed by the interaction of oxides with soda.

Tabl	e 2.
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The results of the Gibbs energy change calculation for possible reactions

Poantian	$\Delta G^{0}$ , kJ						
	500 °C	600 °C	700 °C				
$MoS_2 + 2Na_2CO_3 + 4.5O_2(g) = MoO_3 + 2Na_2SO_4 + 2CO_2(g)$	-1429.5	-1388.7	-1348.3				
$MoS_2 + 3 Na_2CO_3 + 4.5 O_2(g) = Na_2MoO_4 + 2 Na_2SO_4 + 3 CO_2(g)$	-1529.5	-1512.7	-489.9				
$MoO_3 + Na_2CO_3 = Na_2MoO_4 + CO_2(g)$	-99.95	-123.9	-141.7				
$2 \text{ ReS}_2 + 4 \text{ Na}_2 \text{CO}_3 + 9.5 \text{ O}_2(g) = \text{Re}_2 \text{O}_7(g) + 4 \text{ Na}_2 \text{SO}_4 + 4 \text{ CO}_2(g)$	-2784.2	-2719.5	-2655.4				
$\text{Re}_2\text{O}_7(g) + \text{Na}_2\text{CO}_3 = 2 \text{ NaReO}_4 + \text{CO}_2(g)$	-210.8	-201.5	-191.7				
$\text{ReS}_2$ + 2,5 $\text{Na}_2\text{CO}_3$ + 4.75 $\text{O}_2(g)$ = $\text{Na}\text{ReO}_4$ + 2 $\text{Na}_2\text{SO}_4$ + 2.5 $\text{CO}_2(g)$	-1497.5	-1460.5	-1423.5				
$As_2S_3 + 3 Na_2CO_3 + 6 O_2(g) = As_2O_3 + 3 Na_2SO_4 + 3 CO_2(g)$	-2066.6	-2011.2	-1955.6				
$As_2S_3 + 4 Na_2CO_3 + 6 O_2(g) = 2 NaAsO_2 + 3 Na_2SO_4 + 4 CO_2(g)$	-2310.5	-2261.8	-2211.9				
$As_2S_3 + 6 Na_2CO_3 + 7 O_2(g) = 2 Na_3AsO_4 + 3 Na_2SO_4 + 6 CO_2(g)$	-2530.5	-2498.5	-2465.4				
$Sb_2S_3 + 3 Na_2CO_3 + 6 O_2(g) = Sb_2O_3 + 3 Na_2SO_4 + 3 CO_2(g)$	-2057.8	-2010.9	-1954.8				
$Sb_2S_3 + 6 Na_2CO_3 + 7 O_2(g) = 2 Na_3SbO_4 + 3 Na_2SO_4 + 6 CO_2(g)$	-2359.3	-2342.9	-2315.5				

Given the good solubility of sodium molybdate and sodium perrhenate in water, one can assume that these rare metals will be separated from the major non-ferrous metals, iron and the barren rock components during the calcine aqueous leaching. The arsenic and antimony compounds can also pass into aqueous solutions.

Other factors can also influence the components distribution of complex concentrates, for example, in studies [16], revealed was the formation of active, oxidizing particles of oxygen and  $H_2O_2$  as a result of the interaction of sulfides with water due to the catalytic reactivity of sulfide surfaces. These data indicate the possibility of oxidation of sulfide minerals and extraction of metals into solutions when using such oxidants as hydrogen peroxide for leaching of calcines and cakes.

Silica  $\text{SiO}_2$  is practically always present in sulfide concentrates; it forms sodium metasilicate at a roasting temperature of 500–700 °C by a known reaction:

 $\text{SiO}_2 + \text{Na}_2\text{CO}_3 = \text{Na}_2\text{SiO}_3 + \text{CO}_2.$ 

Sodium silicate is soluble in cold water, the concentrated solutions form a colloidal solution of hydrosol  $SiO_2 \cdot n H_2O$ . This can lead to reducing the filtration rate of solutions after the calcine aqueous leaching. Taking into account that  $Na_2SiO_3$  decomposes in hot water, it is possible to pre-isolate  $SiO_2$  from solutions. Besides, so-dium silicate reacts with carbon dioxide forming soda and silica, which can be used to partially regenerate soda and to release silica:

 $Na_2SiO_3 + CO_2 = SiO_2 + Na_2CO_3$ .

# Roasting of a blend of concentrates, aqueous leaching of calcine and acid leaching of the cake

In [10–11], the influence of temperature and duration of roasting on the desulphurization degree of sulphides in the presence of soda was estimated.

In the process under study, the transformation degree of substances can be estimated by the amount of carbon dioxide released during roasting. Roasting of the concentrates blend was implemented at temperatures of 550, 600, 650 °C. Data processing has been carried out by the results of parallel experiments. Fig. 3. shows the dependence for the composition of gases at 550 °C; for other temperatures, the regularities of changes in the composition of gases are similar.

In the waste gases, the SO<sub>2</sub> content was up to  $10-20 \text{ mg/m}^3$  according to the MAG-6T device indications; this is hundredths of a percent of the carbon dioxide content and can be a sign of the sodium sulfate formation from soda through the stage of the sulfur dioxide generation in sulfide oxidation reactions.

Content CO<sub>2</sub> %, O<sub>2</sub> %, CO g/m<sup>3</sup>, SO<sub>2</sub>mg/m<sup>3</sup>



Fig. 3. Dependences of the components content in waste gases on the roasting duration at 550 °C:  $I - CO_2$  (vol %);  $2 - O_2$  (vol %); 3 - CO (g/m<sup>3</sup>);  $4 - SO_2$  (mg/m<sup>3</sup>)

In Fig. 4. are shown the dependences of the fractional conversion characterizing the burden desulphurization and the sodium sulfate occurrence from soda.



Fig. 4. The fractional conversion dependences on temperature and duration of the concentrates blend roasting 1-550 °C; 2-600 °C; 3-650 °C

Relying on the results of roasting a blend of concentrates, the highest values during roasting and subsequent leaching were obtained for a temperature of  $650 \, {}^{\text{o}}\text{C}$ . Further increase in the roasting temperature leads to the charge sintering and reducing the desulphurization indices, and as a consequence, a reduction of copper and zinc extraction into the acid leaching solutions.

After roasting, the resulting calcine weighing 20 g has been leached by distilled water (water volume was 100 ml (L:S = 5:1), temperature was 50–70 °C, duration was 30 min). The pulp has been filtered, and cake was additionally rinsed with water after the filtrate-1 selection into container. The duration of the blue filter filtration was approximately 30–60 minutes. In the water leaching filtrate resented are: Na<sub>2</sub>SO<sub>4</sub>, Na<sub>2</sub>CO<sub>3</sub>, Na<sub>2</sub>SiO<sub>3</sub>, Na<sub>2</sub>MoO<sub>4</sub>, Na<sub>2</sub>ReO<sub>4</sub>, and also perhaps Na<sub>3</sub>AsO<sub>4</sub>, Na<sub>3</sub>SbO<sub>4</sub>.

The presence of sodium molybdate in the aqueous filtrate was evaluated by the reaction of the molybdenum blues production after neutralization of the aqueous solution from pH = 8-9 to pH = 3-4 with sulfuric acid solution

Table 3.	
Balance of the charge roasting at 650 °C,	aqueous and acid leaching of calcine

Came in	Total, g	Cu	Pb	Zn	Fe	Мо	Re	S	С	O <sub>2</sub>	The rest
Charge SK	10.0	0.7	1.2	1.2	1.4	0.13	0.01	2.5	0.0	0.0	2.8
Content, %		6.7	12.2	12.3	14.3	1.27	0.11	25.0	0.0	0.0	28.3
Soda Na <sub>2</sub> CO <sub>3</sub>	9.9	0.0	0.0	0.0	0.0	0.00	0.00	0.0	1.1	1.1	7.7
Content, %		0.0	0.0	0.0	0.0	0.00	0.00	0.0	11.3	10.8	77.9
Air	51.2	0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.0	11.0	0.5
Content, %		0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.0	21.4	78.6
Water	150.0	0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.0	133.4	16.7
Content, %		0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.0	88.9	11.1
Solution H <sub>2</sub> SO <sub>4</sub>	111.0	0.0	0.0	0.0	0.0	0.00	0.00	6.5	0.0	88.1	16.4
Content, %		0.0	0.0	0.0	0.0	0.00	0.00	5.9	0.0	79.4	14.7
Solution H <sub>2</sub> O <sub>2</sub>	11.1	0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.0	10.1	1.1
Content, %		0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.0	90.5	9.5
SUBTOTAL	34w3.2	0.7	1.2	1.2	1.4	0.13	0.01	9.0	1.1	243.6	84.8
Obtained	Total, g	Cu	Pb	Zn	Fe	Мо	Re	S	С	0 <sub>2</sub>	The rest
Roasting gases	48.5	0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.4	2.9	45.1
Content, %		0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.8	5.4	93.8
Recovery, %		0.0	0.0	0.0	0.0	0.00	0.00	0.0	39.5	1.2	99.9
Solution Na <sub>2</sub> SO <sub>4</sub>	171.9	0.0	0.0	0.0	0.0	0.12	0.01	1.3	0.8	149.9	19.8
Content, %		0.0	0.0	0.0	0.0	0.08	0.01	1.1	0.5	87.2	11.2
Recovery, %		0.0	0.0	0.0	0.0	92.0	93.8	14.3	70.3	61.5	43.8
Cake-2 KB	3.2	0.1	1.2	0.1	0.9	0.00	0.00	0.1	0.0	0.4	0.4
Content, %		3.4	38.0	3.3	27.1	0.05	0.01	4.2	0.0	12.6	11.3
Recovery, %		17.5	98.6	9.0	61.2	1.12	3.54	1.2	0.0	0.2	0.9
Solution MeSO <sub>4</sub>	142.0	0.6	0.0	1.2	0.6	0.00	0.00	5.5	0.0	111.1	23.1
Content, %		0.4	0.0	0.8	0.4	0.00	0.00	5.4	0.0	77.3	15.6
Recovery, %		89.3	0.0	95.3	39.3	0.00	0.00	61.6	0.0	45.6	51.1
SUBTOTAL	365.6	0.7	1.2	1.3	1.4	0.12	0.01	6.9	1.2	264.2	88.4
Imbalance	22.4	0.0	0.0	0.1	0.0	-0.01	0.00	-2.1	0.1	20.7	3.9



Fig. 5. X-ray photographs of the acid leaching cake

#### Table 4.

Phase content in an acid leaching cake sample

Compound Name	Formula	S–Q
Lead Sulfate	Pb(SO <sub>4</sub> )	35.6
Silicon Oxide	SiO <sub>2</sub>	25.6
Hematite	Fe <sub>2</sub> O <sub>3</sub>	12
Zinc Iron Oxide (a)	(Zn <sub>.06</sub> Fe <sub>0.94</sub> )(Fe <sub>0.98</sub> Zn <sub>1.02</sub> )O <sub>4</sub>	9.3
Zinc Iron Oxide (b)	Zn <sub>.945</sub> Fe <sub>1.78</sub> O <sub>3.71</sub>	9
Sphalerite, ferroan, syn	Zn <sub>0.825</sub> Fe <sub>0.175</sub> S	8.5

and the following reduction of the resulting molybdenum acid.

The yield of cake-1 of aqueous leaching was 7–8 g. Cake-1 has been leached by a solution of sulfuric acid. Consumption of the acid solution with concentration of 180 g/l was 100 ml, the temperature was 50–70 °C, and the duration was 60 min. On acidic leaching, 10 ml of 30% hydrogen peroxide was added for oxidation of Cu and Cu<sub>2</sub>O. The pulp has been filtered, and cake-2 has been rinsed with hot water after the filtrate selection into container. The duration of the blue filter filtration was approximately 90–120 minutes. In the filtrate presented are: CuSO<sub>4</sub>, ZnSO<sub>4</sub>, FeSO<sub>4</sub>, perhaps H<sub>2</sub>SiO<sub>3</sub>, H<sub>2</sub>MoO<sub>4</sub>, H<sub>2</sub>ReO<sub>4</sub>, also possible H<sub>3</sub>AsO<sub>4</sub>, H<sub>3</sub>SbO<sub>4</sub>.

The calcined products, products of aqueous and acid leaching were analyzed by express analysis on a SPARK-1

*X*-ray fluorescence device. Based on the analysis of the main components (Cu, Pb, Zn, Fe) and desulphurization estimates for the content of S, the author's program Balance has been used to calculate material balances and distribution of components. The indicators of the summary balance obtained for the burden roasting at 650  $^{\circ}$ C are listed in Table 3.

From the data of Table 3 it follows that Mo and Re are extracted into a solution of sodium sulfate at 92.0 and 93.8% on roasting the concentrates with soda and subsequent aqueous leaching of calcine. Copper and zinc are extracted in an acidic leaching solution at 89.3 and 95.3%, and iron passes into a solution at 39.3% and into a cake at 61.2%. The lead remains completely in the acid leaching cake.

Fig. 5 and Table 4 give the results of *X*-ray phase analysis, which has demonstrated the presence of lead sulfate, silica, ferric oxide and zinc ferrites in the acid leaching cake.

The lead-containing sulfate cakes can be treated with soda solutions with concentration of 200-250 g/l for the purpose of carbonization by the following reaction:

$$PbSO_4 + Na_2CO_3 = PbCO_3 + Na_2SO_4$$

The carbonate deposit is further reduced by melting to a crude metal, into which the impurities of noble metals are extracted.

In aqueous and acidic solutions from leaching, quantitative elemental analysis of the atomic absorption spectra has been conducted by atomic absorptive spectrometry on a SavantAA device. In the aqueous solution the content of elements was, mg/l: 0.33 Sb; 8.5 As. In the acidic solution the content of elements was, mg/l: 2315 Cu; 3250 Zn; 2275 Fe; 5.8 As. The presence of SiO<sub>2</sub> in these solutions was not detected, which is explained by relatively low roasting temperatures.

### Conclusions

Oxidative roasting of a blend of concentrates in the presence of soda ash, as well as aqueous and acidic leaching of the calcined products was studied.

The thermodynamic calculation of reactions in the metal sulfide — sodium carbonate — oxygen systems, for the accompanying valuable components of complex raw materials — sulfides of molybdenum rhenium, arsenic, antimony is carried out.

Taking into consideration the analysis of raw materials and products for the main components (Cu, Pb, Zn, Fe and S), the author's program Balance has assisted to calculate the material balances and distribution of components. The calculated parameters allow us to note that Mo and Re are extracted into a solution of sodium sulfate at 92.0 and 93.8% on roasting the concentrates with soda and subsequent aqueous leaching of calcine. Copper and zinc are extracted in an acidic leaching solution at 89.3 and 95.3%, and iron passes into a solution at 39.3% and into a cake at 61.2%. The lead remains completely in the acid leaching cake.

The results of X-ray phase analysis has showed the presence of lead sulfate, silica, iron oxide, difficult-to-leach zinc ferrites in the cake of acid leaching. Lead bullion, which is a collector of impurities of noble metals, can be smelted from sulphate lead cake after carbonization.

The results of the work do not pretend to an interpretation in its entirety of possible implementation options used in the proposed scheme of processes and indicators of the valuable components distribution. The diversity of complex raw materials will require extra investigations and may attract the attention of researchers and experts to this topic.

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