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Methods of silica removal from pyrometallurgical processing wastes of ilmenite concentrate

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ABSTRACT

This article presents a study on the processing of waste dust from electrical smelting of ilmenite concentrates with the removal of silica from them by alkaline and fluoride methods. The study of the smelting dust leaching by caustic soda solutions included investigation of the effect of sodium hydroxide concentration, process time, temperature, S:L ratio. The optimum conditions of concentrate electric smelting dust leaching - temperature 80-90 °C, duration 90-120 minutes, S:L ratio = 1:5, sodium hydroxide solution concentration 110-115 g/dm^3 were determined. The optimum conditions for fluorination of electric melting dust were determined, at which the sublimation degree of silicon fluoride was 84.2 %. Studies have been performed to decomposite obtained silicon-containing sublime in the presence of ammonia agent. The optimum pyrolysis modes that provide the separation of fluoride and silicon oxide - temperature 530-560 °C and duration of 60-80 min have been determined based on the results of thermal analysis and studies on the process duration effect. The silicon oxide content in the obtained product was 96.3%.

Keywords: fine dusts, leaching, sodium hydroxide, silicon dioxide, fluorination

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Introduction

The largest producers of titanium sponge are China, Japan, Russia, Kazakhstan, the USA, and Ukraine [[1], [2], [3]]; and one of the leading suppliers is Kazakhstan enterprise - Ust-Kamenogorsk Titanium-Magnesium Plant JSC (UKTMP JSC) that produces about 18 % of the world sponge titanium production. The raw material used to produce titanium is ilmenite concentrate that is reductively smelted to produce titanium slag and substandard pig iron. UKTMP JSC uses a one-stage electric smelting of ilmenite concentrates to produce titanium slag and pig iron, the charge for smelting is supplied in a loose state accompanied by a high dust content. During the smelting of ilmenite concentrates at 1600-1700 °C, silica contained in the charge is sublimed, and together with gases is entrained into the gas duct system, it condenses as amorphous silica SiO_2 in scrubbers and falls into fine bag filters. Due to the high silica content, the dust cannot be recycled to the smelting process or fed to the chlorinators. In the first case, high silica content causes boiling of the melt and in the second case, the presence of silica will affect the quality of titanium tetrachloride produced during slag chlorination, because subsequently the silica will transfer to titanium tetrachloride and deteriorate the grade of titanium sponge. Because of the inability to recycle the captured dust back into the process, it is

deposited together with other solid waste in designated areas, landfills. Annually at maximum capacity utilization UKTMP produces up to 76,000 t of chloride wastes, including about 600 t of fine sleeve filter dust. Under the influence of natural precipitation and wind, the waste is eroded and dispersed, polluting water and soil basins [4]. The enterprise has to pay huge fines for the maintenance of the accumulated waste.

The main sources of industrial production of precipitated silica are silicate blocks prepared by the fusion of sand with sodium hydroxide [5]. The essence of obtaining precipitated silica from silicate blocks is as follows: the block is obtained by fusion of sand with sodium hydroxide at 1700 ^{0}C, then it is boiled in an autoclave at high temperature and pressure. Amorphous silica is extracted from the obtained solution of sodium silicate after its clarification and dilution by carbonation with carbon dioxide, then neutralization of the solution with sulphuric acid. The process used in industry to produce precipitated silica ("white carbon black") is is energy and labor-intensive.

Waste gases absorbed in the production of wet-process phosphoric acid and superphosphate containing a mixture of gaseous silicon tetrafluoride and hydrogen fluoride produce silica-silica solution [6]. In the studies [[7], [8], [9]], silicon dioxide is obtained by mixing silicicofluoric acid solution or a mixture of hexafluorosilicate and ammonium fluoride solutions with ammonia water. The specific surface area of the resulting white carbon black is 100-220 m^2/g. In the way [10] silica gel is used to produce silica that is a waste product of aluminum fluoride production, in this way the silica gel is treated in suspension by a mixture of ammonia and water steam at a ratio of ammonia to water steam 1:30-100 at 500-700 °C. In another process [11], to obtain silicon dioxide and aluminum fluoride used in aluminum metallurgy, a solution of silicon hexafluoric acid is mixed with an aluminum hydroxide suspension, as a result of their interaction to obtain a solution of aluminum fluoride and silica gel precipitate. In the method [12] inactive silica gel is treated with a mixture of ammonium fluoride and sulphuric acid, with subsequent neutralization of silica gel with ammonia, separation, washing, and drying of silica precipitate. Strong mineral acid H_2SO_4 is used to dissolve silica gel, so an acidic silica solution is formed, and during neutralization with ammonia silica precipitation occurs in the presence of sulphation that significantly deteriorates the quality of the main product. The disadvantage is also the formation of a by-product - ammonium sulfate, the use of which is very limited.

As mentioned above, due to the high silica content, dust from the electric smelting of ilmenite concentrates cannot be recycled back into the smelting process. In this research work, two methods have been shown to remove silica from electrosmelting dusts. The first method is by leaching dust with sodium hydroxide solutions, the second method is by hydrofluorination of dust and sublimation of silicon fluorides. Both methods make it possible to remove silicon from the dust, after which the dust can be returned to electric smelting.

Methods of analysis

X-ray experimental data were obtained on BRUKER D8 ADVANCE apparatus on copper radiation at accelerating voltage 36 kV, current 25 mA.

X-ray fluorescence analysis was performed on a Venus 200 PANalytical B.V. wave dispersion spectrometer. (PANalytycal B.V., Holland).

Chemical analysis of samples was performed on an optical emission spectrometer with inductively coupled plasma Optima 2000 DV (USA, PerkinElmer).

Mapping of elemental and phase composition of samples was performed on electron-probe microanalyzer JXA-8230 by JEOL (Japan).

Thermal analysis was performed on a TG-DTA/DSC synchronous thermal analyzer with a Jupiter STA 449 F3 quadrupole mass spectrometer (Germany).

Materials: sodium hydroxide grade "high" ("Kaustik" JSC, Russian Federation). The fine dust of electric smelting of ilmenite concentrate, provided by UKTMP JSC, Republic of Kazakhstan, the content of the main components is given in Table 1.

Table 1 - Contents of the main components of the electric smelting dust of ilmenite concentrate, wt.%

Content, wt. %						
TiO_2	Fe_2O_3	SiO_2	ZnO	MgO	Cr_2O_3	MnO_2
46.37	26.90	10.04	3.18	1.55	0.45	2.90

Methods of experiments

Experiments on leaching with sodium hydroxide were performed in thermostatic reactors of 0.5 dm^3 volume. The slurry was stirred with a glass stirrer. A certain amount of sodium hydroxide solution was poured into the reactor and heated to a given

temperature. When the temperature reached the desired value a sample of dust was added and leaching was started. At the end of the process, the pulp was filtered and the cake was washed with distilled water. The content of uncontrolled components in the washed cake was determined.

During the experiments, the following components were used: pure sour ammonium fluoride GOST 9546-72, 10 and 25% ammonia, the dust of bag filters for electrofusion of ilmenite concentrates of UKTMP JSC.

The methodology of the experiment was as follows. A sample with thoroughly mixed ammonium bifluoride and dust in a certain ratio was transferred into an alundumina boat that was placed in a steel tube located in a horizontal tubular furnace. Argon was fed through the steel tube and the furnace was heated to a predetermined temperature within a certain time interval. At the end of the experiment the outgassed ammonium hexafluorosilicate was collected at the end of the steel tube and the gas-air mixture was captured in a flask with ammonia water. The ammonium hexafluorosilicate and the remaining char in the flask were subjected to ammonia alkaline hydrolysis. After alkaline hydrolysis amorphous silica was subjected to pyrolysis to distill the remaining fluorine. The content of the components was determined by chemical and X-ray fluorescence methods.

Results and discussion

Destruction of silicate bases of electro-smelting dusts of ilmenite concentrates can be performed by the so-called alkaline desiliconization method that consists of leaching dust in solutions of sodium hydroxide. In this approach the silicates have to be dissolved, with silicon passing into the alkaline solution as a soluble sodium silicate - Na_2SiO_3 and titanium must remain in an insoluble residue.

Physico-chemical properties of the dust of electro smelting of ilmenite concentrate: the results of XRD analysis of the dust are shown in Figure 1.

The data of X-ray diffraction analysis shows that the substance of the dust sample is in the X-ray amorphous state and the background of the diffractogram is high.

It should be noted that iron in the dust is in trivalent state and the harmful impurity silicon is connected with magnesium.

The content of trace impurities and forms of entrails in the dust of electro-smelting of ilmenite concentrate was determined by electron microscopy (Figures 2, 3). The presence of particles of solid solution $nFe_2O_3 \cdot mTiO_2$ was established (Figure 3).

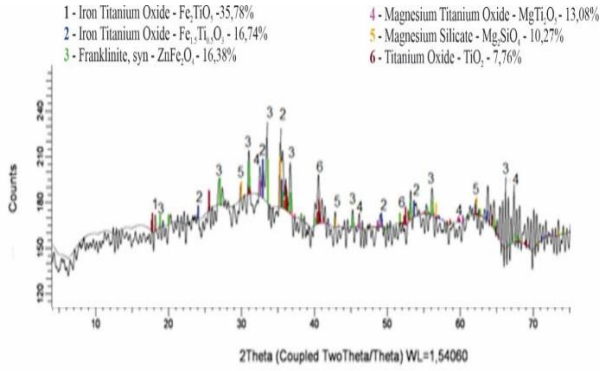

Figure 1 - Diffractogram of electric smelting dust of ilmenite concentrate

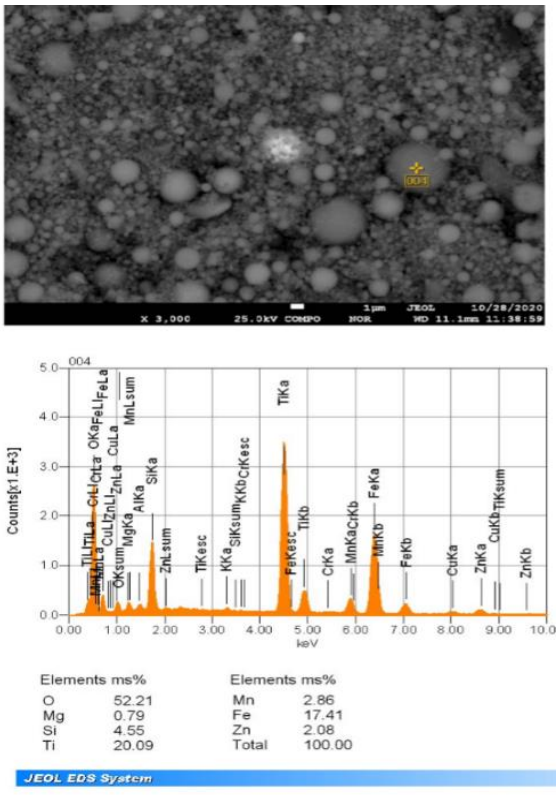

Figure 2 - Image and spectrum of [MnO $2TiO_2$]·[$Fe_2O_3 \cdot TiO_2$] anosovite particles in SiO_2 and ZnO cover

The phase which radiographically characterized as Fe = Mn – TiO [13] system may be referred to anosovite (Figure 2). It is noted that a part of anosovite particles is in the cover from oxides of silicon and zinc (Figure 3) and a part is in the cover from oxides of silicon, zinc, and lead. In addition, rare earth metal phosphates and particles of lead

and zinc oxides are present in the electric smelting dust of the ilmenite concentrate. The image obtained in the secondary electrons showed the fine dispersion of the object. The results of physicochemical investigations of the dust of electro-smelting of ilmenite concentrate showed that part of titanium is bound in hard-to-recover anosovite that can be enclosed in a shell of amorphous silicon oxide.

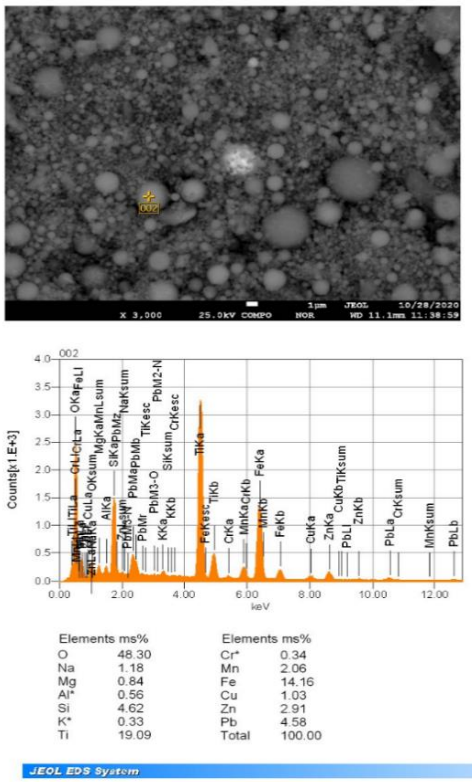

Figure 3 - Image and spectrum of anosovite particles in SiO_2, ZnO, PbO cover

The fine dust condition should contribute to the efficiency of the leaching of the injurious impurity, silicon.

Effect of concentration of sodium hydroxide solution. Study of the influence of concentration of sodium hydroxide solution on the extraction of silicon, chromium, manganese, zinc, and iron in the solution was performed in the concentration range of 50-130 g/dm^3. The duration of the experiments was 2 h, S:L = 1:5. The stirrer speed was 600 rpm.

Figure 4 shows curves of the degree of leaching of controlled elements into the solution. It is seen from the course of the curves that silicon leaches into the solution most completely - 77.7 %. It is explained by the good solubility of sodium silicate in alkaline solutions.

Increasing the concentration of sodium hydroxide in the leaching of electro-smelting dust of ilmenite concentrate led to a decrease in the cake yield (Figure 5).

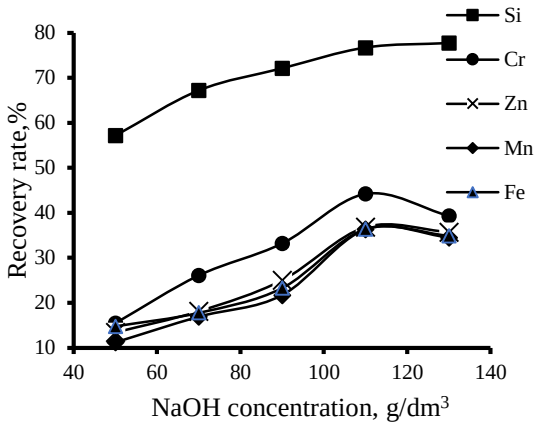

Figure 4 - Dependencies of the degree of leaching of controlled elements into solution on the concentration of sodium hydroxide

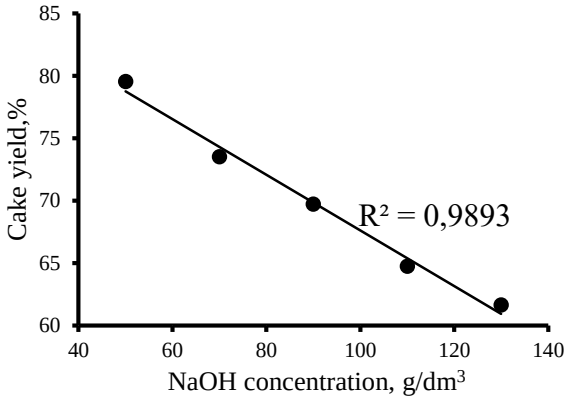

Figure 5 - Dependence of cake yield on sodium hydroxide concentration

The chromium leaching degree is considerably lower – 44.4 %. Franklinite decomposes to form hydroxo complex $Zn(OH)^-_3$ [14]. Silicon and iron in combined presence in the alkaline solution can form different iron-silicon complexes. This fact increases the solubility of iron in alkaline solution [[15], [16]].

The curves of solubility of metal oxides on alkali concentration have ascending and descending branches with a distinct maximum. Under the conditions of current studies, the maximum is reached at sodium hydroxide concentration of 110-115 g/dm^3.

Thus, it was experimentally determined that the optimum concentration of sodium hydroxide for leaching of electro-smelting dust of ilmenite concentrate is 110-115 g/dm^3.

Effect of the leaching process duration. Effect of duration of leaching of silicon, chromium, zinc, manganese, and iron from electrical smelting dust of

ilmenite concentrate was studied in the range of 15-120 minutes, temperature 80 °C, S:L = 1:5, sodium hydroxide concentration 130 g/dm^3. The stirrer speed was 600 rpm.

Figure 6 shows that even in the first 15 minutes of leaching the degree of transition of silicon in an alkaline solution reaches a significant value of 57.7 %. At the same time, the recovery of other controlled impurities does not exceed 13-18 %.

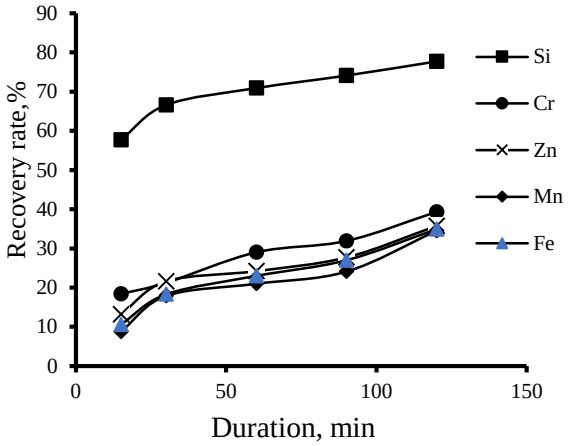

Figure 6 - Effect of leaching duration on the extraction of silicon, chromium, zinc, manganese, and iron in the alkaline solution from the electric smelting dust of the ilmenite concentrate

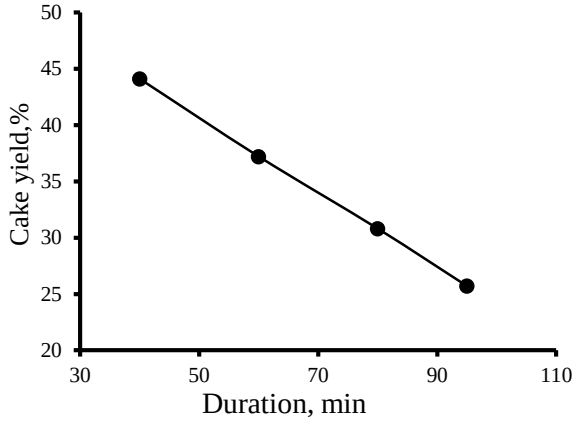

Figure 7 - Dependence of cake yield on the duration of leaching of electrowinning dust of ilmenite concentrate (80 °C, S:L = 1:5, NaOH concentration 130 g/dm^3)

Increasing the duration of the dust processing with an alkaline solution beyond 90 minutes does not cause a significant effect.

With increasing duration of the process of interaction of smelting dust of ilmenite concentrate with alkali-soluble formations pass into solution and cake yield decreases (Figure 7).

Thus, the optimum duration of leaching of electro-smelting dust of ilmenite concentrate with sodium hydroxide solution is 1.5-2 hours.

Effect of temperature on leaching process. The influence of leaching temperature on the extraction of chromium, silicon, zinc, manganese, and iron in solution was studied in the range of 40-95 °C. The duration of the experiment was 2 h, S:L = 1:5, sodium hydroxide solution concentration was 130 g/dm^3. The stirrer speed was 600 rpm.

It follows from the data in Table 2 that with increasing temperature from 40 to 60 °C the silicon extraction degree increases by 23 %, further increasing of leaching temperature from 60 to 80 °C leads to less considerable silicon extraction degree increase - by 13 %. Dust leaching at 95 °C allowed 82.8 % of silicon to be transferred into a solution that is only 5 % more than at 80 °C.

Table 2 - Effect of the temperature of the electric smelting dust leaching process of the ilmenite concentrate on the degree of extraction of the controlled components in the solution, %

Temperature, °C	Cake yield, %	SiO_2	Cr_2O_3	ZnO	MnO	Fe_2O_3
40	88.2	41.8	3.8	6.5	1.0	3.4
60	74.4	64.7	32.8	19.4	16.5	17.8
80	61.6	77.7	39.3	36.8	36.3	36.4
95	51.4	82.8	42.0	43.8	43.5	41.1

The behavior of other controlled impurities is similar to that of silicon when the temperature regime of the leaching process is changed.

Therefore, the optimum temperature for leaching of electro-smelting dust of ilmenite concentrate is 80-90 °C.

Effect of S:L ratio on leaching process. The research of influence of the ratio of smelting dust of ilmenite concentrate to sodium hydroxide solution was performed in the range 1:4÷10 at 80 °C, time - 120 min, stirring speed 600 rpm, sodium hydroxide solution concentration 130 g/dm^3.

Analysis of the data presented in Table 3 showed that changing the ratio of solid to liquid 1:5 or more has little effect on the extraction of chromium, zinc, manganese, and iron in the solution.

An increase in the volume of alkaline solution per unit mass of dust from 1:3 to 1:8 leads to an increase in the degree of silicon extraction into the solution. A further increase in the sodium hydroxide flow rate has practically no effect on the transition of silicon into solution.

Studies of the effect of solid to liquid ratio on the efficiency of the leaching process of electrowinning ilmenite concentrate dust have shown that the ratio of 1:5 is optimum.

Table 3 - Effect of S:L ratio on the recovery of silicon, chromium, zinc, manganese, and iron in solution, %.

S:L	Cake yield, %	SiO_2	Cr_2O_3	ZnO	MnO	Fe_2O_3
1:3	80.0	65.4	22.8	16.7	13.0	13.5
1:5	61.66	77.7	31.3	35.7	34.5	34.9
1:8	56.3	81.1	41.0	37.0	35.3	35.7
1:10	54.7	82.1	42.4	37.7	35.1	35.4

Therefore, optimal conditions of leaching of electro-smelting dust of ilmenite concentrate were determined experimentally: temperature 80-90 °C, duration 90-120 min, ratio S:L = 1:5, sodium hydroxide solution concentration 110-115 g/dm^3. The residue from dust leaching with the content of 46 % TiO_2, 26.4% Fe_2O_3, 3.6 %SiO_2 is returned to the technological process.

The experiments on fluorination of dust from electric smelting of ilmenite concentrate were performed with the help of specially made installation that included argon cylinder or air, manometer, flowmeter, horizontal tubular furnace, refrigerator-condenser, a gas-sink system consisting of two flasks filled with 10 % ammonia-water solution.

As a result of the works the optimum conditions for fluorination of electric melting dust of ilmenite concentrate were determined: the temperature 260 °C, the duration 6 hours, the mass ratio of dust to ammonium hydrodifluoride was 1:1. Under these conditions, the sublimation degree of silicon fluoride was 84.2 %.

According to X-ray phase analysis, the silicon-containing sublime is represented by oxonium hexafluorosilicate and to a small extent by ammonium hexafluorosilicate (Table 4).

Table 4 - Phase analysis of silica-containing substrate (260 °C, 6 h, dust: NH_4HF_2 = 1:0.9)

The component	Formula	Content in the sample, %
Ammonium hexafluorosilicate	$(NH_4)_2SiF_6$	1.9
Oxonium hexafluorosilicate	$(H_3O)_2SiF_6$	98.1

Ammonium and oxonium hexafluorosilicates are highly soluble in water at room temperature. To precipitate silicon oxide it is necessary to act with an alkali, e.g. ammonia by reactions:

$$(NH_4)SiF_6 + 4NH_4OH = SiO_2 + 6NH_4F + 2H_2O \quad (1)$$

$$(H_3O)_2SiF_6 + 6NH_4OH = SiO_2 + 6NH_4F + 6H_2O \quad (2)$$

Precipitation of amorphous silicon oxide according to thermodynamic calculations should be performed in the temperature range of 25 - 100 °C [16]. In a solution containing hexafluorosilicate ion heated to 40 °C 10 % and in the second case 25 % ammonia solution up to pH 7.5 - 8 were injected under active stirring in portions in the first case.

During the investigation of influence of ammonia solution concentration on amorphous silicon oxide precipitation efficiency, it was noticed that the preset pH value is reached in 20-30 minutes. However, the formation and precipitation of silicon oxide flakes require suspension soaking.

With 25 % ammonia solution- about 80 min; and with 10 % ammonia solution -90 min (Figure 8). The composition of the precipitated amorphous product is shown in Table 5.

Table 5 - Contents of main components and impurities in the precipitated amorphous product, wt %

SiO_2	NH_4F	Fe	Cu	Zn	As	Sr	Pb	other
81.6	12.9	0.045	0.005	0.025	0.014	0.003	0.017	5.4

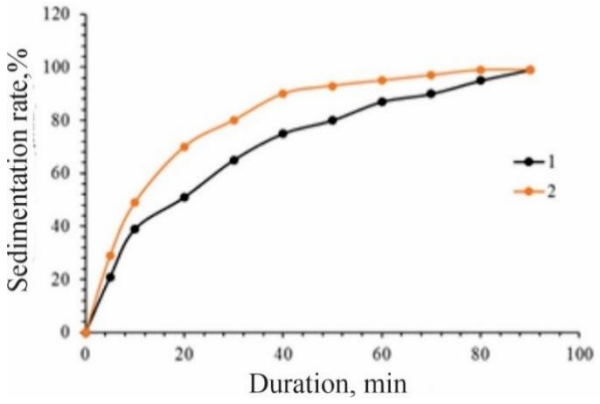

Figure 8 - Dependence of amorphous silicon oxide precipitation

Table 5 shows that the product does not contain any heavy metals or arsenic. The presence of ammonium fluoride is due to its absorption by amorphous particles and cannot be removed by washing the sludge with water. Ammonium fluoride is known to decompose on heating. In this connection, a thermal analysis of the obtained amorphous silicon oxide was performed. The result is presented in Figure 9. The combination of

exothermic effects with peaks at 310.9 °C and 1060.7 °C on the dDTA curve is an indication of amorphous silica. The combination of endothermic effect with extremum at 126.2 °C and exothermic effect with a peak at 1233.1 °C on the dDTA curve characterizes the melting of FeF_2 impurity. The endothermic effect with the extremum at 1144 °C on the DTA curve shows the release of previously adsorbed gases. The weak endothermic effect with the extremum at 1292.3 °C on the DTA curve reflects the sublimation of aluminum fluoride impurity.

The analysis of the DTG curve showed that at 526.4 °C the weight loss of the sample increased due to hydrogen fluoride removal. Therefore, the temperature of 530-560 °C is adopted for the pyrolysis of the product, the composition of which is shown in Table 3. The effect of duration was studied in the range of 20-80 min. The results are shown in Figure 10.

The curve of Figure 10 shows that the pyrolysis process duration of 60-80 min provides the purification of silicon oxide from fluorine at 95-99 %.

The composition of the amorphous silica obtained is shown in Table 6.

on process duration (1 – 10 % NH_4OH; 2 – 25 % NH_4OH).

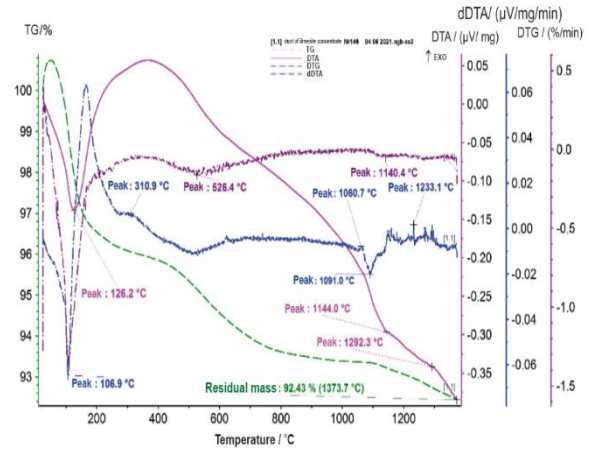

Figure 9 - Derivatogram of amorphous silicon oxide (sample weight 0.088 g)

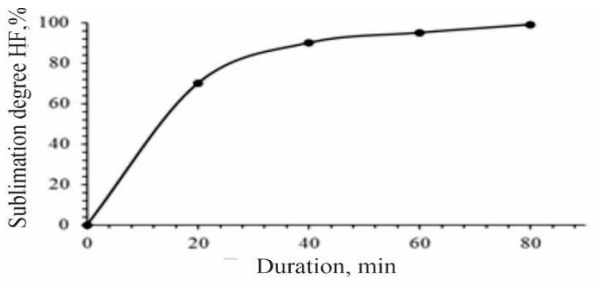

Figure 10 - Effect of pyrolysis duration on the degree of hydrogen fluoride sublimation

Table 6 - Contents of main components and impurities in amorphous silica, wt.% (converted to oxides)

SiO_2	F	Fe_2O_3	Al_2O_3	ZnO	CaO	TiO_2
96.3	ND	0.14	0.16	0.02	0.03	0.15

X-ray phase analysis of the obtained product showed amorphous silicon dioxide monophase.

Therefore, the optimum pyrolysis regime for the separation of fluoride and silicon oxide should be considered as 530-560 °C and a duration of 60-80 min. The conducted studies have shown the possibility of obtaining commodity amorphous silica from waste dust of electric smelting of ilmenite concentrate.

After alkaline hydrolysis of the residue in the boat was obtained titanium-containing product composition, wt. %: 42.5 TiO_2, 26.0 Fe_2O_3, 2.4 MnO_2, 1.3 SiO_2, 0.9 Al_2O_3, 0.2 K_2O, 1.2 ZnO, 0.014 ZrO_2, 0.9 PbO. The product can be returned to electromelting together with ilmenite concentrate.

Conclusions

Physical and chemical study of electric smelting dust of ilmenite concentrate has shown that a part of titanium is bound in hard-to-recover anasovite that can be enclosed in the shell of amorphous silicon oxide. The finely dispersed state of the dust should contribute to the efficiency of the leaching of the harmful impurity - silicon.

Optimal parameters of sodium alkali leaching of electric melting dust of ilmenite concentrate have been determined: NaOH concentration - 110-115 g/dm^3; S:L- 1:5; temperature - 80-90 °C; duration - 90-120 min. The degree of extraction of silicon in the solution was 77,7 %. The titanium-containing product obtained after alkaline leaching of dust contained 48 % TiO_2, 26 % Fe_2O_3 that can be returned into the technological process.

The process of silicon sublimation from fine dusts of electro-smelting of ilmenite concentrates was investigated. The optimum conditions of fluorination of smelting dust of ilmenite concentrate were determined: temperature 260 °C, duration 6 hours, the mass ratio of dust to ammonium bifluoride = 1:1. Under these conditions, the degree of silicon fluoride sublimation was 84.2 %.

The conditions of amorphous silica precipitation from the process duration at 25 % ammonia concentration were studied. Amorphous product with 81.6 % SiO_2, 12.9 % NH_4F was obtained. Optimum pyrolysis conditions that provide the

separation of fluoride and silicon oxide: temperature 530-560 °C and duration 60-80 min have been determined. The content of silicon oxide in the product obtained was 96.3 %.

After alkaline hydrolysis of the cinder, a titanium-containing product containing 48 % TiO_2, 26,0 Fe_2O_3 was obtained that is returned to the technological process.

Conflict of interests. On behalf of all authors, the correspondent author declares that there is no conflict of interests.

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Ильменит концентратын пирометаллургиялық өндеуде түзілген қалдықтардан кремний диоксидін алу әдістері

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ТҮЙІНДЕМЕ

Мақалада ильменит концентраттарын электрлік балқыту кезінде түзілген қалдық шаңды өңдеп, одан кремнийді сілтілі және фторидті әдістермен алу бойынша зерттеулер көрсетілген. Электробалқыту шаңын натрий гидроксиді ерітінділерімен шаймалау, натрий гидроксиді концентрациясының әсерін, процестің ұзақтығын, температураны және Қ:С қатынасын зерттеуді қамтиды. Ильменит концентратын электробалқыту кезіндегі түзілген шаңды шаймалаудың оңтайлы шарттары белгіленді: температура 80–90 °С, ұзақтығы 90–120 мин, қатынасы Қ:С = 1:5, натрий гидроксиді ерітіндісінің концентрациясы 110–115 г/дм^3. Электробалқыту шаңын фторлаудың оңтайлы шарттары анықталды, бұл жағдайда кремний фторидінің сублимациялану дәрежесі 84,2% құрады. Алынған кремнийі бар возгонды аммиак агентінің қатысуымен ыдырату бойынша зерттеулер жүргізілді. Термиялық талдау және процестің ұзақтығының әсерін зерттеу нәтижелері бойынша фторид пен кремний оксидінің бөлінуін қамтамасыз ететін пиролиздің оңтайлы режимдері белгіленді: температура 530–560 °С және ұзақтық 60–80 мин. Алынған өнімдегі кремний оксидінің мөлшері 96,3% құрады.

Түйін сөздер: ұсақдисперсті шаңдар, шаймалау, натрий гидроксиді, кремний диоксиді, фторлау.

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АННОТАЦИЯ

В статье представлены исследования по переработке отвальных пылей электроплавки ильменитовых концентратов с удалением из них кремния щелочным и фторидным методами. В исследование выщелачивания пыли электроплавки растворами едкого натра входило изучение влияния концентрации гидроксида натрия, продолжительности процесса, температуры, соотношения Т:Ж. Установлены оптимальные условия выщелачивания пыли электроплавки ильменитового концентрата: температура 80-90 °С, продолжительность 90-120 мин, соотношение Т:Ж = 1:5, концентрация раствора гидроксида натрия 110-115 г/дм^3. Определены оптимальные условия фторирования пыли электроплавки, при которых степень возгонки фторида кремния составила 84,2 %. Проведены исследования по разложению полученного кремнийсодержащего возгона в присутствии аммиачного агента. На основе результатов термического анализа и исследований по влиянию продолжительности процесса, установлены оптимальные режимы пиролиза, обеспечивающего разделение фторида и оксида кремния: температура 530-560 °С и продолжительность 60-80 мин. Содержание оксида кремния в полученном продукте составило 96,3 %.

Ключевые слова: тонкодисперсные пыли, выщелачивание, гидроксид натрия, диоксид кремния, фторирование.

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