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Деятельность научно-практического журнала «Горные науки и технологии» (Mining Science and Technology (Russia)) направлена на развитие международного научного и профессионального сотрудничества в области горного дела.

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BENEFICIATION AND PROCESSING OF NATURAL AND TECHNOGENIC RAW MATERIALS

Research article

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Prospects for selective-and-advanced recovery of rhenium from pregnant solutions of in-situ leaching of uranium ores at Dobrovolnoye deposit

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Abstract

Analysis of exploration materials and market conditions showed that by-product recovery of rhenium, one of the rarest strategic elements of the periodic system, was not always effective in processing the whole volume of pregnant uranium-bearing solutions. The main goal of the research was to develop an effective method for recovery rhenium from pregnant solutions in in-situ uranium leaching. The objectives of the research were as follows: evaluation of the possibility of selective-and-advanced recovery of rhenium from ores by in-situ leaching method and comparison of the technological advantages of the new proposed method with the known ones. The study involved the analysis of historical geological, mineralogical and geochemical information on the Dobrovolnoye deposit and analysis of technological aspects of by-product recovery of rhenium in the world practice. A selective-and-advanced scheme of rhenium recovery from pregnant uranium-bearing sulfate (sulfuric acid) solutions of the Dobrovolnoye deposit ISL (Russia) using mobile installations was proposed. The process has the following features: zoning of production blocks when constructing injection and extraction (pumping) wells; piping of selective extraction wells into a separate collecting pipe; implementation of advanced rhenium sorption. The process implementation makes it possible to obtain rhenium from economically viable areas of the uranium deposit. The mobile installation includes the following main units: a filter for purification (aftertreatment) to remove suspension, a chain of sorption apparatuses (sorption filters or columns), connecting fittings, control and measuring instruments. The sorption apparatuses are filled with rhenium-selective ionite (ion exchanger). As a selective sorbent for the primary concentration of rhenium from sulfate solutions (pH 2), weakly basic nitrogen-bearing ionites containing amine functional groups of various types can be used. If further concentration of rhenium is required, in order to unify the equipment used, materials with a mobile extractant phase (so-called TVEXs (solid extractants or Levextrel resins in English literature) and so-called "impregnated" or "impregnates"), such as TVEX-DIDA containing diisododecyl amine, or TAA-impregnate containing trialkylamine, can be used. Rhenium desorption from these materials is carried out by an ammonia solution, which allows producing rough ammonium perrhenate from the eluate. Economic aspects of the rhenium selective-and-advanced technology were evaluated. Implementation of the recovery selective-and-advanced technology allows obtaining rhenium from economically-viable areas of the uranium deposit.

Keywords

uranium, in-situ leaching, Kurgan region, rhenium, sulfuric acid, recovery scheme, production block, differentiation, optimization, productivity, selectivity

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ОБОГАЩЕНИЕ, ПЕРЕРАБОТКА МИНЕРАЛЬНОГО И ТЕХНОГЕННОГО СЫРЬЯ

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Аннотация

Анализ геологоразведочных материалов и рыночной конъюнктуры показал, что попутное извлечение одного из самых редких стратегических элементов периодической системы – рения – не всегда эффективно при переработке всего объема продуктивных урансодержащих растворов. Основная цель исследований – разработка эффективного способа извлечения рения из продуктивных растворов при скважинном подземном выщелачивании урана. Задачами исследований являлись: оценка возможности селективно-опережающего извлечения рения из руд способом скважинного подземного выщелачивания и сопоставление технологических преимуществ нового предлагаемого способа с известными. В работе использован анализ геологической, минералого-геохимической информации предыдущих лет изучения месторождения Добровольное и анализ технологических аспектов попутной добычи рения в мировой практике. Предложена селективно-опережающая схема извлечения рения из продуктивных сернокислых урановых растворов подземного выщелачивания урановых руд месторождения Добровольное (Курганская область, Россия) с использованием мобильных установок. Технология имеет следующие отличия: зонирование эксплуатационных блоков при сооружении закачных и откачных скважин; обвязку откачных селективных скважин в отдельный коллектор; осуществление опережающей сорбции рения. Ее осуществление дает возможность получать рений из экономически выгодных участков месторождения урана. Мобильная установка включает следующие основные узлы: фильтр для очистки (доочистки) от взвесей, каскад сорбционных аппаратов (сорбционных фильтров или колонн), соединительную арматуру, контрольно-измерительные приборы. Сорбционные аппараты заполняются селективным на рений ионитом. В качестве селективного сорбента для первичного концентрирования рения из сернокислых растворов (pH 2) могут быть использованы слабоосновные азотсодержащие иониты, содержащие функциональные группы аминов различного типа. При необходимости дальнейшего концентрирования рения с целью унификации используемого оборудования можно применить материалы с подвижной фазой экстрагента (твэксы или импрегнаты), например, ТВЭКС-ДИДА, содержащий диизододециламин, или импрегнат-ТАА, содержащий триалкиламин. Десорбция рения с этих материалов осуществляется раствором аммиака, что позволяет получить из элюата черновой перренат аммония. Оценены экономические аспекты селективно-опережающей технологии рения. Реализация технологии селективно-опережающего извлечения дает возможность получать рений из экономически выгодных участков месторождения урана.

Ключевые слова

уран, скважинное выщелачивание, Курганская область, рений, серная кислота, схема извлечения, добычной блок, дифференциация, оптимизация, продуктивность, селективность

Для цитирования

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Introduction

The main direction in the development strategy of the mineral resource base (MRB) of the Russian Federation uranium mining industry for the next decade is primarily expansion of MRB in the area of operating mining enterprises, JSC "Dalur", JSC "Khiagda" and PJSC "PPGKhO", suitable for the development with the use of two geotechnologies: well in-situ leaching (ISL) and in-situ block leaching [1].

Creation of new modern technologies for natural resources development involves application of environmentally sound methods, which include in-situ leaching, their integrated development and the maximum possible degree of processing for the products demanded by the national economy. Integrated processing of natural resources as a basis for creation of low-waste technologies is understood as the use of all mineral components of raw materials by turning them into useful products by combining several productions within one enterprise. Often, when extracting two or more components from raw materials, the lower concentration limit of processing viability decreases.

Along with the main task of uranium raw materials integrated processing – by-product recovery of its valuable components, which are of an economic nature, utilization of solid waste and recovery of reagents from water and gas waste are also important. Uranium itself is recovered as by-product from copper and gold ores, as well as from phosphorite. The share of such by-product uranium is ~9 % of the whole uranium recovery.

Complex uranium ores are divided into two main groups [2]:

1. Ores in which uranium and associated elements are contained in a single mineral (vanadium and uranium in carnotite, phosphorus and uranium in apatite, niobium and uranium in hatchettolite, zirconium and uranium in malacone). The useful components cannot be separated by mechanical beneficiation methods and can only be separated by chemical processing of the ores.

2. Ores in which uranium and associated elements are represented by different mineral forms or carriers (ores containing non-ferrous metal sulfides and arsenides, native silver, gold, bismuth). In this case, separate production of concentrates is possible by methods of mechanical beneficiation.

The following elements accompany uranium in ores: gold, vanadium, molybdenum, rare earth elements, yttrium, rhenium, copper, nickel, cobalt, arsenic, and phosphorus. Thorium, scandium, and selenium are present to a lesser extent. It should be noted that grade (concentration) of these elements in the ores is often too low to ensure economically viable separation and recovery, but on the other hand, it is high enough to affect the quality of the primary uranium concentrate, "yellow cake". In this connection, the task of integrated processing of uranium raw materials is also connected with the task of improving the quality of the main commercial product of uranium processing.

Profitability of integrated processing is determined by the lower concentration limit, evaluation of which for two-component ores at different uranium grade is given in [3]. It should be noted that the most cost-effective in the world uranium recovery from ores with an average uranium grade of 0.06% was achieved in the course of processing of complex ores of Olympic Dam copper-gold-silver-bearing uranium deposit in Australia.

The most promising method of uranium ore leaching, in-situ leaching (ISL) (with the use of wells), produces so-called pregnant solutions, which along with the target component, uranium, contain a number of by-products, among which rhenium and scandium are of high value and strategic importance. The lower concentration threshold of rhenium in them is less than 0.02 mg/l [3].

Rhenium is one of the least abundant elements of the periodic system. The existence of rhenium was predicted in 1871 by D.I. Mendeleev, who called it eco-manganese. It was discovered in Germany in 1925 as the last stable element in the periodic system. The melting point of this metal is 3180 °C, second only to tungsten. At the same time, rhenium is distinguished by plasticity among the most refractory brittle metals (tungsten, molybdenum, and others). Alloys based on refractory metals have high mechanical strength as well as resistance to thermal deformation, creep, and oxidation. Superalloys – alloys based on metals of group VIII (nickel, iron, and cobalt) are distinguished among them [4]. The most high-temperature cast complex-alloyed nickel-based alloys capable of operating at temperatures of 1050–1100 °C for hundreds and thousands of hours under high static and dynamic loads typically account for 40–50 % of the total weight of an aircraft engine and are widely used in working and nozzle blades, turbine rotor disks, combustion chamber parts [5, 6]. These superalloys also contain other elements such as rhenium, chromium, cobalt, tungsten, tantalum, molybdenum, hafnium, titanium, aluminum. Up to 40% of the total weight of the superalloys belong to them. Alloys of the fifth generation contain ruthenium along with rhenium [7]. Rhenium is one of the most effective alloying components in monocrystalline alloys: its effect on the heat resistance enhancement is due to its increased solubility in nickel γ-solid solution, the increase in the lattice spacing and solidus temperature, the decrease in the alloying elements diffusion coefficients [7].

The development of aerospace engineering in the world is determined by the use of rhenium heat-resistant monocrystalline nickel-based superalloys. The sector of heat-resistant alloys consumes more than 80% of total rhenium production (~70 t, 2021)[1]. This field of rhenium application has dominated since the end of the last century [4]. The remaining part of rhenium is indispensable in the composition of platinum-rhenium catalysts for high quality gasoline and in electronics [4].

The rhenium mineral raw -material base of Russia is presented by low-grade molybdenite concentrates, which can be produced from ores of deposits Sorskoye, Zhirekeskoye, Lobash, etc., and also by fumarole gases of Kudryavy volcano (Iturup island, Kurils, Sakhalin region), included in the State Balance Sheet of Reserves, but not processed.

By-product recovery of rhenium from pregnant solutions of Dobrovolnoye uranium deposit ore ISL (in the area of the operating mining enterprise JSC “Dalur”) seems to be a timely scientific and practical task.

The main goal of the research was to develop an effective method for recovery rhenium from pregnant solutions in in-situ uranium leaching.

The objectives of the research were as follows: evaluation of the possibility of selective and advanced recovery of rhenium from ores by in-situ leaching method and comparison of the technological advantages of the new proposed method with the known ones.

The study involved the analysis of historical geological, mineralogical and geochemical information on the Dobrovolnoye deposit and analysis of technological aspects of by-product recovery of rhenium in the world practice.

Brief analysis of by-product rhenium recovery technologies

Because of ultralow Re crystal abundance ($7 \cdot 10^{-4}$ %), rhenium in world practice is recovered only as a by-product in integrated processing of molybdenum and copper ores. Molybdenum concentrates from copper-porphyry deposits, which provide about 80 % of world production of rhenium (Chile, Peru, USA and Canada), serve as the main raw material for rhenium production. Among other sources the important place belongs to stratiform deposits of copper, from which the main amount of rhenium is extracted in CIS countries. Cuprous ores of the Dzhezkazgan deposit in Kazakhstan remain the largest source of this metal in Asia. Own minerals of rhenium are rare (dzhezkazganite – rhenium sulfide, rheniite – rhenium sulfide) and have no practical value [4].

Uranium ores as an additional raw material source of rhenium are known since 50th of the last century: in the USA, ~1 t of rhenium was produced from molybdenum-uranium ores [4]. In the CIS, stratified-infiltration exogenous deposits developed by in-situ leaching are located in the largest in the world Near-Tien-Shan Uranium-Ore Province. The grade of rhenium in the ores of these deposits ranges from 0.02 to 2 g/t [8].The highest grade of rhenium was found in Central Kyzyl Kum in ores of “uchkuduk type” (0.6-2.0 g/t on average), whereas in adjacent Syr-Darya and Chu-Sarysu provinces the grade was much less (0.2-0.5 g/t Re) [9]. Rhenium was first detected in 1978 at the North Kanimekh deposit in solutions of in-situ uranium leaching. It was found to be most effectively extracted by ISL with the use of oxygenated natural water. The universal form of Re migration in oxygenated groundwater is the perrhenate ion ReO_4^-, the concentration of which reaches as much as milligrams per liter, whereas its common concentration values are about 0.0n µg/L [8].

In Russia, Re content in uranium ore occurrences of the Russian platform was evaluated [10]. The content of rhenium in these objects is insignificant.

Process flow diagrams based, as a rule, on the use of sorption method were developed for recovery of minor valuable elements from pregnant solutions of in-situ leaching [11]. For example, vanadium can be extracted by using VPK chelating ionite (ion-exchanger) and VP-1p anionite (anion exchanger) for its concentration; for scandium recovery, AFI-22 ion-exchanger is applied.

It should be noted that the highest profitability, 540 %, was demonstrated by rhenium recovery [12]. From sulfate pregnant solutions, rhenium together with uranium is absorbed by strong-basic anion exchanger AM-p. Selective desorption of rhenium is carried out with acidic solutions containing nitrate ions (NO_3^- – 80–90 g/L, HNO_3 – 4–4,5 %). Rhenium is extracted from eluates (with Re concentration of 10–15 mg/L) by a solution of trialkylamine fraction C7–C9 in kerosene with the addition of decanol as modifier. The content of rhenium in the extract reaches 8 g/L, which allows its re-extraction with ammonia solution in the solid-phase version. Flow diagram of by-product recovery of rhenium from ISL sulfate solutions is presented in Fig. 1 [11].

To extract rhenium from circulating solutions of in-situ leaching after uranium sorption (The Republic of Uzbekistan), the composite extracting polymer KEP-200 by Purolite company was used [12, 13]. Recent studies of rhenium sorption from model sulfate (pH 1.54) and bicarbonate (pH 7.1) solutions demonstrated the efficiency of strong-basic anion exchanger V0-020 [15]. From 2008 to 2011, at two operating sorption installations of Navoi Mining and Metallurgical Combine, ~3 tons of ammonium perrhenate of AP-0 grade were produced [15, 16].

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[1] USGS Mineral Commodity Yearbook 2021. https://doi.org/10.3133/mcs2021

In the territory of the Russian Federation, rhenium was not extracted from solutions of uranium in-situ leaching on commercial scale.

Characteristics of the developing Dobrovolnoye deposit

Development of deposits by the ISL method in the Trans-Urals is conducted by JSC Dalur uranium mining company, located in the village of Uksyanskoye, Dalmatovskiy district of Kurgan region. The district is well developed and is crossed by the Sverdlovsk-Kurgan-Petropavlovsk and Chelyabinsk-Kurgan railroads, as well as dense network of regional and national roads. The routes of the main oil pipelines pass through the district.

Uranium mineralization in the Trans-Urals territory (Fig. 2) is mainly presented by uranium deposits and occurrences in the Upper Jurassic - Lower Cretaceous paleo-valleys, cut into the crystalline rocks of the pre-Jurassic basement. The currently developed Miass and Uysko-Tobolsky ore districts, which include three deposits: Dalmatovskoye, Khokhlovskoye, Dobrovolnoye and a large group of ore occurrences, are of commercial importance.

Paleo-valleys extend for tens of kilometers at the width of 2–5 km. The deposits are elongated (up to 20 km long) ribbon-like zones of uranium-bearing rocks in paleo-valleys. The source of uranium and associated metals is probably the rocks of the Jurassic basement, which underwent intense chemical weathering.

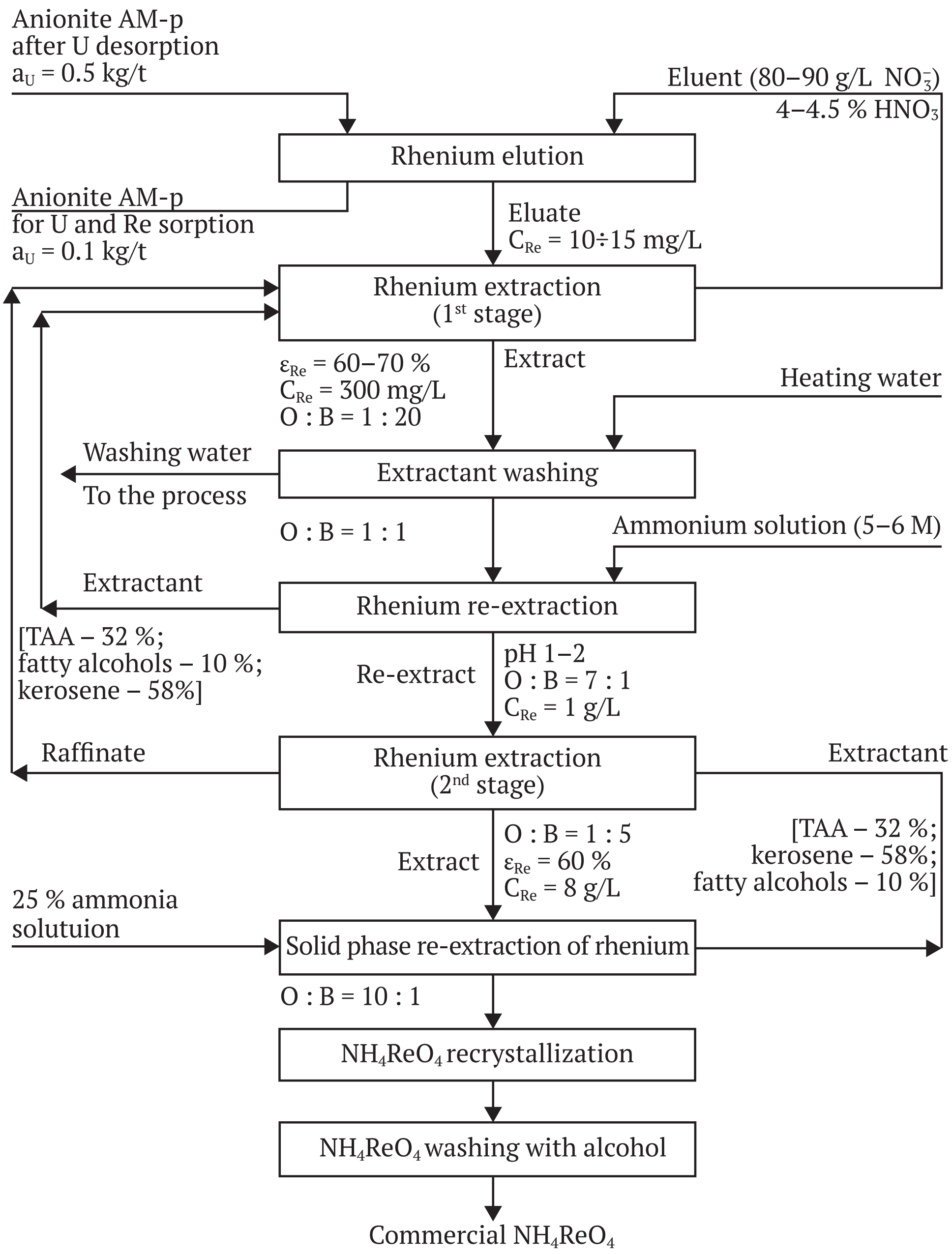

Fig. 1. Flow diagram of rhenium by-product recovery from uranium in-situ leaching solutions [11]

Gravel-sandy sediments rhythmically alternating in the section well permeable with clayey aquicludes allow distinguishing three water-permeable sub-horizons (aquifers) in the section. In the absence of an aquiclude between sediments of the first and second rhythms, the lower and middle aquifers are combined into one. The upper aquifer lies at a depth of 435–517 m. The average thickness is 13 m. The upper aquiclude for the aquifer is presented by clays of Koskolskaya red-colored strata, and the bottom aquiclude is composed of clayey sediments 3–15 m thick. The filtration coefficient varies from 0.8 to 23.5 m/day. The aquifers occur at depths of 513–567 m. Their average thickness is 30 m, varying from 4.5 to 44 m. The upper aquiclude is presented by clayey sediments of the second rhythm of 3–20 m thick, and the lower one is the basement rocks. The filtration coefficient varies from 5.0 to 20.7 m/day.

Chemistry of the Middle-Upper Jurassic ore-bearing aquifer system within the deposit is characterized by prevailing hydrocarbonates, sulfates, and sodium chlorides. The waters are saline with salinity of 11.1–13.0 g/dm^3, pH 7.2–7.5. Uranium concentration in the water is up to $7.8 \cdot 10^{-6}$ g/dm^3, hydrogen sulfide concentration is up to 1.5 mg/dm^3, Eh ranges from +60 to +120 mV. The aquifer waters are high-pressure. Estimated water head above the day surface is 41–47 m.

Mineral composition of uranium ores of Trans-Urals deposits is uniform and typical for exogenous deposits of hydrogenous type. Ore-bearing sandy sediments up to 20 m thick have aluminosilicate composition, %: SiO_2, 78.5; Al_2O_3, 12.5; CO_2, 0.2 (with maximum content not exceeding 2.0); iron, 1–3; sulfide sulfur, 0.1–1.6. More than 95 % of uranium occur in its own mineral form of uranium oxides (nasturan), represented by several generations, and silicates (coffinite). The ratio of nasturan and coffinite in the ores is approximately 1:1. The radioactive equilibrium coefficient is ~1. In uranium grade, the ores belong to the class of poor and low-grade; the average uranium grade in ores of the deposits is 0.03–0.05 %. The age of mineralization determined by the uranium-lead method for all the deposits is about 140 million years.

Technological properties of the ores of the deposits are identical; the ores are suitable for uranium leaching with sulfate (sulfuric acid) solutions.

Rhenium-rare earth-uranium deposit Dobrovolnoye was discovered in 1986 by Turgai Crew No. 89. The State Balance Sheet of Reserves accounted for in the deposit: 339.0 tons of uranium in C1 category; 7060.0 tons of uranium in C2 category; 5656 tons of uranium in P1 category. Depth of occurrence of the uranium ores is 480–700 meters. Average uranium grade in ore bodies of the deposit is 0.028–0.057 %. Resources and reserves of rhenium are 5.59 tons in P1 category and 10.6 tons in C2 category[2].

Fig. 2. Dobrovolnoye deposit location map

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[2] Luchinin I. L. Report on the results of preliminary exploration of rhenium-rare earth-uranium deposit Dobrovolnoye. Turgai Party No. 89. Yekaterinburg; 1994.

Rhenium, except for its own mineral form (rhenium sulfides), was identified in the form of sorption impurity in nasturan, pyrite, marcasite and clay matter.

The contents of rhenium in the ores by grade classes are as follows: up to 1 g/t – 71 %, from 1 to 10 g/t – 29 %. High grades of the rhenium are confined to subzones of poor “gray” uranium ores and their “rich” subzones; 15–20 % of the rhenium ores are located in subzone of “whitish” rocks. A schematic section and distribution of rhenium along the exploration section (profile) are shown in Fig. 3.

Selective-and-advanced recovery of rhenium from pregnant sulfate solutions of in-situ leaching

Currently, multivariate methods of recovery of minerals by in-situ leaching, particularly for uranium, are known. At the same time, only rare elements – rhenium and scandium – are so far extracted from pregnant solutions after uranium recovery. Rhenium is extracted on commercial scale in Uzbekistan at the Navoi Mining and Metallurgical Combine [13]. Other methods described in the patents of Russia[3], Kazakhstan[4] and others, as well as in papers [13, 16–19] and monographs [11], prescribe the recovery of rhenium from the whole volume of uranium-bearing solutions.

The main drawback of the known methods is that recovery of rhenium and other associated components from pregnant uranium solutions is supposed to be carried out after sorption of uranium from the pregnant solution – from the so-called “sorption mother liquors”. In this case it is necessary to process equivalent volumes of solutions with low concentrations of the associated components in the total volume of the solutions. For example, the concentration of rhenium in pregnant uranium solutions is 0.3–0.8 mg/L (0.5 mg/L in average), which predetermines the necessity to construct additional modules for processing of the solutions, commensurable with the core uranium production. The capital and operating costs of such production can withstand the profitability of the by-product recovery only when peak market prices for the metals are reached. This makes by-product recovery of rhenium and other rare elements under such process flow sheets vulnerable.

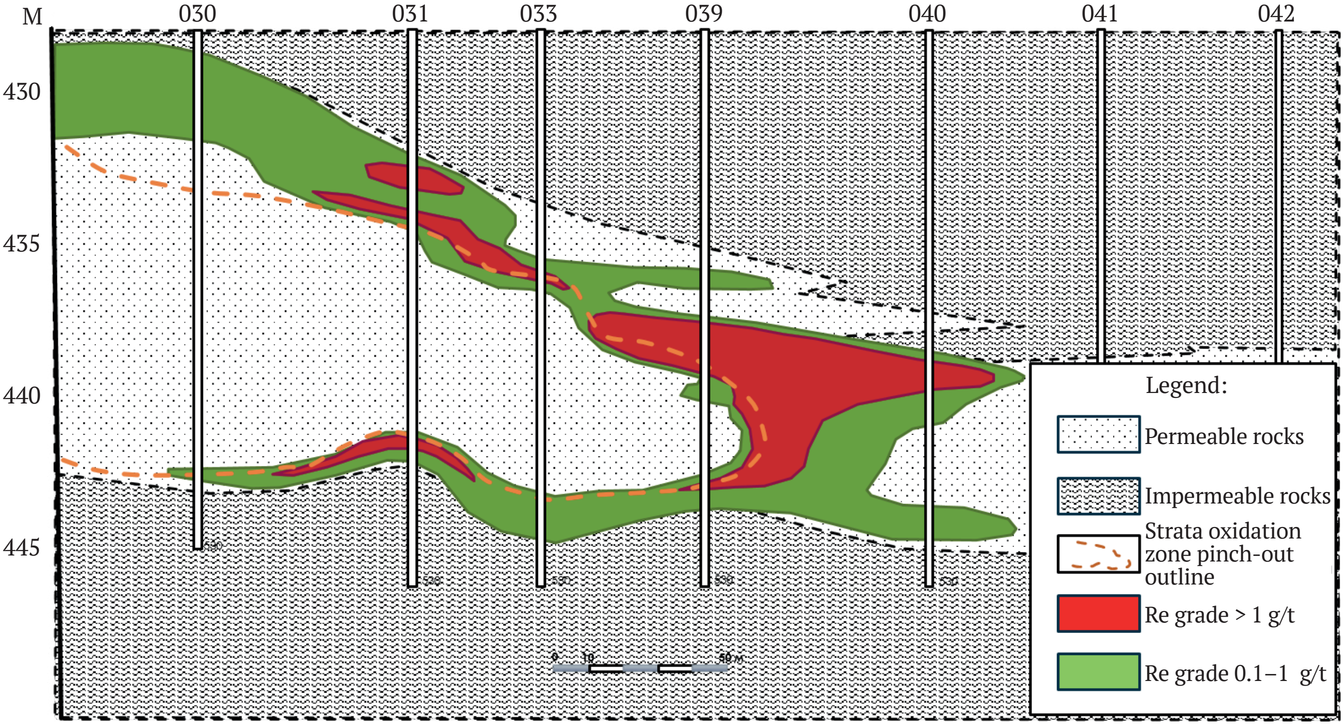

Fig. 3. Distribution of rhenium along the profile 65+400 at Dobrovolnoye deposit

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[3] Tolstov E. A., Mikhin O. A., Pershin M. E. et al. Method for rhenium recovery from solutions. RF Patent No. 2294392 dated 24.05.2005.

[4] Kozhakhmetov S. K., Kopbaeva M. P., Panova E. N. et al. Method of processing pregnant solutions of in-situ leaching of rhenium-containing uranium ores. RK Patent No. 22659 dated 02.04.2012; Kozhakhmetov S. K., Kopbaeva M. P., Panova E. N. et al. Method of processing pregnant solutions of in-situ leaching of rhenium-containing uranium ores. RK Patent No. 23507 dated 02.04.2012; Kozhakhmetov S. K., Kopbaeva M. P., Panova E. N. et al. Method of processing pregnant solutions of in-situ leaching of rhenium-containing uranium ores. RK Patent No. 26780 dated 02.04.2012.

It is possible to calculate preliminary average concentration of rhenium (C_m, g/dm^3) in pregnant solution during the whole leaching period on the basis of its extracted quantity and total volume V, m^3, of the pregnant solution using the following formula [20]:

$$C_m = 10^2 P_m \frac{\varepsilon}{V},$$

where P_m – total ore reserves in the section, t; ε – degree of metal recovery, fractions of unit; V – volume of leaching solution, m^3.

At initial grade of rhenium in ores more than 2 g/t, concentration of rhenium in pregnant solution is more than 1 mg/L (Fig. 4).

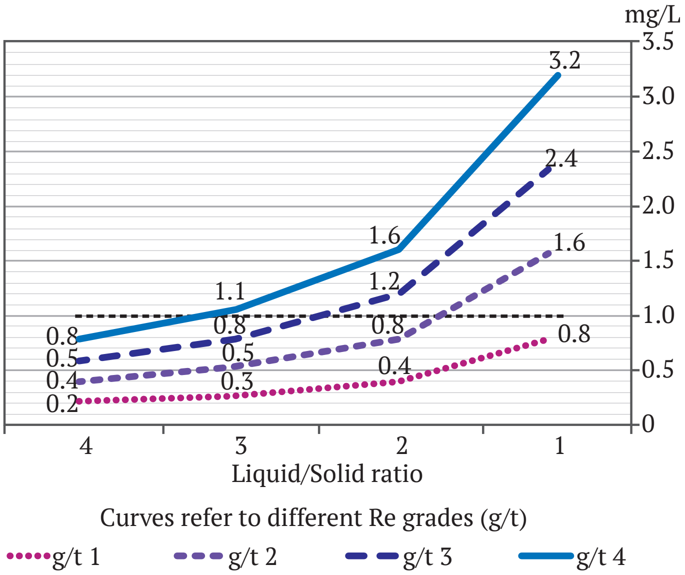

Fig. 4. Rhenium concentration in solution as a function of rhenium ore grade

In this paper we considered the possibility of changing the sequence of operations for by-product recovery of rhenium from solutions by using selective-advanced recovery[5]. An example of the implementation of such a method of recovery is shown in Fig. 5.

Ore production block (*1*) shall be drilled by technological injection (*2*) and pumping (*3*) wells which shall be tested with leaching solution by the "Push-Pull" method after their construction. Based on the results of the analysis of solutions for rhenium, the zones (*4*) of pumping wells location with rhenium concentration in the solution less than 1 mg/l and the zones (*5*) of pumping well location (*6*) with rhenium concentration more than 1 mg/l were differentiated. For the pumping wells with rhenium concentration more than 1 mg/l, piping was created to direct the solutions into a separate pumping collecting pipe (*7*), and solutions were sent for advanced sorption (*9*) with the following production of ammonium perrhenate. After the rhenium sorption, the uranium-containing solutions entered cumulative collecting pipe (*8*) and undergone standard uranium sorption/desorption cycles at the facility for processing uranium-containing solutions (*10*) to produce the finished product ("yellow cake"). The "sorption mother liquor" are supplemented with sulfuric acid and sent through collecting pipe (*11*) for ore leaching through injection wells (2).

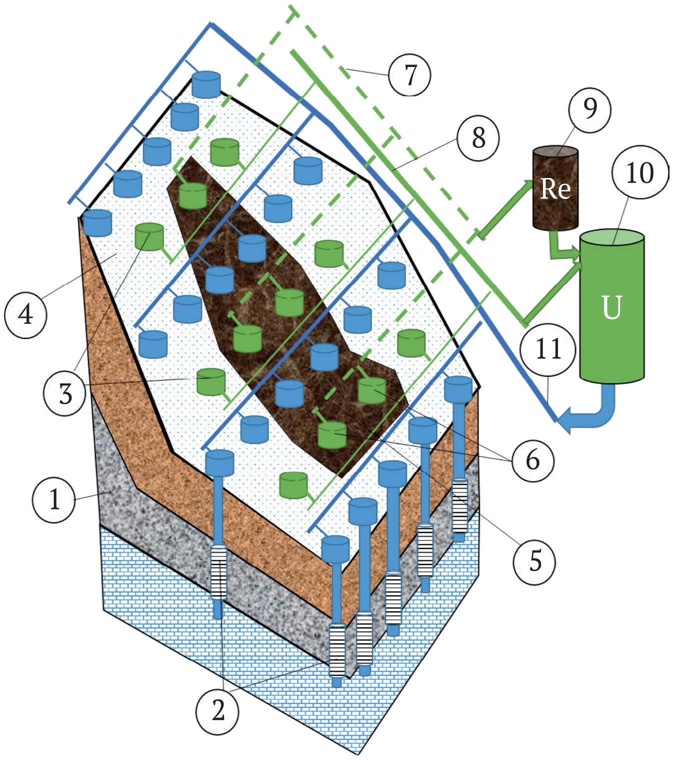

Fig. 5. Schematic diagram of the selective-advanced recovery of rhenium

A variant of a mobile version of the sorption installation for the conditions of in-situ leaching is currently under development.

The installation can provide timely recovery of by-product components through the method of selective-advanced processing of rhenium-containing solutions in the mobile version. The mobile installation includes the following main units: a filter for purification (aftertreatment) to remove suspension, a chain of sorption apparatuses (sorption filters or columns), connecting fittings, control and measuring instruments. The sorption apparatuses are filled with a rhenium-selective ionite (ion exchanger). The installation operates as follows: pregnant solution, containing valuable by-product components, after removal of impurities at the filter or in pond with aftertreatment at the filter, goes to sorption apparatus filled with a ion-exchanger.

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[5] Rudenko A. A., Troshkina I. D., Danileiko V. V. Method of extraction of minerals by in-situ leaching. Positive decision on the application for invention No. 2020129776 (priority from 08.09.2020) dated 28.09.2021

After complete saturation with valuable component the sorption apparatus is switched off and transported to the core shop of uranium solutions processing or to the nearby facility on processing of rhenium-containing materials for washing, desorption and the subsequent concentration of rhenium to produce, for example, rough ammonium perrhenate of AR-1 grade. The sorption apparatus with the regenerated ion exchanger is returned for connection to the installation apparatus cascade.

Depending on the infrastructure of the in-situ leaching wellfield it is possible to carry out washing, desorption and subsequent concentration operations (process stages) to produce the commercial salt in the area occupied by the mobile installation, with increasing the number of necessary apparatuses.

As a selective sorbent for the primary concentration of rhenium from sulfate solutions (pH 2), weakly basic nitrogen-bearing ionites containing amine functional groups of various types, modified activated carbon, fibrous materials of different composition[6] can be used [4, 21, 22–24]. Sorbents based on natural raw materials – lignin [25] and composites with bamboo [26] – were prepared for rhenium recovery from acid solutions.

Interaction of rhenium in solution in the form of perrhenate ion with frequently used in practice weakly basic anion exchangers containing secondary amine groups occurs through the mechanism of ion exchange of perrhenate ion ReO_4^- to sulfate ion (the form in which the anion exchanger is present). The reaction can proceed according to the equation:

$$(RNH_2)SO_4^{2-} + 2ReO_4^- \leftrightarrow 2RNH_2(ReO_4) + SO_4^{2-},$$

where R is hydrocarbon radical.

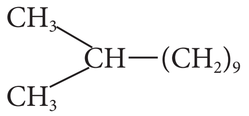
If further concentration of rhenium is required, in order to unify the equipment used, materials with a mobile extractant phase, such as Levextrel resins or impregnates [35], sorption on which allows avoiding fire-hazardous extraction process stage, can be used. The impregnates are produced by impregnating the carrier with the extractant. An alternative way to create materials of this type is the introduction of the extractant into the matrix during its synthesis, to be carried out, for example, in the mode of suspension copolymerization of styrene with divinylbenzene in the presence of a radical polymerization initiator. The materials produced by this method are called solid extractants (“tvex” in Russia; in the English-language literature, these are called Levextrel resins). Despite fundamentally different methods of production, impregnates and Levextrel resins are similar in structure and properties. Unlike ion exchangers, these sorption materials do not swell in aqueous solutions and retain their mechanical strength in both wet and air-dry state that allows their use in modern high-performance equipment.

In terms of density, bulk weight, mechanical strength impregnates and Levextrel resins are similar to sorbents, whereas the mechanism of recovery of components by impregnates and Levextrel resins is similar to the extraction mechanism. The capacitive and kinetic characteristics of these materials are determined mainly by the porous structure, type and state of the extractant in the matrix. A characteristic feature of Levextrel resins and impregnates is the absence of chemical bond between the extractant and the polymer matrix. Due to the liquid phase mobility, these materials have improved kinetic characteristics.

Extractants of different classes are used in their production. TVEX-DIDA containing diisododecyl amine was developed and produced in Research Institute VNIIHT JSC. Interaction of perrhenate ion with this extractant proceeds according to the reaction:

$$(R_2NH_2)_2SO_4 + 2ReO_4^- \leftrightarrow 2R_2NH_2(ReO_4) + SO_4^{2-},$$

where R –



is hydrocarbon radical.

The sorption characteristics of this material and impregnates containing amines, such as trialkylamine (TAA-impregnate), studied at the Mendeleev Russian University of Chemical Technology, allow to use them for concentration of rhenium [28]. Desorption of rhenium from these materials is carried out by ammonia solution, which allows producing rough ammonium perrhenate (from the eluate). Its following purification by electrodialysis method allows improvement of quality of the finished rhenium commercial product to grade AP-0 (GOST 31411-2009 “Ammonium perrhenate. Technical Specifications”. Date of the GOST enactment: 01.01.2011) [4, 14, 18].

The implementation of processes using Levextrel resins or impregnates allows to eliminate the use of toxic, volatile and fire-hazardous organic diluents, avoid difficulties associated with the separation of hard-to-dissolve emulsions (“third phase”), reduces the pollution of waste water with organic reagents – compared with liquid extraction process. The processes with the use of impregnates and Levextrel resins are significantly less energy-intensive than traditional sorption processes due to high degree of metal concentration.

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[6] Troshkina I. D., Balanovsky N. V., Vanin I. A. et al. Method for the recovery of rhenium from uranium solutions. RF Patent No. 2627838 dated 23.03.2016.

Economic aspects of the method of selective-advanced recovery of rhenium from pregnant solutions of uranium in-situ leaching

Economic efficiency of the method of selective-advanced recovery of rhenium is due to the following:

– costs of ore extraction and preparation for processing, etc. are deducted in estimating the cost of rhenium production;

– the impacts on the sorbent designated for extracting the main component, uranium, decreases;

– the quality of the uranium commercial product is improved;

– the volume of rhenium solutions to be processed decreases;

– the capital cost of construction of the processing plant for rhenium is reduced;

– the operating costs and associated expenses decrease;

– the economic attractiveness of the deposit increases.

Conclusions

1. For expanding the mineral raw material base, the method of selective-and-advanced recovery of rhenium, one of the valuable associated components of complex polymetallic uranium ores of the Dobrovolnoye deposit (JSC Dalur, Russia), very important metal of superalloys for the aerospace industry of the national economy was proposed.

2. For the first time the proposed technology has the following features:

– zoning of production blocks when constructing injection and extraction (pumping) wells;

– piping of selective extraction wells into a separate collecting pipe;

– implementation of advanced rhenium sorption.

3. Implementation of the recovery selective-and-advanced technology allows obtaining rhenium from economically-viable areas of the uranium deposit. In this case a mobile facility can be used as the equipment for the technology implementation, the main unit of which is a sorption unit filled with a rhenium-selective ion-exchanger from a group of weakly basic nitrogen-containing ion-exchangers with functional groups of amines of various types. If further concentration of rhenium is required, in order to unify the equipment used, materials with a mobile extractant phase, such as TVEX-DIDA (Levextrel resin) containing diisododecyl amine, or TAA impregnate containing trialkylamine, can be used. Rhenium desorption from these materials is carried out with an ammonia solution, which allows obtaining rough ammonium perrhenate from the eluate.

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BENEFICIATION AND PROCESSING OF NATURAL AND TECHNOGENIC RAW MATERIALS

Research article

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Selection of recycled water electrochemical conditioning parameters for preparation of diamond-bearing kimberlite for froth separation

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Abstract

The main reason of decreasing diamond recovery through froth separation is their surface hydrophilization by hypergene minerals and technogenic films, crystallized from the supersaturated aqueous phase, fixed on the surface structurally or by adhesion. Various types of physical actions, including thermal and ultrasonic treatment of the initial feed of froth separation, are recommended to increase the diamond-bearing kimberlite beneficiation process performance, providing cleaning of the surface of diamonds due to destruction of their accretions with rock minerals and removal of film hydrophilizing coatings from the surface of diamond crystals. A sample of kimberlite material with a given content of diamonds of 1.5–2 mm in size was used as a subject of research in the process of froth separation. The results of thermodynamic calculations and experimental research have substantiated the necessity of using electrochemical conditioning of recycled water for increasing the efficiency of diamond surface cleaning in froth separation operation when using the process of thermal treatment of initial ore feed. The use of diaphragmless electrochemical conditioning of recycled water increases the efficiency of thermochemical dissolution of hydrophilizing compounds on the surface of diamonds through reducing the concentration of calcium and carbonate ions as well as through shifting the medium pH to 6.1–6.5. The measurements of the limiting wetting angle showed that the maximum effect of increasing the diamond surface hydrophobicity was achieved when heat and electrochemical treatment were used together. Laboratory studies showed the possibility of increasing flotation diamond extraction from 65.7 to 91.4 % through application of electrochemical conditioning of recycled water. The optimum parameters of diaphragmless electrochemical treatment of recycled water of the froth separation cycle in conditions of application of pulp heat treatment: current density of 175–200 A/m^2 and power consumption of 1.2–1.5 kWh/m^3. Tests carried out at processing plant No. 3 of the Mirny GOK (Mining and Processing Complex) (Mirny, Yakutiya) showed that the application of the developed froth separation process intensification method with the use of thermal treatment of pulp and electrochemical diaphragmless treatment of recycled water allowed increasing the recovery of diamonds of +0.5–2 mm size by 4.9–5.1 %.

Keywords

diamond, extraction, mining, kimberlite, hydrophilizing compounds, purification, froth separation, recycled water, electrochemical conditioning, heat treatment, Mirny GOK, Yakutiya

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ОБОГАЩЕНИЕ, ПЕРЕРАБОТКА МИНЕРАЛЬНОГО И ТЕХНОГЕННОГО СЫРЬЯ

Научная статья

Выбор параметров электрохимического кондиционирования оборотной воды при подготовке алмазосодержащих кимберлитов к пенной сепарации

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Аннотация

Основной причиной снижения извлечения алмазов методом пенной сепарации является гидрофилизация их поверхности вследствие структурно или адгезионно закрепившихся на ней гипергенных минералов и выкристаллизованных из пересыщенной водной фазы техногенных пленок. Существенное повышение извлечения алмазов в концентрат пенной сепарации может быть достигнуто на основе комбинирования физических и физико-химических методов воздействия на твердую и жидкую фазу рудной пульпы, что определяет актуальность исследований. В качестве предмета исследований в процессе пенной сепарации использована проба кимберлитового материала с заданным содержанием алмазов крупностью 1,5–2 мм. Результатами термодинамических расчетов и экспериментальных исследований обоснована необходимость применения электрохимического кондиционирования оборотной воды для повышения эффективности очистки поверхности алмазов в операции пенной сепарации при применении технологии тепловой обработки исходного рудного питания. Использование бездиафрагменного электрохимического кондиционирования оборотной воды повышает эффективность термохимического растворения гидрофилизирующих образований на поверхности алмазов за счет снижения концентрации ионов кальция и карбоната, а также смещения pH среды до 6,1–6,5. Проведенными измерениями краевого угла смачивания показано, что максимальный эффект увеличения гидрофобности поверхности алмазов достигается при совместном применении тепловой и электрохимической обработки. Лабораторными исследованиями показана возможность повышения извлечения алмазов при флотации с применением электрохимического кондиционирования оборотной воды с 65,7 до 91,4 %. Выбраны оптимальные параметры бездиафрагменной электрохимической обработки оборотной воды цикла пенной сепарации в условиях применения технологии тепловой обработки пульпы: плотность тока 175–200 А/м^2 и расход электроэнергии 1,2–1,5 кВтч/м^3. Проведенными на обогатительной фабрике № 3 Мирнинского ГОКа (г. Мирный, Якутия). испытаниями показано, что применение разработанного режима интенсификации процесса пенной сепарации с применением тепловой обработки пульпы и электрохимической бездиафрагменной обработки оборотной воды позволяет повысить извлечение алмазов из класса +0,5–2 мм на 4,9–5,1 %.

Ключевые слова

алмазы, извлечение, кимберлиты, гидрофилизирующие образования, очистка, пенная сепарация, оборотная вода, электрохимическое кондиционирование, тепловая обработка, обогатительная фабрика, Мирнинский ГОК, Якутия

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Introduction

The main reason of decreasing diamond recovery through froth separation is their surface hydrophilization by hypergene minerals and technogenic films, crystallized from the supersaturated aqueous phase, fixed on the surface structurally or by adhesion [1, 2]. Hydrophilic compounds on diamonds converge the degree of hydrophobicity of their surface and the surface of kimberlite minerals that reduces selectivity of diamond-containing product froth separation process in beneficiation flow sheets [3, 4]. Various types of physical actions, including thermal and ultrasonic treatment of the initial feed of froth separation, are recommended to increase the diamond-bearing kimberlite beneficiation process performance, providing cleaning of the surface of diamonds due to destruction of their accretions with rock minerals and removal of film hydrophilizing coatings from the surface of diamond crystals [2, 5, 6]. Significant increase of diamond recovery into froth separation concentrate can be achieved through applying a combination of physical and physical-chemical action methods to solid and liquid phases of diamond-bearing material pulp [7, 8]. The combination of diaphragmless electrochemical treatment of the froth separation cycle recycled water with thermal treatment of the initial material entering the froth separator was proposed as an effective physical-and-chemical method for modifying the properties of mineral components and the liquid phase of the diamond-containing pulp [9], which provided the change in the ionic-molecular composition of the aqueous phase and encourages solving the problem of cleaning the diamond surface from the surface hydrophilizing compounds.

In order to improve the efficiency of the proposed combined process, it is necessary to reveal the regularities of dissolution processes of surface compounds on the diamond crystals when using electrochemical conditioning of recycled water of the froth separation cycle, which allow selecting the method of the most intensive and complete removal of hydrophilizing impurities from the diamond surface and restoration of their natural floatability.

Research methods

The chemical analysis of the surface compounds on diamonds was performed by electron-probe X-ray spectral analysis (EPXSA) with the use of an electron microscope Jeol-5610 and an energy dispersive attachment (EDA) [10]. The thickness and character of spreading of the selected salient, point and film mineral compounds were determined during scanning of the crystal surface. Assessment of changes in the composition of the surface compounds on diamonds was performed by infrared spectroscopy using a Bruker FTIR spectrophotometer with standard techniques and databases for deciphering infrared spectra [11, 12]. Modeling the processes formation – dissolution of hydrophilizing compounds on the surface of minerals was carried out by plotting and analysis of thermodynamic stability diagrams for calcium and magnesium compounds in coordinates lg[CO_3] – pH based on thermodynamic analysis of chemical reactions [13]. To estimate the diamond surface hydrophobicity, the method of measuring three-phase limiting wetting angles was applied. The studies were carried out at an OCA 15EC device using standard techniques [14]. A drop of the aqueous phase (1 mm in diameter) was applied with a syringe to flat area of the diamond surface, which was pretreated with the initial recycled water of the froth separation cycle to achieve the effect of technogenic hydrophilization, and then with its electrochemical conditioning product obtained in the established treatment conditions.

A sample of kimberlite material with a given content of diamonds of 1.5–2 mm in size was used as a subject of research in the process of froth separation. Preparation of diamonds for the experimental studies included chemical cleaning of their surface, which was repeated for each individual experiment and consisted of the following operations: diamond crystals were dried, washed in carbon tetrachloride, alcohol, distilled water, and treated with concentrated solution of hydrochloric acid, which removes mineral films. To obtain reliable results, each experiment was repeated twice. The prepared diamonds were mixed with a selected kimberlite grain size of −1+0.5 mm, and the resulting sample was fed to froth separation into a laboratory facility. After the experiment was completed, the diamonds were extracted from the concentrate and tailings, weighed and fed back into the purification cycle.

Process tests were carried out according to the accepted technique of semi-industrial tests using the current products of the diamond-containing kimberlite beneficiation process flow sheet with the plant reagent treatment scheme.

1. Analysis of composition and conditions of destruction of hydrophilizing compounds on the diamond surface

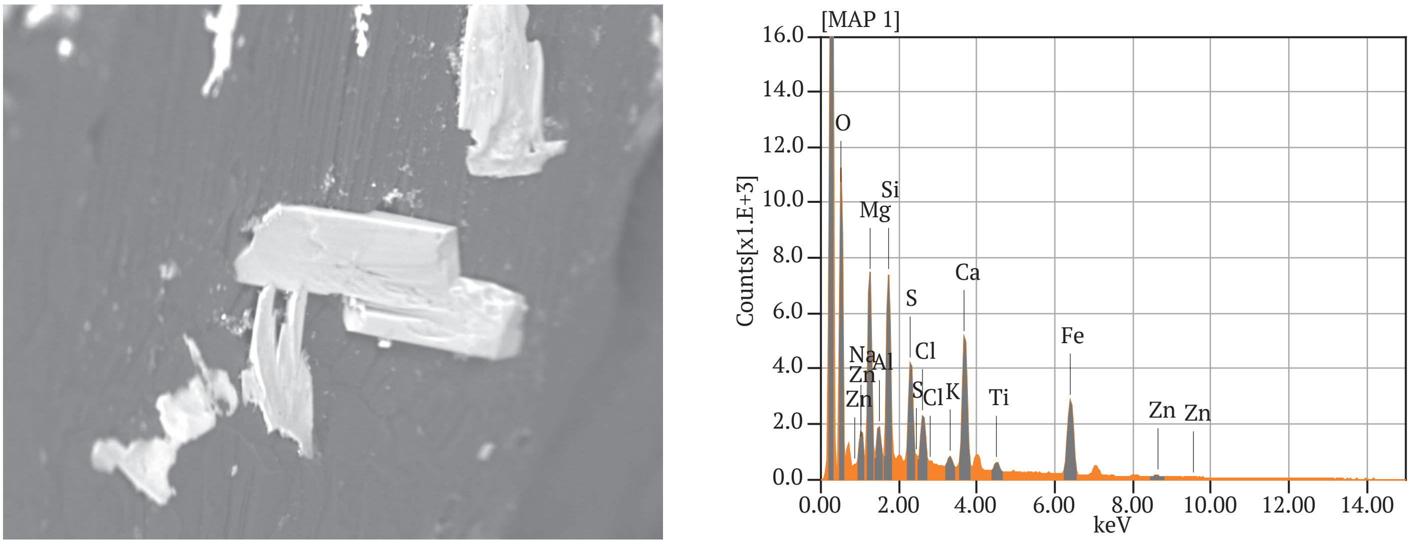
The elemental and phase composition of compounds on the diamond surface was determined by method of X-ray photoelectron spectroscopy. On the hydrophilic surface of the crystals, it was detected the presence of relief mineral compounds of carbonate-silicate composition (Fig. 1), film compounds of predominantly carbonate composition, and slurry coatings of predominantly aluminosilicate composition.

Analysis of the study results showed that the share of mineral compounds on hydrophobic diamonds does not exceed 25 % of the surface area, whereas on hydrophilic diamonds the share of mineral compounds reaches 70 % and their thickness is 2–3 times greater than on hydrophobic diamonds.

Mineral compounds containing carbonates and silicates, as well as, in limited amounts, sulfates, chlorides and other salt-like minerals were identified on the surface of the studied diamonds by the method of infrared spectroscopy based on the character of location of absorption peaks in the spectra. The presence of carbonate minerals was established through characteristic IR absorption bands in the 1400–1500 cm^{-1} range. In most cases the band of valence vibrations (1400–1500 cm^{-1}) was asymmetrical and polymodal that was typical for calcite-dolomite series carbonates.

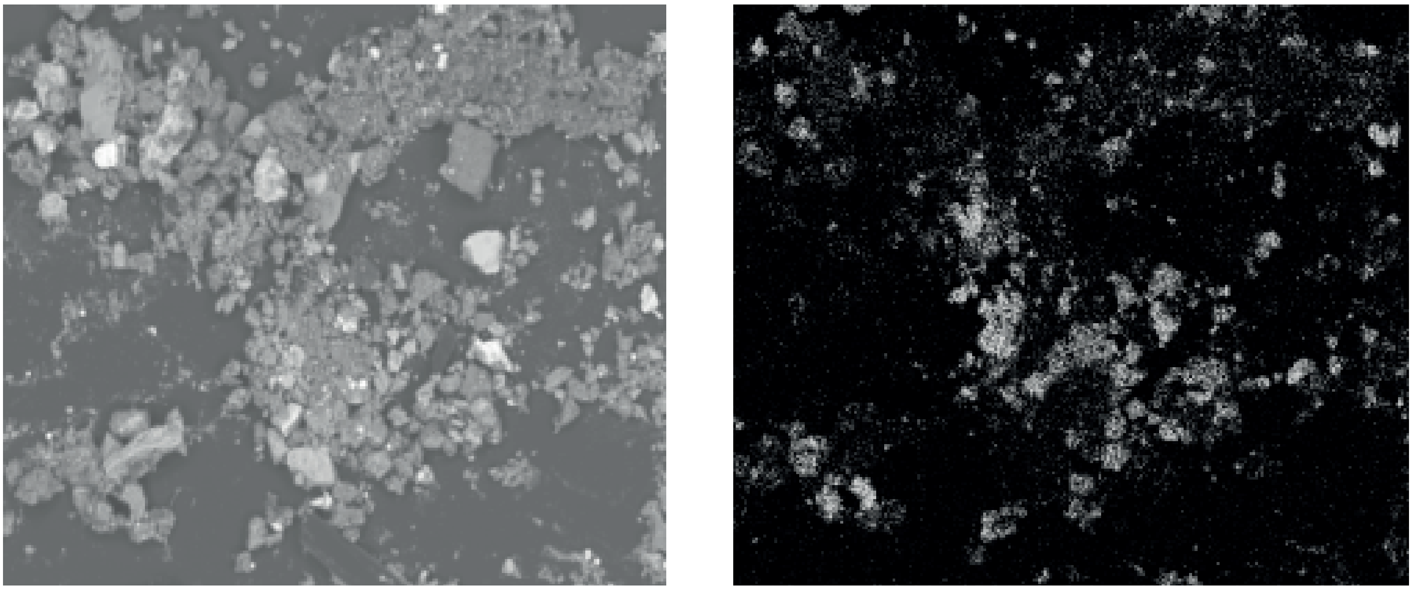
Semi-quantitative analysis of very thin (up to 1 μm) film compounds on the surface of diamonds revealed high surface concentration of calcium in them. The ratio of the sum of magnesium, silicon and aluminum to calcium did not exceed 20 %. This allowed concluding that the thin films on the diamond surface had predominantly calcium-carbonate composition.

Micro-X-ray spectral analysis of the mineral form distribution pattern showed that calcium minerals (mainly calcite) were dispersed together with silicates and occurred on the diamond surface both as individual compact forms and as uniformly distributed films and microspheres (Fig. 2).



a *б*

Fig. 1. Image of the carbon replica (*a*) and X-ray spectrum (*b*) of the mineral formation on the hydrophilic diamond



a *б*

Fig. 2. Images of a combination of mineral compounds (*a*) and distribution of calcium (*b*) on the diamond surface (calcium is shown by white color)

Historical studies established that the most effective method for destruction of such mineral forms of the given composition and, correspondingly, their removal from the surface of diamonds was thermal treatment of ore slurry of the diamond-bearing material froth separation cycle [2].

Purification of the diamond surface during heat treatment proceeds through the thermomechanical mechanism, which implies destruction of intergrown pieces of diamonds with other kimberlite minerals during heating due to different linear expansion [6]. The use of heat treatment is simultaneously a factor influencing the mechanism and rate of dissolution-crystallization processes of the mineral impurities in question. On the one hand, heating of the medium promotes dissolution of simple calcium carbonates and complex magnesium carbonates on the diamond surface due to decreasing carbonic acid solubility [15]. On the other hand, at elevated temperatures, solubility of carbonate minerals decreases, and their crystallization is observed [16]. Therefore the effect of the medium heating on stability of calcium and magnesium carbonate minerals manifests itself at certain conditions which can be determined by thermodynamic analysis of crystallization – dissolution processes.

Decomposition of calcium and magnesium carbonates is described by dissolution reactions, (reactions 1, 2), as well as reactions of transformation of magnesium carbonate and hydroxy carbonate into magnesium hydroxide (reactions 3, 4) [17, 18]:

$$CaCO_3 + H^+ = Ca^{2+} + HCO_3; \tag{1}$$

$$MgCO_3 + H^+ = Mg^{2+} + HCO_{3-}; \tag{2}$$

$$MgCO_3 + 2H_2O = Mg(OH)_2 + CO_3^{2-} + 2H^+; \tag{3}$$

$$MgCO_3 \cdot Mg(OH)_2 \cdot 3H_2O = 2Mg(OH)_2 + HCO_{3-} + H^+ + H_2O. \tag{4}$$

Similar reactions occur involving minerals that are salts of silicic acid:

$$CaSiO_3 + H^+ = Ca^{2+} + HSiO_3; \tag{5}$$

$$MgSiO_3 + H^+ = Mg^{2+} + HSiO_3; \tag{6}$$

$$Ca_2SiO_4 + 4H^+ = 2Ca^{2+} + H_4SiO_{4-}; \tag{7}$$

$$Mg_2SiO_4 + 3H^+ = 2Mg^{2+} + H_3SiO_4. \tag{8}$$

Proceeding of the considered reactions results in intensive dissolution or destruction of carbonate and silicate minerals, and their mechanical or thermomechanical removal from the surface of diamonds is also facilitated.

Modeling of the processes of hydrophilizing compounds dissolution on the surface of minerals was carried out by construction and analysis of thermodynamically stable calcium and magnesium compounds diagrams in coordinates lg[CO_3] – pH based on thermodynamic analysis of chemical reactions involving calcium minerals. These minerals form a basis of the film hydrophilizing coatings on diamonds, crystallizing (coatings) from the liquid phase in technological processes of beneficiation, and cement matrix of the relief polymineral accretions, formed as a result of hypergenic processes in kimberlite [6].

Formation of calcium carbonate is possible both at the stage of grinding of diamond-bearing material and in the subsequent process of its froth separation. This was proved by location of area 1 at a noticeable distance from the phase boundary (Fig. 3). The thermodynamic calculations showed that heat treatment leads to dissolution of surface films of calcium-carbonate composition on the diamond surface at the expense of significant removal of carbon dioxide from the aqueous phase (less than 10^{-5} mol/l, Fig. 3, area 2). Lower intensity of heat treatment did not provide dissolution of calcium carbonate due to lower decrease in concentration of carbonic acid ions.

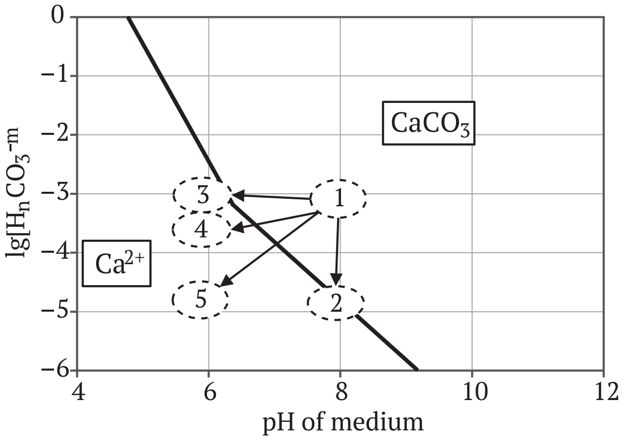


Fig. 3. Diagram of thermodynamic stability of $CaCO_3$ (calcite) in the liquid phase of froth separation, where *1* – without using preparation techniques; *2* – when using heat treatment; *3* – at chemical regulation of the medium pH; *4* – when using electrochemically treated recycled water; *5* – at combined application of electrochemical and thermal treatment

Under the conditions considered, intensification of dissolution of carbonate compounds on the surface of diamonds can be also achieved through acidification of the aqueous phase (Fig. 3, area 3).

Electrochemical conditioning of recycled water of the froth separation cycle leads to changes in both pH values of the ore slurry aqueous phase and concentrations of ion-molecular components in it. For example, the application of diaphragmless electrochemical conditioning of recycled water led to 30–40 % decrease in concentrations of calcium and magnesium ions and bicarbonate ions in it, and the aqueous phase thus became less supersaturated in relation to the process of formation of calcium carbonates [4]. However, as can be seen from Fig. 3 (area 3), the magnitude of acidification (decreasing pH) achieved through electrochemical treatment of recycled water and the corresponding decrease in the concentration of calcium ions may be insufficient to dissolve calcium carbonate precipitates.

The most effective way to solve the problem of dissolution of carbonate compounds on the surface of diamond crystals is a combination of thermal and electrochemical treatment of diamond-containing ore pulp and recycled water of froth separation respectively, at which maximum possible dissolution of calcium carbonate is achieved (Fig. 3, area 4).

Similarly, application of the combined technology of the pulp thermal treatment and electrochemical conditioning of recycled water provides destruction of the main magnesium carbonates ($MgCO_3Mg(OH)_2nH_2O$).

Thus, the obtained results of thermodynamic calculations explained the efficiency of the combined application of thermal treatment of ore slurry and the electrochemical method of the aqueous phase ionic-molecular composition regulation in the diamond-containing material froth separation cycle through creating optimal conditions for dissolution of hydrophilizing compounds on the surface of diamonds.

2. Determination of optimal parameters of electrochemical treatment of the froth separation cycle recycled water

Taking into account the above-mentioned recommendations for using the electrochemical conditioning of recycled water of the froth separation cycle as an additional method, activating the process of dissolution of carbonate hydrophilizing compounds on the surface of diamond crystals, a set of experimental studies to determine the optimal parameters of the liquid phase, providing maximum efficiency of the method application was implemented.

At the first stage of the experimental researches, stability of hydrophilizing compounds on diamond surface was estimated by method of determination of limiting pH values of calcium carbonate precipitation in liquid phase of aqueous systems at different mineralization and temperatures.

To determine dissolution conditions of carbonate compounds under the given conditions, two series of experiments with use of low-mineralized and highly mineralized aqueous systems were carried out. In the experiments, control measurements of pH of aqueous phase in the conditions of the sediment crystallization were executed (Table 1).

The boundary values of pH of sedimentation were calculated from equations for the reaction 1 constants taking into account the actual concentrations of calcium ions and carbonates. Formation of calcium carbonate precipitates occurs in more alkaline region (pH = 6.4–7.2) than their dissolution (pH = 6.0–6.57). The results obtained agree with the data of other researchers, explaining such effect by the necessity of conditions of "supersaturation" in the ions forming crystallizing compounds for their precipitation [16]. Comparison of the calculated data with the actual pH values of formation and dissolution of the precipitates showed that the calculated boundary pH values of the calcium carbonate stability area are close to the experimental data.

The obtained results allow estimating the interval of pH at which the stability of carbonate films decreases, and their dissolution becomes possible both at normal and at elevated temperatures. However, the calculated pH values of the aqueous phase cannot be used as the only criterion for its ability to dissolve the mineral impurities due to the significant influence on the processes of crystallization – dissolution of such additional factors as temperature, concentrations of ionic-molecular components, ionic strength of the solution [17, 18].

The ability of aqueous phase to dissolve or precipitate salts, particularly calcium carbonates, in related areas of industry (oil production and heat power engineering) is evaluated using specialized complex criteria. As such a complex criterion for the ability of recycled water of diamond-containing material froth separation to dissolve hydrophilizing films on the surface of diamond crystals, a pH-based criterion, the Ryznar Stability Index (JS_t) is used [19]:

$$JS_t = 2 \cdot \mathrm{pH_S} - \mathrm{pH}. \qquad (9)$$

Table 1

Calculated and experimentally measured pH of calcium carbonate dissolution

Experimental conditions	$[Ca^{2+}]$, mg/L	T °C of aqueous phase	$[CO_3]$ concentration, mg/L	pH of precipitate formation		pH of precipitate dissolution
				Calculated value	Revised value	Revised value
Low mineralized aqueous system	2150	20	30	6.48	6.6	6.2
		70	4	6.92	7.2	6.57
Highly mineralized aqueous system	7000	20	30	6.23	6.4	6.0
		70	4	6.62	6.88	6.34

The Ryznar Index compares the actual pH value of the medium with the required value (pH_S) for the formation of calcium carbonate, which is calculated using the following equation:

$$pH_S = pK_2 - p\Pi P CaCO_3 - \lg[Ca^{2+}] - \lg Alk_o + \\ + 2{,}5\sqrt{I}, \qquad (10)$$

where pK_2 is the negative logarithm of the H_2CO_3 2nd stage dissociation constant; $p\Pi P CaCO_3$ is the negative logarithm of the solubility product of $CaCO_3$ (calcite); $[Ca^{2+}]$ is the concentration of Ca^{2+} ions in the aqueous phase, mg/l; Alk_o is the total alkalinity of the aqueous phase, mg-eq/l; I is the ionic strength of the aqueous phase.

At values of $JS_t > 8.7$ the aqueous phase tends to dissolve carbonates, and $CaCO_3$ precipitate is not formed or dissolves easily. At $8.7 > JS_t > 6.9$ the aqueous phase is characterized as "moderately aggressive" in which $CaCO_3$ precipitate is not formed and dissolves slowly. At $6.9 > JS_t > 6.4$, the aqueous phase is "stable"; at $6.4 > JS_t > 3.7$, $CaCO_3$ precipitates from the aqueous phase; at $JS_t = 3.7$, the aqueous phase is supersaturated with the precipitate-forming ions that stipulates intensive formation of $CaCO_3$ precipitate [16].

The technology of diaphragmless electrochemical treatment of recycled water of diamond-containing ore material froth separation, which provides simultaneous saturation of the aqueous phase with fine bubbles of electrolytic gases and decreasing concentration of hardness salts in it, was proposed to regulate pH of aqueous phase [7]. Analysis of the influence of diaphragmless treatment of water system on its properties showed the following: at diaphragmless electrochemical conditioning of recycled water of the froth separation cycle at processing plant No. 3 of Mirny Mining and Processing Complex, a shift of pH to more acidic area (pH = 6.2–6.7) takes place. At the same time, an equally important decrease in the concentration of calcium ions and the sum of concentrations of carbonic acid derivatives takes place, which provides intensification of calcium carbonate dissolution. According to the calculated values of stability index JS_t (Table 2), recycled water that has undergone electrochemical conditioning at power consumption of more than 0.5 kWh/m³, changes its properties and passes from the category "prone to calcium carbonate precipitation" to the category "stable", i. e. not prone to precipitate formation ($JS_t = 6.9$). At further increasing consumption of electricity to more than 1.0 kWh/m³ the recycled water passes into the category "moderately aggressive", i.e. capable of dissolving calcium carbonate precipitate ($JS_t \geqslant 7.05$).

Similar results were obtained at electrochemical conditioning of recycled water of processing plant No. 12 of Udachninsky GOK (Mining and Processing Complex), which was characterized by higher mineralization, in particular, increased concentration of calcium ions (Table 2).

As a result of the electrochemical treatment at power consumption of 1.0 kWh/m³ the recycled water of the Udachninsky processing plant passes into the category of "stable" ($JS_t = 6.87$, see Table 2), and at electricity consumption of 1.5–2 kWh/m³, into the "medium aggressive" category ($JS_t = 6.94–7.43$, see Table 2). It should be noted that for the more saturated with calcium ions recycled water of processing plant No. 12 (Udachninsky GOK) the transition to the state, in which dissolution of calcium carbonates takes place,

Table 2

Ionic composition and characteristics of carbonate equilibrium indices at varying intensity of diaphragmless treatment of recycled water

Power consumption, kWh/m³	pH of aqueous phase	Concentration of ions, mg/l			Values of carbonate equilibrium indices		
		Ca^{2+}	HCO_3^-	CO_3^{2-*}	S_1	pH_s	JS_t
Recycled water of the Mirny Processing Plant No. 3							
No treatment	7.6	2389	61.2	0.11	17.00	7.12	6.52
0.3	7.5	2302	62.4	0.091	13.35	7.15	6.82
0.5	7.2	2120	46.5	0.034	4.61	7.10	6.90
1.0	7.1	1833	44.2	0.026	3.04	7.05	7.00
1.5	6.9	1545	41.5	0.015	1.59	7.11	7.33
2.0	6.7	1455	31.5	0.0073	0.73	7.04	7.80
Recycled water of Udachninsky GOK Processing Plant No. 12							
No treatment	8	2900	32	0.33	27.11	7.41	6.13
0.3	7.9	2880	28	0.25	18.82	7.47	6.28
0.5	7.7	2650	26	0.16	10.18	7.31	6.47
1.0	7.5	2500	21	0.074	3.93	7.31	6.87
1.5	7.3	2355	14	0.031	1.64	7.50	6.94
2.0	6.9	2189	10	0.011	0.56	7.46	7.43

is achieved at greater consumption of electricity for its electrochemical conditioning.

To evaluate the effectiveness of the selected boundary values of the JS_t criterion and the electrochemical treatment modes, the method of measuring the three-phase limiting wetting angles of the surface of diamond crystals was applied. The studies were carried out at an OCA 15EC device using standard techniques [14]. A drop of the aqueous phase was placed with a microsyringe on a certain area of a diamond surface, which was pretreated with recycled water that underwent the electrochemical conditioning. The measurements results showed that the hydrophobicity of naturally-hydrophilic diamonds increases in the process of interaction with electrochemically treated recycled water (the hydrophobicity was determined by the value of the three-phase limiting wetting angle). The best result – the maximum increase in the limiting wetting angle of the hydrophilized diamonds was achieved when combining thermal conditioning of the diamonds and diaphragm-free electrochemical treatment of recycled water (Fig. 4, curve 2). The increment of the three-phase limiting wetting angle (33 %) achieved by the combined treatment is significantly higher than the sum of the limiting wetting angle increments of the separate application of the heat treatment technology (17 %, Fig. 4, curve 1).

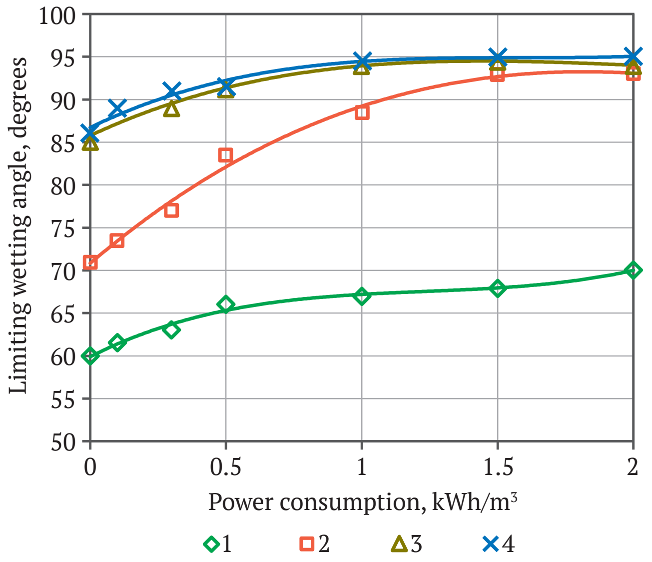


Fig. 4. Curves of the three-phase limiting wetting angle when using thermal treatment of diamonds and diaphragmless electrochemical treatment of recycled water, where *1*, *2* – naturally-hydrophilic diamond; *3*, *4* – naturally-hydrophobic diamond; *1*, *3* – without thermal treatment; *2*, *4* – with thermal treatment

When naturally-hydrophobic diamonds come in contact with recycled water, electrochemically treated in diaphragmless electrolyzer, an increase in the three-phase limiting wetting angle is also observed. A similar result, the maximum degree of surface hydrophobicity, was also achieved with the combined treatment of naturally-hydrophobic diamonds (Fig. 2, curve 4). Analysis of the obtained results showed that the maximum degree of diamond hydrophobicity restoration was achieved at the consumption of electric power (during processing of recycled water in the diaphragmless electrolyser) of more than 1.5 kWh/m^3.

3. Selection of modes of combined conditioning technology of initial feed and recycled water of froth separation cycle

Selection of rational parameters of electrochemical conditioning of recycled water for the preparation of diamond-bearing material – feed of the froth separation cycle – was justified by the results of the laboratory research, carried out in conditions close to the plant conditions. The reagent scheme included feeding bunker fuel F5 (1.3 kg/t) as a collector and sodium hexametaphosphate (0.3 kg/t) as a depressor of rock minerals.

In the research, the diamond-containing material entering the froth separation process was treated with open steam at 60–90 °C. Electrochemical conditioning of the froth separation recycled water was carried out in a diaphragmless electrolyzer EKVB-0.5. The treated recycled water was fed to the attrition and desliming process. The flow rate of water through the electrolyzer was kept constant due to requirements of the production process. Electricity consumption for conditioning of the recycled water was regulated by changing the electrolyzer electrode current density.

The results of the experiments showed that thermal treatment of froth separation feed (diamond-bearing material pulp) allows to increase diamond recovery from 65.7 to 84.1 % (Table 3) at heating the pulp to 80–85 °C with duration of steam treatment from 1 to 2 minutes. Simultaneous applying thermal treatment of the froth separation feed (diamond-bearing material pulp) and electrochemical conditioning of the recycled water increased the diamond recovery to 91.4 % (see Table 3).

Table 3

Diamond recovery into froth separation concentrate when using heat treatment of initial feed and electrochemical conditioning of recycled water

Temperature of initial froth separation feed, °C	At five values of electricity consumption for recycled water conditioning, kWh/m^3				
	0	0.5	1.0	1.5	2.0
	Recovery of diamonds into concentrate, %				
14	65.7	69.1	72.5	72.8	73.4
60	72.3	77.8	81.7	81.9	82.3
80	75.1	84.0	88.3	88.5	88.7
85	84.1	89.4	91.4	91.4	91.3
90	83.9	89.9	90.9	90.8	90.9

Table 4

Parameters of the froth separation semi-industrial tests with the use of electrochemical conditioning of recycled water and thermal treatment of the initial feed

Test condition	Froth separation feed temperature, °C	Electricity consumption for water conditioning, kWh/m^3	Current density on the electrodes, A/m^2	Diamond recovery, %
Check sampling	11–14	–	–	76.5
With heat treatment of froth separation feed	80–85	–	–	77.5
With thermal treatment of froth separation feed and electrochemical conditioning of recycled water	81–86	1.2	175	81.4
		1.5	200	81.6

The increase of electric power consumption for electrochemical conditioning of recycled water above 1.5 kWh/m^3 (when the electrode current density is more than 200 A/m^2) is ineffective, because it doesn‘t lead to an increase in diamond recovery. Heating of the froth separation feed to 90 °C also turned out to be inexpedient because of the identified decrease in diamond recovery. The conducted studies showed a positive synergetic effect of using the combined technology of processing diamond-containing material in the froth separation cycle, which consisted in a significant increase in diamond recovery, significantly exceeding this indicator when heat treatment of the initial diamond-bearing material feed and electrochemical treatment of the recycled water of the froth separation cycle were used separately.

The developed technology of diamond hydrophobic properties restoration in the froth separation cycle of diamond-bearing material was tested during pilot tests in the froth separation cycle of processing plant No. 3 at Mirny GOK. Electrochemical conditioning modes were selected based on the laboratory test data. Recycled water that passed electrochemical conditioning in the diaphragmless electrolyzer was fed to the initial feed desliming unit (removal of grain size class −0.5 mm).

Parameters of thermal treatment of diamond-bearing material feed (the feed temperature of 80–86 °C) also corresponded to the results of the historical studies. Reagent treatment scheme of the froth separation corresponded to the industrial one used in the laboratory researches as well. The analysis of the aqueous phase in the froth separation process confirmed the shift of the medium pH to acidic area (6.1–6.5).

The conducted tests showed that application of the developed method of intensification of the froth separation process with application of the selected modes of thermal treatment of the initial feed (temperature 80–86 °C) and electrochemical diaphragmless treatment of recycled water (Q = 1.2–1.5 kWh/m^3) provided increasing diamond recovery by 4.9–5.1 % (Table 4).

Conclusions

The efficiency of application of recycled water electrochemical conditioning, which provided increasing probability of removal of hydrophilizing compounds from the surface of diamonds during thermal treatment of diamond-bearing feed of froth separation, was shown using the findings of thermodynamic calculations. Rational parameters of diaphragmless electrochemical conditioning of recycled water in the froth separation cycle of diamond-bearing kimberlite with the use of the combined technology for restoration of hydrophobic properties of diamond crystal surface were determined. It was experimentally shown that the maximum increase of hydrophobicity and floatability of diamonds was achieved at the electrolyzer electrode current density of 175–200 A/m^2 and power consumption of 1.2–1.5 kWh/m^3. Pilot tests showed that application of the combined technology, including thermal treatment of diamond-bearing feed and electrochemical conditioning of recycled water of diamond-bearing material froth separation cycle with the selected parameters provided 4.9–5.1 % increase in 2 + 0.5 mm size diamond recovery.

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Research article

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Chemical reactions and conditions of mineral formation at tailings storage facilities of the Russian Far East

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Abstract

Cassiterite-sulfide and polymetallic deposits of the Far Eastern Region (FER) were mined by both open-pit and underground methods. This resulted in the emerging numerous mine workings and tailings storage facilities (TSFs) (abandoned without reclamation in latest decades) and the formation of mining technogenic mineralogical systems. Sulfide component of minerals in the mining technogenic system is subjected to hypergenic and technogenic processes (oxidation and hydrolysis reactions). As a result, highly concentrated technogenic solutions are formed, from which minerals of various classes precipitate. In this connection the purpose of this study was formulated as follows: to show the possibility of crystallization of technogenic minerals from micropore technogenic solutions. In achieving this goal the following tasks were solved: to demonstrate the possible reactions of oxidation and hydrolysis of technogenic minerals at the tailings storage facilities; to identify Eh-pH parameters of their precipitation from highly concentrated solutions; to determine their possible associations. The studies involved field observations and computations with the use of "Selector" software package. The study findings allow demonstrating possible chemical reactions and physico-chemical conditions of mineral formation for the following elements: Fe, Cu, Pb, Zn, Sb, Mg, Al, and Ca, including the following classes of minerals: oxides and hydroxides, sulfates, carbonates, arsenates and silicates. The paper presented for the first time the crystallization reactions of secondary minerals (37 ones) and their physico-chemical conditions. It was found that secondary minerals: jarosite, pitticite, siderite, tenorite, poznyakite, antlerite and ktenasite crystallize in the interval of positive temperatures, while scorodite, chalcantite, broshantite, cerussite, starkeyite, epsomite and rostite originate in cryogenic conditions (below 0 °C). All other minerals, the possibility of precipitation of which was shown in the paper, crystallized in the whole considered temperature interval, from −25 °C to +45 °C. Field studies and modeling data on formation of technogenic waters (solutions) and crystallization of secondary minerals on the surface of and inside tailings at the tailings storage facilities of the Far East showed high intensity of technogenic processes. Since the tailings storage facilities were not reclaimed, the process of environmental pollution, including the hydrosphere, would last for many decades.

Keywords

mining, minerals, beneficiation, wastes, tailings storage facility, technogenic minerals, hypergene minerals, sulfides, association, paragenesis, slime and drainage waters

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ТЕХНОЛОГИЧЕСКАЯ БЕЗОПАСНОСТЬ В МИНЕРАЛЬНО-СЫРЬЕВОМ КОМПЛЕКСЕ И ОХРАНА ОКРУЖАЮЩЕЙ СРЕДЫ

Научная статья

Химические реакции и условия минералообразования на хвостохранилищах Дальнего Востока России

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Аннотация

Касситерит-сульфидные и полиметаллические месторождения дальневосточного региона отрабатывались как открытым, так и подземным способами, что привело к появлению многочисленных горных выработок и хвостохранилищ (в период перестройки брошены без рекультивации), а также к формированию горнопромышленных техногенных систем. Сульфидная составляющая полезных ископаемых в горнопромышленной техногенной системе подвергается гипергенным и техногенным процессам (реакциям окисления и гидролиза). В результате формируются высококонцентрированные техногенные воды, из которых выпадают минералы из различных классов. В связи с этим сформулирована цель данной работы – показать возможность кристаллизации техногенных минералов из микропоровых растворов техногенных вод. При достижении данной цели решались следующие задачи: показать возможные реакции окисления и гидролиза техногенного минералообразования на хвостохранилищах; выявить Eh-pH параметры их выпадения из высококонцентрированных растворов; определить их возможные ассоциации. Исследования проведены с помощью натурных наблюдений и программного комплекса «Селектор». Полученные результаты позволяют продемонстрировать возможные химические реакции и физико-химические условия формирования минералов: Fe, Cu, Pb, Zn, Sb, Mg, Al и Ca из различных классов: оксидов и гидроксидов, сульфатов, карбонатов, арсенатов и силикатов. В статье впервые показаны реакции кристаллизации вторичных минералов (37 наименований) и их физико-химические условия. Установлено, что в интервале положительных температур кристаллизуются вторичные минералы: ярозит, питтицит, сидерит, тенорит, познякит, антлерит и ктенасит, а в криогенных условиях: скородит, халькантит, брошантит, церуссит, старкеит, эпсомит и ростит. Все остальные минералы, возможность выпадения которых показана в статье, кристаллизуются во всем рассматриваемом интервале – от –25 до +45 °C. Натурные наблюдения и полученные при моделировании данные по формированию техногенных вод и кристаллизации вторичных минералов на поверхности и в толще хвостов обогащения на хвостохранилищах Дальнего Востока позволили показать высокую активность техногенных процессов. Так как хвостохранилища не рекультивированы, то процесс загрязнения окружающей среды, включая гидросферу, продлится на многие десятилетия.

Ключевые слова

горное дело, полезные ископаемые, обогащение, отходы, хвостохранилище, техногенные минералы, гипергенные минералы, сульфиды, ассоциация (associare), парагенезис, шламовые и дренажные воды

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Introduction

Mining industry in the Far Eastern Region (FER) developed rather intensively in the last 60–120 years. This produced significant impact on the Region's environment. Cassiterite-sulfide and polymetallic deposits were mined by both open-pit and underground methods. The main ore minerals in the deposits are cassiterite, chalcopyrite, galena, and sphalerite, and, among the sulfides – pyrite, pyrrhotite, and arsenopyrite are common, and, for the Dalnegorsk district, argentite, acanthite, pyrargyrite, and jamesonite should be added [1, 2]. The main extractable ore elements are: Sn, Cu, Pb, Zn, and associated (minor ones): Ag, In, Bi. It is known that not more than 2 % of the extracted ore volume are transformed into concentrate, while the residual 98 % belong to concentration waste (tailings) stockpiled in tailings storage facilities. In the Komsomolsky district, the processing tailings are stored in three tailings storage facilities with total area of 130.5 ha and volume of 37.3 million tons, in the Kavalerovsky district, in six TSFs: 18 hectares and 49 million tons, respectively. In the Dalnegorsk district, there are two tailings storage facilities of the Central processing plant (CPP) with area of 825,000 m^2 and amount of tailings of 56.6 mln tons, and two tailings storage facilities at the Krasnorechenskaya processing plant (KPP): 300,000 m^2 and 6.8 million tons. In the processing tailings, sulfides account for about 4% of the total volume, and the rest is represented by silicate and carbonate minerals of the host rocks. In Komsomolsky and Kavalerovsky districts, the host rock minerals include quartz, chlorite, sericite, tourmaline, epidote, calcite, etc., and, in Dalnegorsk district, they include potassium feldspar, micas, garnets, calcite, etc. The host rock minerals contain the following elements: K, Na, Ca, Al, Mg, and Si. At different points of the tailings storage facilities the percentage contents of sulfides and host rock minerals differ and range from 5 to 95 % and vice versa from 95 to 5 %. It should be noted that the ratio between the sulfides themselves and the host rock minerals at different points in the tailings storage facilities varies both horizontally and vertically.

The processing tailings are permanently exposed to various climatic factors, such as changes in daily and seasonal temperature, precipitation in the form of rain and snow. Hypergene processes in the form of oxidation and hydrolysis reactions of the sulfide component, which are accompanied by the formation of sulfuric acid and destruction of minerals of the host rocks, actively occur in them. This leads to the appearance of highly concentrated micropore solutions and the formation of slime and drainage waters. A number of technogenic minerals precipitate from them (Figures 1–4). Acidic solutions can be partially neutralized by the carbonate components of processing tailings. The creation of a mining technogenic system (adits, pits, trenches, clearings, and tailings storage facilities) promotes the transition of hypergenic processes to the technogenic stage. Over time, technogenesis can lead to the complete destruction of tailings storage facilities (technogenic deposits) and their loss.

The process of technogenic minerals crystallization is clearly seen not only in the underground mine workings, but also in the pit (see Fig. 1) of recycled tailings (on TSF wall), on the vertical cut (clearing) surface (see Fig. 2) and on the surface of tailings (see Fig. 3, 4). 35 technogenic minerals of Fe, Cu, Pb, Zn, Sb, Mg, Al, Ca of different classes: oxides and hydroxides, sulfates, carbonates, arsenates and silicates were found by different mineralogical methods in the Far East Region [2].

Research objectives and tasks

The purpose of this study was formulated as follows: to show the possibility of crystallization of technogenic minerals from micropore technogenic solutions (slime and drainage waters). In achieving this goal, the following tasks were solved: 1 – to demonstrate the possible reactions of oxidation and hydrolysis of technogenic minerals at the tailings storage facilities; 2 – to identify Eh-pH parameters of their precipitation from highly concentrated solutions; 3 – to determine their possible associations.

Research techniques

Modeling is one of the methods for analyzing the processes of alteration of sulfide-containing processing tailings in the course of interaction with oxygen and water at tailings storage facilities, formation of drainage waters and precipitation of technogenic minerals from them. Similar studies were conducted


Fig. 1. Technogenic formations on the wall of the tailings storage facility (Komsomolsky district)


Fig. 3. Technogenic formations on the surface of the tailings storage facility (Kavalerovsky district)


Fig. 2. Technogenic formations on the vertical surface of the clearing (TSF in Kavalerovsky district)


Fig. 4. Technogenic formations on the surface of the tailings storage facility (Dalnegorsk district)

in Namibia [3], Slovakia [4], Brazil [5] and many other countries [6–9]. Computer physicochemical modeling is used to solve various environmental problems in hydro-geochemistry.

These studies were carried out by the authors using “Selector” software package (its development leader was I.K. Karpov in A.P. Vinogradov Institute of Geochemistry of SB of RAS). The software package implements a convex programming approach to the calculation of equilibria in heterogeneous systems by minimizing thermodynamic potentials. One of the key features of this package is the possibility of implementing calculation of complex chemical equilibria in isobaric-isothermal, isochoric and adiabatic conditions in multisystems, where an aqueous electrolyte solution, gas mixture, liquid and solid hydrocarbons, minerals as solid solutions and one-component phases, melts and plasma can be present simultaneously [10]. The calculation of thermodynamic properties of the components in the “Selector” is performed for various temperatures, pressures and activity coefficients. In the calculation of isothermal changes of thermodynamic functions the following is used: equations of dependence of changes in the volume of condensed phases on temperature and pressure; semiempirical equations of gas states at the given parameters. Thermodynamic properties of aqueous solution components in the region up to 1000 °C and 5000 bar are calculated using the modified HKF (Helgeson-Kirkham-Flowers) model [11, 12]. The activity coefficients of the components of the aqueous electrolyte solution are calculated using the Debye-Hückel equation modified by Helgeson [11].

Initial thermodynamic information (Gibbs energy, enthalpy, entropy, and coefficients for the heat capacity equation) was taken from publications[1] [13–20]. The missing parameters were calculated and provided by V.A. Bichinsky and A.A. Tupitsyn, employees of the Vinogradov Institute of Geochemistry, Siberian Branch of the Russian Academy of Sciences.

The following unified thermo-barometric conditions were used in the modeling: T from −25 to +45 °C and P = 1 atm. The systems under consideration were open to the atmosphere. The chemical composition of the atmosphere was calculated on the basis of the study of R. Horn [21]. The composition of 10 kg of the atmosphere in moles includes: Ar – 3.2, C – 0.10, N – 539.48, O – 144.85. Composition of rainwater is as follows [22]: N_3^-, N_2^-, NH_4^+, NH_4N_3, HNO_2, NH_4NO_3, NH_4OH, NH_4NO_2, NH_3, H_2CO_3, NCO_3^-, CO_3^{2-}, $C_2O_4^{2-}$, CH_4, O_2, H_2, N_2, Ar, He, Kr, Ne, OH^-, H^+, H_2O, NO_3^-, HNO_3 (pH = 5.66). In each version of the models, independent components were included in the calculations: the most probable gases, dissolved species, and hypogene and hypergene minerals. Oxidation of the processing tailings was modeled for the association of sulfide minerals and host rock minerals taking into account their percentage ratio. The sulfide/host rock ratio was taken at 5:95, 10:90, 20:80, 40:60, 80:20, and 95:5. More than 500 model variants were simulated and analyzed for each of the considered districts [23–25].

The study findings allowed demonstrating possible chemical reactions and physico-chemical conditions of mineral formation for the following elements: Fe, Cu, Pb, Zn, Sb, Mg, Al, and Ca, including the following classes of minerals: oxides and hydroxides, sulfates, carbonates, arsenates and silicates.

Chemical reactions and physico-chemical parameters of recent mineral formation at the tailings storage facilities of the Far Eastern Region

The study findings showed that the following secondary Fe minerals can crystallize at the tailings storage facilities: goethite, hydrogoetite, fibroferrite, jarosite, pitticite, siderite, and scorodite. For their formation, the presence of iron-containing sulfides: pyrite, pyrrhotite, arsenopyrite or chalcopyrite is required, and these minerals are constantly present in the processing tailings. Where the oxidation reactions of sulfides and crystallization of minerals from slime and drainage waters coincide with those published earlier [26], they will not be presented here for mine waters.

The temperature conditions and Eh-pH parameters of goethite precipitation in tailings are as follows: 0.5–0.97 V, 5.4–13.7, and 0.42–0.95 V, 5.1–13.3 (hereafter, Eh-pH parameters will be given for two temperature intervals of formation, first for temperatures from −25 to 0 °C and then from 0 to 45 °C). Goethite canoccur in thermodynamically stable association (paragenesis) with all minerals whose crystallization conditions are given below. It was identified in the whole considered temperature range and practically at any values of sulfides/host rock minerals ratio in the processing tailings. Hereinafter, the term “association” (or “associates with” when a mineral occurrence is described), is used within the meaning of “thermodynamically stable association” as a synonym of term “paragenesis”. Besides, only thermodynamic stability of minerals was analyzed, while the questions of reaction kinetics were out of scope of this study.

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[1] Shock E. SUPCRT 1992–1998 Database Database. Geopig, Arizona State University. URL: http://geopig.asu.edu/sites/default/files/slop98.dat

Hydrogoetite was identified only in the wastes of the old tailings storage facility of Krasnorechenskaya processing plant, where it was formed at negative temperatures in the range of Eh-pH parameters from 0.6 to 0.7 V and ph 9.7–13.7 in paragenesis with valentinite, gypsum, fibroferrite, calcite and cerussite. In addition, it was identified at sulfide content in the system of 5 to 20 % at 35–45 °C at the system parameters of Eh = 1.06–1.07 V and pH 2.3–2.4, but in association with goethite, plumbojarosite, and fibroferrite.

Fibroferrite at the tailings storage facilities precipitates throughout the whole temperature range under consideration at Eh = 0.58–1.15 V and pH 1.3–13.7.

The precipitation of *jarosite* requires potassium, which was identified in the solution in the form of KSO_4^- and K^+ ions. Consequently, the reactions of its formation due to oxidation of main iron-containing sulfides (pyrite or pyrrhotite) can be represented as:

1) $3FeS_2 + KSO_4^- + 13.5O_2 + 3H_2O + 9\bar{e} \rightarrow KFe_3[SO_4]_2(OH)_6 + 5SO_4^{2-}$;

2) $3FeS_2 + K^+ + 13.5O_2 + 3H_2O + 9\bar{e} \rightarrow KFe_3[SO_4]_2(OH)_6 + 4SO_4^{2-}$;

3) $3FeS + KSO_4^- + 7.5O_2 + 3H_2O + 3\bar{e} \rightarrow KFe_3[SO_4]_2(OH)_6 + 2SO_4^{2-}$;

4) $3FeS + K^+ + 7.5O_2 + 3H_2O + 3\bar{e} \rightarrow KFe_3[SO_4]_2(OH)_6 + SO_4^{2-}$.

It crystallizes in the range of positive temperatures at the following solution parameter: Eh = −0.82–1.16 V and pH 1.1–8.0 and associates (hereinafter this means “thermodynamically stable association” or “paragenesis”) with goethite, gypsum, plumbojarosite and adamine in the Pb-Zn ore processing tailings of the Central Processing Plant and in the 3rd tailings storage facility in the Komsomolsky district. In the latter case, the paragenesis also comprises the following minerals: alunogen, antlerite, ktenasite, vudvardite, pitticite, magnesite, siderite, smithsonite, olivenite, clinoclase, vivianite, allophane, and nontronite.

The next Fe mineral is *pitticite*, for the formation of which oxidation of arsenopyrite is sufficient by the following reaction:

$$2FeAsS + 8.5O_2 + 2H_2O + H^+ + 6\bar{e} \rightarrow$$
$$Fe_2[AsO_4][SO_4](OH) \cdot 2H_2O + SO_4^{2-} + AsO_4^{3-}.$$

It was identified in the processing tailings in the 3rd tailings storage facility in the Komsomolsky district in the paragenesis described above for jarosite. The mineral precipitates in the interval of 15–45 °C from solutions with parameters Eh = 0.76–0.84 V and pH 5.9–8.0.

For the formation of *siderite* during the oxidation of pyrite and pyrrhotite, calcite must be present in the system: $FeS_2 + CaCO_3 + 4O_2 + 2H_2O + 2\bar{e} \rightarrow FeCO_3 + CaSO_4 \cdot 2H_2O + SO_4^{2-}$ or/and $FeS + CaCO_3 + 2O_2 + 2H_2O \rightarrow FeCO_3 + CaSO_4 \cdot 2H_2O^-$ (Eh = 0.76–0.84 V and pH 5.85–7.98 in the temperature range of 15–45 °C). It was found in the tailings of the 3rd tailings storage facility in Komsomolsky district in association with jarosite and gypsum.

Scorodite in the processing tailings was found only at low temperatures from −25 to 0 °C and the following solution parameters: Eh = 0.6–0.97 V, pH 5.3–12.5. Scorodite can occur in paragenesis with the majority of the minerals of the technogenic system.

Among the technogenic Cu minerals in the tai-lings storage facilities, the following ones were identified: tenorite, chalcantite, poznyakite, wroewolfeite, antlerite, broshantite, olivenite and clinoclase, and, if Cu and Zn arpresent in the system, ktenasite; Cu and Al, vudvardite; Cu and Pb, duftite and bayidonite.

Tenorite was identified in the processing tailings in the Kavalerovsky district. It was formed as a result of oxidation of chalcopyrite through the following reaction: $CuFeS_2 + 5.5O_2 + H^+ + 5\bar{e} \rightarrow CuO + FeO \cdot OH + 2SO_4^{2-}$, in the range of positive temperatures, at Eh = 0.42–0.53 V and pH 11.8–13.3, and the ratio sulfide/host rock of 5:95 and 10:90. It occurs in paragenesis with goethite, gypsum, fibroferrite, alunogen, vudwardite, anglesite, magnesite, duftite, allophane, and montmorillonite.

The parameters of the solutions from which chalcantite precipitates are as follows: Eh = 0.51–1.15 V and pH 1.9–13.2, in the temperature range from −25 to −5 °C. It associates with goethite, gypsum, fibroferrite, vudvardite, wroewolfeite, antlerite, starkeyite, alunogen, magnesite, allophane, and montmorillonite.

Poznyakite was established when simulating the oxidation process of processing waste of Vysokogorskoe deposit, Kavalerovsky district, in the absence of arsenopyrite in them, in the range of 25–35 °C. It is formed from the solution with parameters Eh = 0.98 V and pH 3.9, at the ratio sulfide/host rock of 80:20, in paragenesis with gypsum, antlerite, wroewolfeite, fibroferrite, alunogen and anglesite.

Wroewolfeite was observed in temperature interval of 0–25 °C in association with poznyakite, where it precipitated from solutions with the following parameters: Eh = 0.98–1.1 V and pH 3.9–4.0. In addition, it was identified in the interval from −25 to 0 °C at Eh = 1.0–1.1 V and pH 2–4. In this case, posnyakite and antlerite disappeared from the paragenesis, but starkeyite and chalcantite were added.

Antlerite in the tailings of TSFs in Komsomolsky district occurs in association with jarosite, where it originates at Eh = 0.78–0.84 V and pH 5.9–7.4, T = 25–45 °C. In Kavalerovsky district, it was identified in association with poznyakite at Eh = 0.98–1.0 V and pH 3.4–3.9, T = 35–45 °C.

The following conditions are typical for *broshantite* crystallizing at the tailings storage facility of the Central Processing Plant in Dalnegorsk district: temperature: −25 to 0 °C, Eh = 0.6–0.7 V and pH 9.5–13.0, the ratio sulphide/host rock of 5:95 and 10:9, and the paragenesis: goethite, gypsum, calcite, adamine, montmorillonite.

Ktenasite crystallizes in the processing tailings (3rd tailings storage facility, Komsomolsky district) when oxidizing chalcopyrite and sphalerite by reaction: $5CuFeS_2 + 5ZnS + 30.5O_2 + 11H_2O + H^+ + 27\bar{e} \rightarrow (Cu, Zn)_5[SO_4]_2(OH)_6 \cdot 6H_2O + 5FeO \cdot OH + 13SO_4^{2-}$. The conditions for its existence are as follows: T = 15–20 °C, Eh = 0.76–0.77 V and pH 7.8–8.0. It occurs in the association with jarosite described above.

Vudvardite precipitates together with goethite or fibroferrite in the presence of chalcopyrite and aluminum ions in processing tailings: Al^{3+}, $Al(OH)_2{}^+$, AlO_2^- and AlO^+ following the reactions:

1) $4CuFeS_2 + 2Al^{3+} + 22O_2 + 12H_2O + 20\bar{e} \rightarrow Cu_4Al_2[SO_4](OH)_{12} \cdot 4H_2O + 4FeO \cdot OH + 7SO_4^{2-}$

or

2) $4CuFeS_2 + 22O_2 + 32H_2O + 2Al^{3+} + 12\bar{e} \rightarrow Cu_4Al_2[SO_4](OH)_{12} \cdot 4H_2O + 4Fe[SO_4](OH) \cdot 5H_2O + 3SO_4^{2-}$,

or

3) $4CuFeS_2 + 2Al(OH)_2^+ + 21O_2 + 10H_2O + 16\bar{e} \rightarrow Cu_4Al_2[SO_4](OH)_{12} \cdot 4H_2O + 4FeO \cdot OH + 7SO_4^{2-}$,

or

4) $4CuFeS_2 + 19O_2 + 30H_2O + 2Al(OH)_2^+ + 8\bar{e} \rightarrow Cu_4Al_2[SO_4](OH)_{12} \cdot 4H_2O + 4Fe[SO_4](OH) \cdot 5H_2O + 3SO_4^{2-}$, etc.

It associates practically with all minerals of technogenic and hypergenic mineralization, and Eh-pH parameters of its crystallization are as follows: Eh = 0.5–1.15 V, pH 1.8–13.2 and 0.42–0.99 V, pH 4.9–13.3.

Olivenite, duftite, bayidonite, and clinoclase are Cu minerals of the arsenate class, which are characteristic of cassiterite-sulfide ore processing tailings in the Komsomolsky and Kavalerovsky districts. Chemical reactions of olivenite, duftite, and bayidonite formation are given earlier in [26], and those for clinoclase are as follows: $FeAsS + 3CuFeS_2 + 18O_2 + 23H_2O + H^+ + 7\bar{e} \rightarrow Cu_3[AsO_4](OH)_3 + 4Fe[SO_4](OH) \cdot 5H_2O + 3SO_4^{2-}$. Olivenite and bayidonite crystallize from the simulated solutions throughout the whole temperature range under consideration. Under cryogenic conditions (negative temperatures) their crystallization parameters coincide: Eh = 0.5–1.1 V, pH 2.4–13.8. In the interval of positive temperatures, their crystallization parameters are as follows: Eh = 0.86–1.0 V, pH 3.9–6.5 and Eh = 0.86–1.09 V, pH 2.5–6.5, respectively. At negative temperatures, duftite crystallization parameters are as follows: Eh = 0.5–1.1 V, pH 2.5–12.5, and in the interval of 0–45 °C, duftite and clinoclase were established at Eh = 0.42–1.09 V, pH 2.5–13.3 and Eh = 0.76–0.83 V, pH 6.2–8.0, respectively. They were found in association with many technogenic minerals.

Then let us consider the precipitation of Pb technogenic minerals from the solutions formed due to the oxidation of processing tailings: anglesite, cerussite, plumbojarosite (Pb and Fe) and corresponding Zn minerals: adamine and smithsonite.

Anglesite is more often identified in polymetallic ore processing tailings. It was found throughout the whole studied temperature range at the following Eh-pH parameters: Eh = 0.51–1.2 V, pH 0.9–13.2 and Eh = 0.57–1.15 V, pH 1.3–10.6. It occurs in association with valentinite, gypsum and fibroferrite in the Krasnorechenskaya Processing Plant (KPP) tailings storage facility, and with goethite, fibroferrite, vudvardite, starkeyite, alunogen, gypsum, chalcantite, wroewolfeite, allophane and montmorillonite in the Kavalerovsky district's TSFs.

Plumbojarosite was identified in simulated solutions of tailings oxidation with the following parameters: Eh = 0.7–1.2 V, pH 0.9–10.4, and Eh = 0.76–1.38 V, pH 1.1–8.0. It is present in paragenesis with most of the technogenic minerals.

Cerussite is carbonate, and its formation from solutions requires calcite presence in the system. The reaction of its crystallization is as follows: $PbS + CaCO_3 + 2O_2 + 2H_2O \rightarrow PbCO_3 + CaSO_4 \cdot 2H_2O$. It was identified in cryogenic conditions in the temperature range from −25 to −20 °C, at Eh = 0.6 V and pH 13.4–13.7, in the association with valentinite, hydrogoetite, gypsum, fibroferrite at the KPP tailings storage facility, at the simulated ratio sulfide/host rock of 5:95 and 10:90.

Adamine was found only in the presence of sphalerite and arsenopyrite in the processing tailings and was identified at the Central Processing Plant (CPP)'s tailings storage facility: Eh = 0.6–1.2 V, pH 0.9–13.0 and Eh = 0.65–1.2 V, pH 0.9–10.5, where it associated with goethite, gypsum, epsomite, broshantite, chalcantite, anglesite and plumbojarosite.

Smithsonite was identified in the models of tailings oxidation at TSF's of the Komsomolsky district. The chemical reaction of its crystallization in conditions of oxidation of sphalerite in the presence of calcite is as follows: $ZnS + CaCO_3 + 2O_2 + 2H_2O \rightarrow ZnCO_3 + CaSO_4 \cdot 2H_2O$, at conditions: Eh = 0.79–0.97 V, pH 5.3–6.8 (−25 °C), Eh = 0.75–0.84 V, pH 5.7–8.0. It can associate with most of the technogenic minerals.

The only Sb mineral was *valentinite*, which crystallized in the tailings storage facilities of Kavalerovsky district. The solution Eh-pH parameters for valentinite crystallization were as follows: Eh = 0.56–1.1 V, pH 2.6–13.7, and Eh = 0.66–0.98 V, pH 3.7–10.0. It was present in the association mentioned above for calcite.

Since the oxidation of the sulfide components of processing tailings and the generation of sulfuric acid leads to decomposing minerals of host rocks, and the solutions are enriched with Mg, Al and Ca ions, it is expedient to connsider the chemical reactions of formation of starkeyite, epsomite, magnesite, gibbsite (hydrargillite), alunogen, rostite, allophane, kaolinite, nontronite, montmorillonite, gypsum and calcite.

Chemical reaction of *starkeyite* precipitation from the aqueous solution is as follows: $Mg^{2+} + SO_4^{2-} + 4H_2O \rightarrow MgSO_4 \cdot 4H_2O$. The solution parameters are as follows: Eh = 0.5–1.14 V and pH 1.9–13.2, the temperature range is from −25 to −5 °C. In the tailings of the Komsomolsky district, it occurs in association with gypsum, alunogen, fibroferrite, anglesite, chalcantite, and nontronite, and, in the tailings of Kavalerovsky district, goethite, vudvardite, wroewolfeite, magnesite, olivenite, duffite, bayidonite, skodite, allophane and montmorillonite (instead of nontronite) should be added to the paragenesis.

Epsomite was established in the models at the Central Processing Plant tailings storage facility at low temperatures from −25 to −5 °C and the following Eh-pH parameters of the system: Eh = 0.66–1.2 V, 0.9–11.7, in the presence of sulfides above 20%, in association with goethite, gypsum, chalcantite, anglesite, plumbojarosite, calcite, adamine, and montmorillonite. The reaction of its crystallization is as follows: $Mg^{2+} + SO_4^{2-} + 7H_2O \rightarrow MgSO_4 \cdot 7H_2O$.

Magnesite precipitates from the solution by the following reaction: $Mg^{2+} + CO_3^{2-} \rightarrow MgCO_3$. It is formed throughout the whole temperature range under consideration at the solution parameters: Eh = 0.5–1.14 V, pH 1.9–13.2 и Eh = 0.42–0.86 V, pH 5.7–13.3. In the interval of positive temperatures it associates with goethite, fibroferrite, gypsum, alunogen, vudvardite, smithsonite, duftite, allophane and montmorillonite, while at negative temperatures the association is added with gibbsite, olivenite, bayidonite and scorodite.

Gibbsite (*hydrargillite*) can crystallize with involvement of different ion species: Al^{3+}, $Al(OH)_2^+$, AlO_2^-, AlO^+, existing in slime and drainage water solutions, by reactions:

1) $Al^{3+} + H_2O + O_2 + H^+ + 3\bar{e} \rightarrow Al(OH)_3$;
2) $Al(OH)_2^+ + H^+ + 0.5O_2 + 2\bar{e} \rightarrow Al(OH)_3$;
3) $AlO_2^- + H_2O + H^+ \rightarrow Al(OH)_3$;
4) $AlO^+ + H_2O + 0.5O_2 + H^+ + 2\bar{e} \rightarrow Al(OH)_3$.

It was identified in the processing tailings of the Komsomolsky district. The parameters of the simulated solutions were as follows: Eh = 0.89–0.95 V, pH 5.7–6.3, between −25 and −5 °C, and at Eh = 0.77–0.8 V, pH 6.5–7.4, 25–40 °C. Its association coincides with the paragenesis for magnesite.

Alunogen crystallization requires presence of the same ions in the solution, which were established for the formation of gibbsite. 4 possible chemical reactions of the mineral crystallization are as follows:

1) $2Al^{3+} + 3SO_4^{2-} + 17H_2O \rightarrow Al_2[SO_4]_3 \cdot 17H_2O$;
2) $2Al(OH)_2^+ + 3SO_4^{2-} + 15H_2O \rightarrow Al_2[SO_4]_3 \cdot 17H_2O + O_2 + 4\bar{e}$;
3) $2AlO_2^- + 3SO_4^{2-} + 17H_2O \rightarrow Al_2[SO_4]_3 \cdot 17H_2O + 2O_2 + 8\bar{e}$;
4) $2AlO^+ + 3SO_4^{2-} + 17H_2O \rightarrow Al_2[SO_4]_3 \cdot 17H_2O + O_2 + 4\bar{e}$.

It was found throughout the whole studied temperature range at Eh = 0.7–1.15 V, pH 1.3–8.0 and Eh = 0.53–1.16 V, pH 1.3–13.2. Alunogen occurs in paragenesis with almost all technogenic minerals in cassiterite-sulfide tailings in TSFs of the Komsomolsky and Kavalerovsky districts.

Rostite precipitates from solutions with the following Eh-pH parameters: Eh = 0.58–0.6 V, pH 11.8–12.0, in the temperature range from −10 to −5 °C, at the tailings storage facilities of Komsomolsky district through the following reactions:

1) $Al^{3+} + SO_4^{2-} + 5{,}5H_2O + 0{,}5O_2 + \bar{e} \rightarrow Al[SO_4](OH) \cdot 5H_2O$;
2) $Al(OH)_2^+ + SO_4^{2-} + 4{,}5H_2O \rightarrow Al[SO_4](OH) \cdot 5H_2O + 0{,}5O_2 + \bar{e}$;
3) $AlO_2^- + SO_4^{2-} + 5{,}5H_2O \rightarrow Al[SO_4](OH) \cdot 5H_2O + O_2 + 3\bar{e}$;
4) $AlO^+ + SO_4^{2-} + 5{,}5H_2O \rightarrow Al[SO_4](OH) \cdot 5H_2O + 0{,}5O_2 + \bar{e}$.

It was identified in the following association: goethite, gypsum, alunogen, vudvardite, starkeyite, magnesite, scorodite, duftite, mimetesite, kaolinite, montmorillonite.

Allophane formation requires presence of Si in the system (in addition to Al), which exists in solution in the form of $[Si(OH)_2]^{2+}$ ion; hence, it is possible to write 4 reactions of its crystallization:

1) $2Al^{3+} + [Si(OH)_2]^{2+} + 7H_2O + 2O_2 + 8\bar{e} \rightarrow Al_2O_3 \cdot SiO_2 \cdot 8H_2O$;
2) $2Al(OH)_2^+ + [Si(OH)_2]^{2+} + O_2 + 5H_2O + 4\bar{e} \rightarrow Al_2O_3 \cdot SiO_2 \cdot 8H_2O$;
3) $2AlO_2^- + [Si(OH)_2]^{2+} + 7H_2O + \bar{e} \rightarrow Al_2O_3 \cdot SiO_2 \cdot 8H_2O + 0{,}5O_2$;
4) $2AlO^+ + [Si(OH)_2]^{2+} + 7H_2O + O_2 + 4\bar{e} \rightarrow Al_2O_3 \cdot SiO_2 \cdot 8H_2O$.

It was identified in paragenesis with all the considered technogenic minerals, at the following Eh-pH parameters of the simulated solutions: Eh = 0.5–0.95 V, pH 5.7–13.7, and Eh = 0.42–0.87 V, pH 5.4–13.3.

Kaolinite can precipitate from the solution by the following 4 reactions:

1) $2Al^{3+} + 2[Si(OH)_2]^{2+} + 2{,}5O_2 + 10\bar{e} \rightarrow Al_2[Si_2O_5](OH)_4$;
2) $2Al(OH)_2^+ + 2[Si(OH)_2]^{2+} + 1{,}5O_2 + 6\bar{e} \rightarrow Al_2[Si_2O_5](OH)_4 + 2H_2O$;
3) $2AlO_2^- + 2[Si(OH)_2]^{2+} + 0{,}5O_2 + 2\bar{e} \rightarrow Al_2[Si_2O_5](OH)_4$;

4) $2AlO^{+} + 2[Si(OH)_2]^{2+} + 1{,}5O_2 + 6\bar{e} \rightarrow Al_2[Si_2O_5](OH)_4$.

The conditions of its formation from solution are as follows: Eh = 0.59–0.73 V, pH 7.9–12.3, in the temperature range from −10 to +45 °C. It associates with most of the described technogenic minerals in the Komsomolsky and Kavalerovsky districts.

Among other minerals from the class of silicates established in conditions of oxidation of the tailings were *nontronite* and *montmorillonite*. Fe and Si ions are required for their crystallization from solution. The chemical reaction of nontronite formation is as follows:

$$2Fe^{3+} + 4HSiO_3^{-} + H_2O + 0.5O_2 + 6\bar{e} \rightarrow \\ Fe_2[Si_4O_{10}](OH)_2 \cdot 2H_2O$$

and montmorillonite

$$2Fe^{3+} + 4HSiO_3^{-} + 2Al^{3+} + 2Mg^{2+} + nH_2O + 10\bar{e} \rightarrow \\ (Al, Fe, Mg)_2[Si_4O_{10}](OH)_2 \cdot nH_2O + 2H^{+}.$$

The solution parameters at which nontronite precipitates: Eh = 0.79–1.15 V, and Eh = 0.75–1.15 V, in the pH range of 1.3–8.0, and those for montmorillonite: Eh = 0.5–1.1 V, pH 2.7–13.9 и Eh = 0.42–1.0 V, pH 3.3–13.3. Nontronite is typical for the tailings of TSFs of Komsomolsky district, and montmorillonite, for the tailings of TSFs of Kavalerovsky and Dalnegorsk (Central Processing Plant) districts. They are present in paragenesis with most of the minerals under consideration.

Gypsum is one of the most common minerals in the ore mining-and-processing technogenic systems of the Far Eastern Region. It can occur in paragenesis with all the mentioned minerals. The reaction of its formation is as follows: $Ca^{2+} + SO_4^{2-} + 2H_2O \rightarrow CaSO_4 \cdot 2H_2O$, at the precipitation conditions of: Eh = 0.5–1.2 V, pH 0.9–13.9 and Eh = 0.57–1.16 V, pH 1.1–11.6.

Calcite can crystallize by the following reactions:

$$Ca^{2+} + CO_3^{2-} \rightarrow CaCO_3$$

or

$$Ca^{2+} + HCO_3^{2-} \rightarrow CaCO_3 + H^{+} + \bar{e}.$$

It was identified only in the tailings of TSFs of the Dalnegorsk district, where at the CPP it was present in association with goethite, gypsum, broshantite, adamine, montmorillonite, and, at the KPP, with valentinite, goethite, hydrogoethite, gypsum, fibroferrite, cerussite. Parameters of the solutions for its formation were as follows: Eh = 0.56–0.77 V, pH 8.3–13.7 and Eh = 0.66–0.76 V, pH 7.1–9.6.

Conclusion

The use of the "Selector" software package allowed to analyze the processes of present-day mineral formation in the processing tailings (in tailings storage facilities) and to present the reactions of their crystallization from solutions, as well as to estimate the physico-chemical conditions of the minerals formation: Eh-pH parameters and temperature intervals, and establish their paragenesises and ionic composition of the solutions.

Among the main ions of sulfide ore components, the following ones were established in the composition of the slime and drainage water solutions: Cu^{+}, Cu^{2+}, $CuOH^{+}$, Pb^{2+}, $PbOH^{+}$, $Pb(SO_4)_2^{2-}$, Zn^{2+}, ZnO_2^{2-}, $ZnOH^{+}$, $Zn(SO_4)_2^{2-}$, As^{5+}, AsO_4^{3-}, $Sb(OH)_2^{+}$, SbO_2^{-}, SO_4^{2-}, HSO_4^{-}, many of which were toxic [27, 28]. The main ions of the components of the host rocks were as follows: Ca^{2+}, $CaOH^{+}$, K^{+}, Mg^{2+}, Al^{3+}, $Al(OH)_2^{+}$, AlO_2^{-}, AlO^{+}, $[Si(OH)_2]^{2+}$.

The possibility of precipitation of 37 secondary minerals of Fe, Cu, Pb, Zn, Sb, K, Ca, Mg and Al from the classes of oxides and hydroxides, sulfates, carbonates, arsenates and silicates from the technogenic solutions of tailings storage facilities was demonstrated. Oxidation of sulfide components of processing tailings leads to crystallization of secondary Fe minerals: goethite, hydrogoethite, fibroferrite, jarosite, pitticite, siderite and scorodite. Among the Cu mine-rals, tenorite, chalcantite, poznyakite, wroewolfeite, antlerite, broshantite, olivenite, and clinoclase were established, and together with Zn, ktenasite, with Al, vudvardite, and with Pb, duftite and bayidonite. The secondary Pb minerals included anglesite and cerussite; in the presence of Fe, plumbojarosite was formed. Among the hypergene minerals of Zn, adamine and smithsonite were identified. The only Sb mineral was valentinite. Action of sulfate solutions (originated due to oxidation of sulfides) on the host rocks in the tailings leads to precipitation of the following secondary minerals from the technogenic water solutions: For Mg, starkeyite, epsomite and magnesite; for Al, gibbsite, rostite, alunogen, allophane and kaolinite; for Ca, gypsum and calcite. The presence of Fe or Fe together with Al and Mg in the highly concentrated technogenic water solutions leads to the formation of silicates: nontronite or montmorillonite.

It was found that secondary minerals: jarosite, pitticite, siderite, tenorite, poznyakite, antlerite and ktenasite crystallize in the interval of positive temperatures, while scorodite, chalcantite, broshantite, cerussite, starkeyite, epsomite and rostite originate in cryogenic conditions (below 0 °C). All other minerals, the possibility of precipitation of which was shown in the paper, crystallize in the whole considered temperature interval, from −25 to +45 °C.

The obtained data can be an important contribution to the mineralogy of hypergene and technogenic processes and assessment of environmental conditions in the Far Eastern Region. pH of the slime and drainage water ranges from strongly acidic to strongly alkaline. After precipitation of technogenic minerals, the total mass of which reaches 230 g, the concentration of ore elements in the solution can be up to 340 g/L.

Field studies [1, 2] and modeling data on formation of technogenic waters (solutions) and crystallization of the secondary minerals on the surface of and inside tailings at the tailings storage facilities of the Far Eastern Region allowed demonstrating high intensity of the technogenic processes. It was found that the excess of the baseline concentrations of both elements of sulfide ores and their host rocks in the TSF's slime and drainage waters reaches tens of thousands times. Consequently, the tailings storage facilities polluted the environment and the hydrosphere for a century[2] [29–31], but since they were not reclaimed, this process would continue for many decades to come.

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CONSTRUCTION OF MINING ENTERPRISES AND UNDERGROUND SPACE DEVELOPMENT

Research article

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Determination of technological parameters of rock freezing systems based on the condition of maintaining design thickness of ice wall

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Abstract

Artificial freezing ensures the formation of a temporary ice wall around the shaft under construction, which prevents groundwater penetration into the shaft and increases the strength of rocks around the unsupported walls of the shaft until the permanent support is erected. The purpose of the study is to carry out thermotechnical calculation of ice wall with subsequent theoretical analysis of changing ice wall thickness with shifting to the passive freezing stage. The idea of the study is to determine these technological parameters based on the condition of maintaining the design ice wall thickness at the stage of passive freezing. The methodology and results of thermotechnical calculation of ice wall for the clay layer as applied to the case of the shafts under construction of a potash mine in the Republic of Belarus are presented. The thermal calculation of the ice wall was carried out numerically in the ANSYS software package using the finite element method. The findings of the numerical multiparameter modeling allowed theoretical analysis of ice wall thickness decrease with shifting to the passive freezing stage with higher brine temperature. The decrease in ice wall thickness was studied both during normal operation of the freezing station and at emergency operation mode caused by the failure of one of the freezing columns. Special attention in the analysis was paid to studying the influence of the duration of the active freezing stage and the distance between the columns on the decrease in the ice wall thickness. When analyzing changes in ice wall thickness at different distances between the freezing columns, it was found that the most common column spacing in the range from 1.1 to 1.3 m requires observing restrictions on the duration of active freezing to prevent a critical decrease in ice wall thickness during the passive freezing stage or decreasing the distance between the freezing columns. In this case, preservation of positive dynamics of ice wall thickness growth is ensured. For the clay layer considered in the study and the distance between the columns from 1.1 to 1.3 m, the minimum time of active freezing is also about 4.3 months. As a result of the analysis, the technological parameters of the freezing system (duration of the active freezing stage and the distance between the freezing columns) were determined, at which the ice wall thickness at the passive freezing stage did not become lower than the minimum permissible values calculated based on the strength and creep conditions.

Keywords

construction, mine shaft, rocks, groundwater, freezing, ice wall, thermotechnical calculation, modeling, process parameters, freezing column, emergency mode

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СТРОИТЕЛЬСТВО ГОРНЫХ ПРЕДПРИЯТИЙ И ОСВОЕНИЕ ПОДЗЕМНОГО ПРОСТРАНСТВА

Научная статья

Определение технологических параметров систем замораживания пород из условия поддержания проектной толщины ледопородного ограждения

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Аннотация

Искусственное замораживание обеспечивает формирование вокруг строящегося ствола временного ледопородного ограждения (ЛПО), препятствующего проникновению подземных вод в ствол и повышающего прочность горных пород в окрестности незакреплённых стенок ствола до возведения постоянной крепи. Целью исследования является проведение теплотехнического расчета ЛПО с последующим теоретическим анализом изменения толщины ЛПО при переходе на стадию пассивного замораживания. Идея исследования заключается в определении этих технологических параметров исходя из условия поддержания проектной толщины ЛПО на стадии пассивного замораживания. Представлена методика и результаты теплотехнического расчета ЛПО для слоя глины применительно к случаю строившихся стволов одного калийного рудника в республике Беларусь. Теплотехнический расчет ЛПО проводился численно в программном комплексе ANSYS с использованием метода конечных элементов. Результаты численного многопараметрического моделирования позволили провести теоретический анализ уменьшения толщины ЛПО при переходе на стадию пассивного замораживания с более высокой температурой рассола. Исследовалось уменьшение толщины ЛПО как при нормальном режиме работы замораживающей станции, так и в аварийном режиме работы, связанном с выходом из строя одной из замораживающих колонок. Особое внимание при анализе уделялось исследованию влияния длительности стадии активного замораживания и расстояния между колонками на уменьшение толщины ЛПО. При анализе изменения толщины ЛПО при различных расстояниях между замораживающими колонками получено, что для наиболее распространенных расстояний между колонками в интервале от 1,1 до 1,3 м требуется соблюдать ограничения по длительности активного замораживания для предотвращения критического уменьшения толщины ЛПО на стадии пассивного замораживания либо уменьшать расстояние между замораживающими колонками. В этом случае будет обеспечено сохранение положительной динамики роста толщины ЛПО. Для рассмотренного в работе слоя глины и расстояний между колонками от 1,1 до 1,3 м минимальное время активного замораживания также составляет около 4,3 мес. В результате проведенного анализа определены такие технологические параметры системы замораживания (длительность стадии активного замораживания и расстояния между замораживающими колонками), при которых толщина ЛПО на стадии пассивного замораживания не становится ниже минимально-допустимых значений, рассчитанных из условий прочности и ползучести.

Ключевые слова

строительство, шахтный ствол, горные породы, подземные воды, замораживание, ледопородное ограждение, теплотехнический расчет, моделирование, технологические параметры, замораживающая колонка, аварийный режим

Финансирование

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Introduction

The construction of mine shafts in watered soils and rocks is carried out with the use of special methods. One of the most widespread special methods of shaft construction at potassium mines is artificial freezing of rocks [1, 2]. The purpose of artificial freezing is to form a temporary ice wall around the shaft under construction, which prevents groundwater penetration into the shaft and increases the strength of rocks around the unsupported walls of the shaft until the permanent support is erected [3–5].

The sinking of a mine shaft starts only after the formation of a closed contour of ice wall with a design thickness which is determined on the basis of strength and creep conditions [6, 7]. Calculation of ice wall for strength and creep is usually carried out for a certain uniform negative temperature of the rocks composing it [8]. Since in practice the temperature distribution in an ice wall volume is heterogeneous, an important issue in determining the ice wall thickness based on the calculated temperature field is the selection of isotherms, which correspond to the ice wall boundaries. In practice, as a rule, two types of isotherms are used [5]:

1) temperature of actual freezing of water in the pores (about 0 °C);

2) temperature at which strength and rheological properties of frozen rocks were measured (−4 ... −12 °C).

Based on the published information on artificial freezing of rocks [3, 5, 9], it can be assumed that both methods of selecting ice wall boundary isotherms are applicable at the stage of active freezing of the rock mass, when the freezing brine temperature takes minimum values, and the brine flow rate takes maximum values. However, it was shown in [5] that during the passive freezing stage with higher temperatures and lower brine flow rates, there may be a temporary decrease in ice wall thicknesses to values below the minimum allowable ones, determined based on the strength and creep conditions. To a greater extent, such a decrease in ice wall thickness is characteristic of the second method of selecting ice wall boundary isotherms. A decrease in the ice wall thickness is observed during passive freezing both in the normal operation mode of the freezing station, and in the emergency mode, which may consist in the failure of one or more freezing columns [10].

Another important issue is the selection of the distance between neighboring freezing columns and, as a consequence, the total number of freezing columns [11–14]. The issue of changes in ice wall thickness at the passive freezing stage calculated at different distances between freezing columns was not sufficiently investigated in the existing Russian and foreign literature. This issue is relevant in terms of ensuring the reliability of ice wall and the safety of mining operations in shafts under construction with the use of artificial freezing method.

The present study continues the research presented in the paper [5]. The purpose of both the previous and the present paper was to carry out thermotechnical calculation of ice wall with subsequent theoretical analysis of changing ice wall thickness with shifting to the passive freezing stage. In the previous paper [5], the emphasis was placed on carrying out a comparative analysis of calculated ice wall thicknesses using different isotherms and the dependence of ice wall thickness decrease at the passive freezing stage. The present paper attempted to deepen and continue the earlier analysis of the dynamics of ice wall thickness at the passive freezing stage. Particular attention was paid to the influence of the duration of the active freezing stage and the distance between the columns on the ice wall condition. The idea of the study is to determine these technological parameters based on the condition of maintaining the design ice wall thickness at the stage of passive freezing.

Mathematical model

We considered the problem of freezing a rock mass by a circular circuit of freezing crowns under the brine scheme. It was assumed that the following physical processes play an essential role in the formation of ice wall in a rock mass [15]:

1) conductive heat transfer (thermal conductivity);

2) phase transition of water in the pores of the rock mass;

3) heat transfer between the rock mass and the brine circulating in the columns.

As a result of movement of brine with negative temperature in the freezing columns, the surrounding rock mass is gradually cooled and frozen. An ice zone is formed in it, where the pore water is considered completely frozen, and a cooling zone, in which the rock mass is not frozen, but has a lower temperature than in natural conditions at a depth in question. The so-called transition zone (mushy zone), where ice and water are present simultaneously, is also sometimes distinguished between the ice zone and the cooling zone [16, 17].

When modeling heat transfer in a frozen rock mass, the following list of simplifications is accepted:

1) the rock mass has isotropic and homogeneous thermophysical properties in the ice and cooling zones;

2) the phase transition of pore water occurs completely in some small given temperature interval;

3) the vertical component of heat fluxes is negligibly small compared to the horizontal one;

4) at the initial moment of time, the rock mass is completely water-saturated;

5) water in the pore space of the rock mass is considered stationary;

6) local thermal equilibrium between solid rock particles, water, and ice in each elementary volume of the watered rock mass;

7) the centers of the mouths of the freezing columns are located on a circle, and the columns themselves are oriented strictly vertically and separated from each other by the same distance.

The third assumption makes it possible to switch from a three-dimensional problem to a two-dimensional one. However, this assumption requires additional comments. It is applicable only if we consider a mid-section of a horizontal layer of rocks of sufficiently large thickness (more than 10 m), and the time interval of modeling in this case is also limited (less than 200 days). In this case, the influence of vertical heat fluxes will initially take place only at the boundaries of the considered layer of rocks with the adjacent upper and lower layers. As time passes, this effect will spread deep into the rock layer and, at a certain point, when it reaches its midline horizontal section, will lead to a significant distortion of the temperature field as compared to the purely two-dimensional case; from that timepoint, the third assumption cannot be applied.

The introduced assumptions 1, 3, 5, and 7 allow concluding that there is rotational symmetry in the problem. This significantly simplifies the geometric model and allows not to consider the horizontal section of the rock mass as a whole, but to consider its separate sector bounded by the two main planes of the ice wall [5]. The geometric model of the sector of a frozen rock mass is shown in Fig. 1. This model was further used for numerical calculations.

In the geometric model of the rock mass layer, there are several boundaries, *S*, *V*, *B* and *I*. The boundary *I* represents the inner boundary of the computational domain. It "cuts off" and removes from the consideration a small volume of the rock mass near the origin of coordinates (the center of the freezing contour). It was introduced to avoid constructing a finite-element mesh around an acute angle touching the rotational symmetry axis of the computational domain. This makes it possible to improve the quality of mesh elements and increase the stability of the numerical solution. The boundary I should be shifted as much as possible to the rotational symmetry axis of the domain so that the "cut off" volume of the rock mass is negligibly small compared to the total volume of the rock mass subjected to thermal influence.

The mathematical model of the rock mass subject to the thermal influence of the freezing columns is based on the energy balance equation in enthalpic form [18, 19]:

$$\frac{\partial H(T)}{\partial t} = \left[\frac{\partial}{\partial x}\left(\lambda \frac{\partial T}{\partial x}\right) + \frac{\partial}{\partial y}\left(\lambda \frac{\partial T}{\partial y}\right)\right], \qquad (1)$$

$$\lambda = \lambda_{lq}(1-\varphi_{ice}) + \lambda_{sd}\varphi_{ice}, \qquad (2)$$

where H – specific enthalpy of rocks, J/m^3; x, y – orthogonal coordinates, m; t – time, s; λ_{lq}, λ_{sd} – thermal conductivities of the rock mass in the cooling and ice zones, respectively, W/(m · °C); λ_{ice} – rock iceness, m^3/m^3.

The energy balance equation is supplemented by the equations of state:

$$H(T) = H_0 + \begin{cases} \rho_{lq} c_{lq}(T - T_{lq}) + \rho_w nL, & T_{lq} \le T \\ \rho_w nL \cdot (1 - \varphi_{ice}), & T_{sd} \le T < T_{lq}, \\ \rho_{sd} c_{sd}(T - T_{sd}), & T < T_{sd} \end{cases} \qquad (3)$$

$$\varphi_{ice}(T) = \begin{cases} 1, & T < T_{sd} \\ (T_{lq} - T)/(T_{lq} - T_{sd}), & T_{sd} \le T < T_{lq}, \\ 0, & T_{lq} \le T \end{cases} \qquad (4)$$

where H_0 – reference specific enthalpy of rocks at temperature I_{sd}, J/m^3; c_{lq}, c_{sd} – specific heat capacities of the rock mass in the cooling and ice zones, respectively, J/(kg · °C); ρ_{lq}, ρ_{sd} – densities of the rock mass in the cooling and ice zones, respectively,

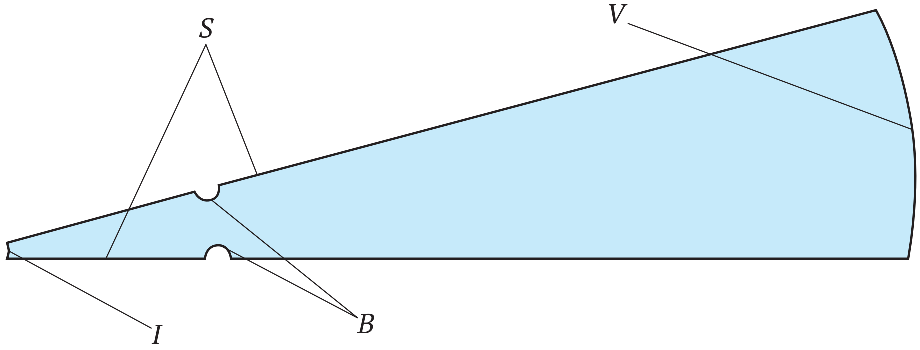

Fig. 1. Computational domain and its boundaries:
S – symmetry; *B* – walls of freezing columns; *V* – outer boundary corresponding to intact rock mass; *I* – inner boundary

kg/m^3; T_{lq} – temperature of the beginning of pore water crystallization (or liquidus temperature), °C; T_{sd} – temperature of the beginning of pore ice melting (or solidus temperature), °C; L – specific heat of pore water crystallization, J/kg; n – porosity of the rock mass; ρ_w – density of water, kg/m^3.

In addition, model (1)–(4) is supplemented by boundary and initial conditions:

$$\left[\lambda\frac{\partial T}{\partial N}-\alpha(T_{fb}(t)-T)\right]\Bigg|_B=0, \tag{5}$$

$$T\big|_V=T_0, \tag{6}$$

$$\left.\frac{\partial T}{\partial N}\right|_S=0, \tag{7}$$

$$\left.\frac{\partial T}{\partial N}\right|_I=0, \tag{8}$$

$$T\big|_{t=0}=T_0, \tag{9}$$

where $T_{fb}(t)$ – brine temperature in the freezing columns, °C; T_0 – temperature of intact rock mass at a distance from the freezing circuit, °C; α – coefficient of heat transfer from the rock mass to the brine through the wall of the freezing column, W/(m$^2\cdot$°C); N – coordinate along the normal to the considered boundary of the computational domain, m.

The water phase transition from liquid to solid state and vice versa is accounted for in the model by setting the nonlinear function (3) of specific enthalpy H dependence on temperature T. In a short temperature interval $[T_{sd}, T_{lq}]$ this function increases sharply by a value equal to the latent heat of the phase transition $\rho_w nL$ in the unit volume of the watered rock mass. This approach to accounting for the phase transition is called enthalpic and is widely used in problems with phase transformations in solids [16, 18].

Same to studies [5, 10], the emergency operation mode of the freezing system is connected with shutdown (failure) of one of the freezing columns at the moment of transition to the passive freezing mode. The freezing column shutdown is modeled by setting a zero heat transfer coefficient in the time interval starting from the moment of transition to the passive freezing mode.

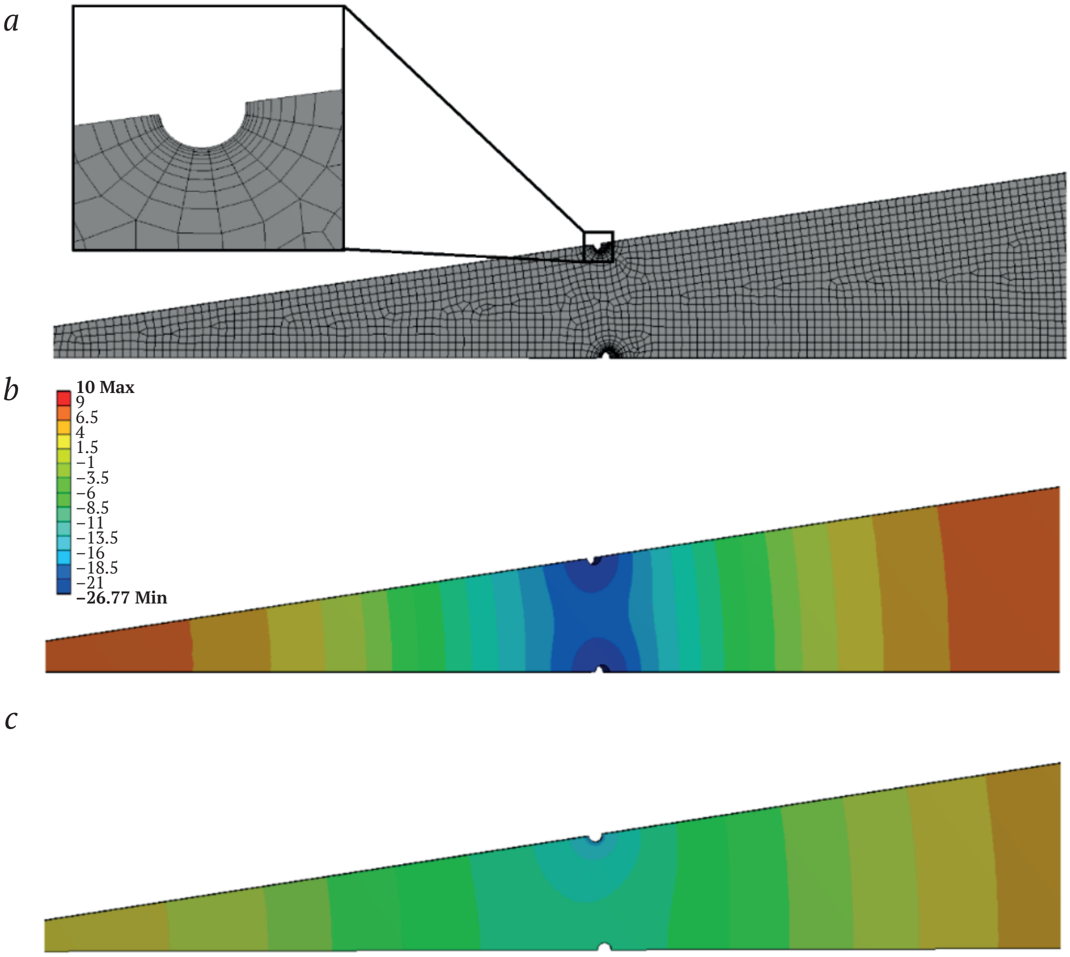

Fig. 2. Finite-element model of the rock mass layer (*a*), calculated temperature distribution in it at the stage of active freezing (*b*) and at the stage of passive freezing during emergency shutdown of one of the columns (*c*)

The authors believe that the passive freezing mode is the most dangerous in terms of faults and accidents. This is due to the fact that, firstly, at the beginning of the passive freezing mode (stage), a change in the operating mode of the freezing station occurs, which itself increases the risk of failure of individual elements of the freezing system. Secondly, shaft sinking and supporting are carried out exactly at the stage of passive freezing. In the practice of shaft construction, there were many cases when freezing columns failed due to severe deformation of rocks before the erection of the advancing concrete support [2, 20, 21]. In addition, shutdown of part of the freezing columns is one of the possible measures to reduce the amount of heat withdrawn from the rock mass during the passive freezing stage [14, 22].

In addition, at the end of the active freezing stage, additional measuring procedures are often carried out in shafts under construction, aimed at verifying that the continuous ice wall reaches the specified thickness [23, 24]. These measurements can be carried out both in control and in freezing wells (e.g., ultrasonic control or thermometric measurements). This also increases the risk of damage to the freezing columns.

Numerical modeling technique

A numerical solution for the problem (1)–(9) was found using the finite element method in the ANSYS software package (Thermal Transient module). The solution was constructed on a grid consisting of rectangular elements (see Fig. 2, *a*). The size of the grid elements in the computational domain and the grid thickening parameters near the freezing columns were selected on the basis of preliminary modeling so as to ensure that the solution was independent of the discretization method.

As parameters for the numerical calculations, the initial data for the rock freezing project for the conditions of the potash mine shafts under construction in the Republic of Belarus were taken. The clay layer was investigated as one of the most thermally conductive layers in the interval of the being frozen rocks. The main thermophysical properties of the rock layer under consideration are presented in Table 1. The density of rocks in the ice zone was assumed to be equal to the density of rocks in the cooling zone. The thermal conductivities in the ice and cooling zones presented in the Table are atypical for the clay layer in question. This is due to several factors: the presence of sand interlayers in the clay, high in-situ pressure (about 1 MPa), and low porosity. In addition, it should be noted that the rock thermal conductivity and humidity values indicated in Table 1 were slightly (within 15 %) corrected in comparison with their initial values in the process of thermophysical model adjustment based on temperature measurements in control thermal wells at the construction site.

Table 1

Thermophysical properties of clay layer

Property	Value
Thermal conductivity (ice zone), W/(m·°C)	4.30
Thermal conductivity (cooling zone), W/(m·°C)	2.64
Heat capacity (ice zone), J/(kg·°C)	900
Heat capacity (cooling zone), J/(kg·°C)	1712
Initial temperature of rocks, °C	10
Liquidus temperature, °C	−0.4
Solidus temperature, °C	−0.9
Density (kg/m^3)	1840
Porosity, decimal quantities	0.158

At the active freezing stage, freezing brine moves in the columns at constant temperature of −30.4 °C. When passing to the passive freezing stage, the brine temperature uniformly rises to −20 °C during 5 days. The flow rate of the freezing brine during the active and passive freezing stages is constant and amounts to 240 m^3/h. The heat transfer coefficient calculated according to the method given in [5] is 62.5 W/(m^2·°C). The outer boundaries of the ice wall were determined by the isotherm T_d = −8 °C, at which the design thickness of the ice wall was calculated based on the strength and creep conditions.

The radius of the outer boundary of the computational domain is 40 m, and the radius of the inner boundary (cut off zone) is 0.25 m. The contour of the freezing columns has a radius of 8 m. The freezing columns have an outer diameter of 0.146 m and an inner diameter of 0.136 m. The distance between the centers of neighboring freezing columns was assumed to be about 1.2 m (the case of 42 freezing columns, distant from each other at the same distance).

Fig. 2, *b* shows the calculated temperature distribution in the sector under consideration for the moment of time of 50 days (the active freezing stage). The smallest value of the ice wall thickness at the stage of active freezing was observed along the lock plane of the ice wall, while at the passive stage, at emergency shutdown of one of the freezing columns (see Fig. 2, *c*, time moment of 100 days), it was observed along the main ice wall plane. For this reason, the minimum of the two thickness values calculated along the main plane and the lock planes of the ice wall was taken as the ice wall thickness.

Influence of the active freezing time on the ice wall condition

Time dependences of ice wall thickness were calculated for several different variants of transition to passive freezing: after 50, 100 and 150 days (Fig. 3). The dashed line represents the curves corresponding to the emergency shutdown of one of the freezing columns, and the solid line represents the accident-free transition to the passive freezing stage.

Analyzing Fig. 3 allowed concluding that during the transition to passive freezing in both emergency and normal modes, a short-term decrease in the ice wall thickness took place. This was evidenced by the characteristic “depressions” on the curves of ice wall thickness dynamics. They were especially characteristic for the case when the duration of the active freezing stage was minimal and amounted to 50 days. As the duration of active freezing increased, the depth of these “depressions” decreased, and eventually, starting from a certain point in time, the decrease in the ice wall thickness at the passive freezing stage stopped. In the case under consideration, this point in time was between 100 and 150 days.

The depth of the “depressions” in the ice wall thickness – time curves in Fig. 3 can be estimated by introducing the following quantitative criterion: the maximum ice wall thickness decrease ΔE [5]:

$$\Delta E = \max(0;\ E_a - E_d), \tag{10}$$

where E_d is the design (minimum permissible) thickness of ice wall, which was achieved at the end of the active freezing stage, m; E_a is the minimum thickness at the passive freezing stage, m.

In [5], a detailed quantitative analysis of this criterion for two rock layers, clay and chalk, was produced. As a result, it was found that at small durations of active freezing of a rock mass (less than 100 days), the decrease in ice wall ΔE thickness essentially nonlinearly depends on the duration of active freezing that was connected with the features of heat transfer near the internal and external ice wall fronts.

In the present paper, we focused on another interesting fact about criterion (10). For longer periods of active freezing of a rock mass (more than 100 days), the value ΔE decreases with increasing duration of the stage of active freezing of rocks and eventually turns to zero, both in the case of trouble-free operation of the freezing system and in the emergency case of shutdown of one of the freezing columns. Physically this conclusion is logical in view of the fact that the longer the stage of active freezing lasts, the deeper the rock

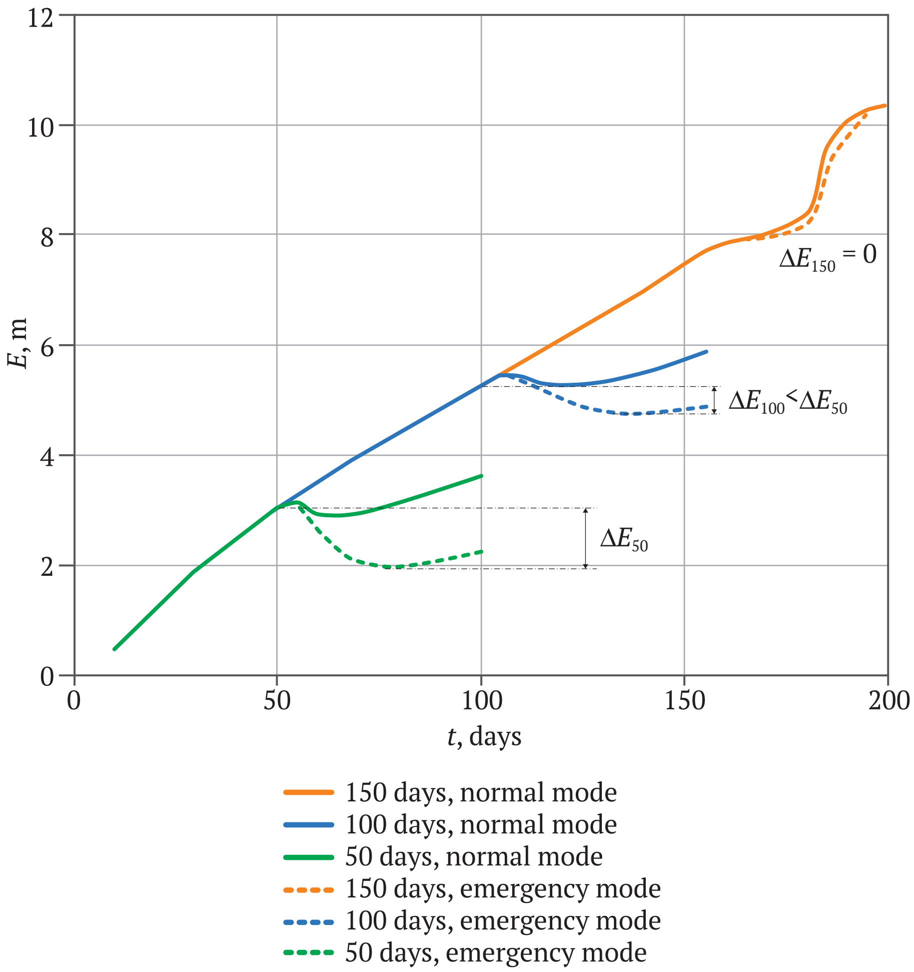

Fig. 3. Dynamics of ice wall thickness E (m) during the transition to passive freezing

mass can be cooled and frozen and, consequently, the higher the persistence (“inertia”) of heat flows in it. Persistence in this case means the ability of the frozen rock mass to maintain negative temperatures for a long time after the cooling productivity of the freezing station was reduced.

In view of this conclusion, it is reasonable to select the duration of active freezing of rocks based on condition $\Delta E = 0$. This condition, in fact, means that at the stage of passive freezing, the ice wall thickness should not decrease to the values below the design ones. For the rock layer in question, the minimum duration of the active rock freezing stage, at which $\Delta E = 0$, is met, is about 130 days or about 4.3 months.

Influence of column spacing on ice wall formation

It is of interest to study the dependence of criterion ΔE on the distance between two adjacent freezing columns. In the present study, such analysis was carried out for the clay layer. Fig. 4 shows the dependences of the criterion ΔE on the distance a between two neighboring freezing columns for active freezing duration of 50, 100, 120, 130 and 150 days for the case of emergency shutdown of one of the freezing columns, as obtained by numerical simulation. Five different distances between the freezing columns were considered: 0.72, 0.96, 1.2, 1.44 and 1.68 m. Analysis of the distances between the freezing columns of 0.7 m and less is meaningless due to the difficulty or impossibility of implementing such distances in practice, taking into account the designed deviations of the positions of the freezing columns from the vertical.

Quite a natural fact follows from Fig. 4: the criterion ΔE is a monotonically increasing function of the distance between neighboring freezing columns. The longer the active freezing period, the stronger the rock mass is cooled and frozen and the lower the value of criterion (10), and the smaller the negative effect of the sudden shutdown of the freezing column. On the whole, the functional form of the criterion $\Delta E(a)$ is significantly non-linear.

For 50 days of active freezing, non-zero reductions in ice wall thickness were observed for all considered freezing column spacing values. For 150 days of active freezing, criterion (10) turned to zero for all the considered distances, except for 1.68 m. For the most common in practice distances between freezing columns (from 1.1 to 1.3 m), the ice wall thickness will decrease if the time of active freezing is less than 130 days.

As noted in [5], the ice wall thickness by the isotherm of actual water freezing (around 0 °C) does not decrease and maintains a positive growth rate throughout the entire period of passive freezing, regardless of the duration of active freezing. At the same time, as can be seen from the calculations produced here, the decrease in ice wall thickness by the −8 °C isotherm is significant, and hence the decrease in the average ice wall temperature is also significant, if the latter is calculated by the isotherm of actual water freezing. This indicated imaginary reliability of

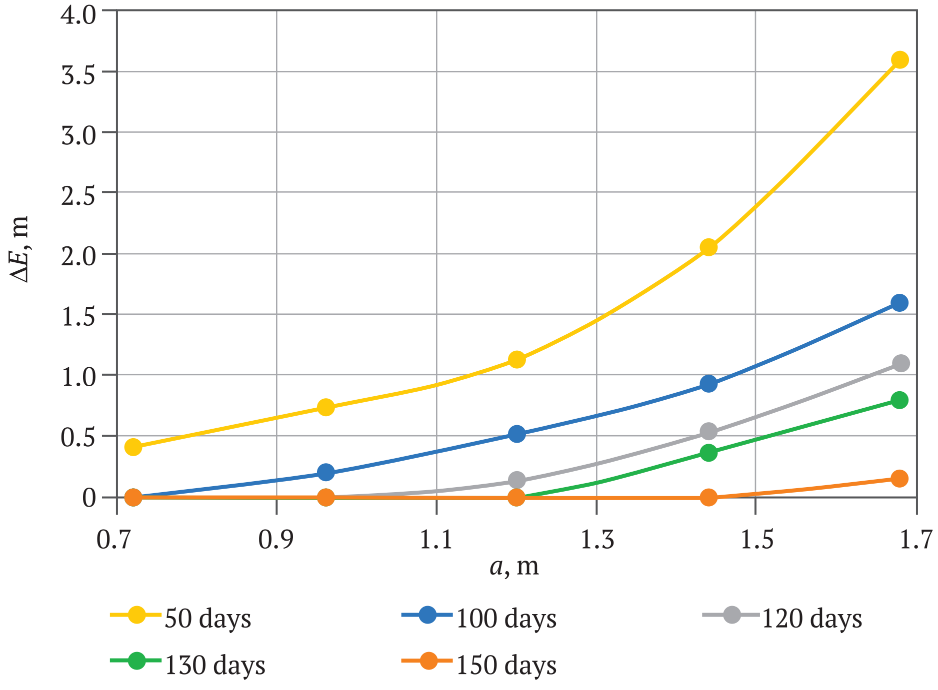

Fig. 4. Maximum ice wall thickness decrease ΔE (m) as a function of the distance between two neighboring freezing columns (m)

ice wall, the thickness of which was determined by the isotherm of actual freezing of water.

The safe distance between the freezing columns, at which no significant decrease in ice wall thickness takes place, depends on the design time of active freezing of the rock mass. Longer active freezing time is more preferable, since in this case it is possible to reduce the decrease in ice wall thickness caused by the transition to the passive freezing mode and the shutdown of one of the freezing columns. If condition $\Delta E = 0$ m, is taken as a safety criterion when selecting the distance a, then for 100 days of active freezing the value of a is about 0.7 m, while for 150 days of active freezing, the value amounts to about 1.5 m. Taking into account that in practice the distances between neighboring freezing columns are usually in the range of 1.1 to 1.3 m, it is sufficient to require that the time of active freezing of the rock mass is not less than 130 days (about 4.3 months). The estimated safe time of active freezing is true only for the considered clay layer.

For larger values of distance a, the time of active freezing should be selected on the basis of Fig. 4 or similar quantitative assessments using criterion (10) or similar criteria. Naturally, selection of the time of active freezing of rocks should also be based on a number of other criteria, for example, the minimum time of freezing of rocks to the design thicknesses. This criterion, together with the proposed criterion ΔE will make it possible to determine the optimal operating mode of a freezing station during the stages of active and passive freezing.

It is important to note that the results obtained in the paper correspond to the case of a fairly sharp increase in the brine temperature during the transition to passive freezing and further maintenance of this value over time throughout the passive freezing stage. There is also an alternative approach to ensuring a given ice wall thickness at the passive freezing stage, involving smooth changing the brine temperature (and flow rate, if necessary) over a long time interval. However, this alternative approach turns out to be not always applicable due to the limited technical capabilities: often the refrigeration equipment used in shaft construction does not allow flexible control of the refrigeration capacity [25].

Conclusion

A theoretical study of changes in the ice wall thickness during the passive freezing stage in normal and emergency operation modes of a freezing station was carried out. As an example, the clay layer from the interval of being frozen rocks as applied to the case of the shafts under construction of a potash mine in the Republic of Belarus was considered. The main findings of the study are given below:

1. When selecting the duration of the active rock freezing stage, it is necessary to take into account the condition of maintaining the design thickness of the ice wall at the passive freezing stage. Fulfillment of this condition strongly depends on the extent to which the rock mass was cooled into the depth. For the clay layer in question, it was found that the minimum duration of the active rock freezing stage, which ensured the design ice wall thickness during passive freezing, was about 4.3 months.
2. When analyzing the changes in ice wall thickness at different distances between the freezing columns, it was found that the most common column spacing in the range from 1.1 to 1.3 m requires observing restrictions on the duration of active freezing to prevent a critical decrease in ice wall thickness during the passive freezing stage or decreasing the distance between the freezing columns. In this case, preservation of positive dynamics of ice wall thickness growth is ensured. For the clay layer considered in the study and the distance between the columns from 1.1 to 1.3 m, the minimum time of active freezing is also about 4.3 months.

The above conclusions are important in view of the current trend to reduce the time of active freezing of rocks in order to accelerate the construction of mine shafts. For example, for the conditions of the shafts under construction at several potash mines in Russia and Belarus the design time of active freezing is from 3 to 4 months. It should be remembered that reducing the active freezing time may not be safe due to a potential decrease in ice wall thickness and loss of its continuity when shifting to passive freezing with possible emergency shutdown of one or more freezing columns.

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Development of automatic system for Unmanned Aerial Vehicle (UAV) motion control for mine conditions

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Abstract

Underground mining operations are connected with significant risks of technogenic accidents, which can be catastrophic. Mitigating the consequences of such phenomena directly depends on the reliability and efficiency of information about the state of parameters of many technological processes, mine workings and facilities located in them. At failure of standard systems of industrial telemetry in conditions of underground mining the creation of new information channels and places of information measurement becomes practically impossible in case of emergency situation development. This predetermines necessity of use of essentially new systems of gathering and transfer of the information, based on robotized autonomous complexes. The task of acquiring reliable information about the situation in an emergency mine working with the help of drones (unmanned aerial vehicles or UAV) in order to make rational decisions in the course of the rescue operation is quite relevant. The aim of the paper was to develop a system of automatic control of an unmanned aerial vehicle (UAV) movement in confined space of a mine working, with significant perturbations of the mine air flow. The mathematical model of UAV movement in mine conditions, based on Euler angles or quaternions, was substantiated. The method of positioning through triangulation with the use of radio beacons was accepted as the basic method that allowed to determine the current position of an UAV. It was proposed to solve the problem of creation of the automatic system for an unmanned aerial vehicle movement control with the use of a hierarchical multiloop control system. The route planning algorithm was formed on the basis of the Dijkstra algorithm. For this purpose, discretization of the future motion space was performed, a labeled connected graph was constructed, on which the arc weights were the distances between the route points. A simulation experiment was implemented. The average deviation from the planned trajectory when flying at a speed of 10 m/s with payload mass up to 0.6 kg did not exceed 1 m, and the maximum deviation was unacceptably large. When flying at 6 m/s with payload mass up to 0.6 kg the average deviation did not exceed 0.3 m, and the maximum deviation, 1.2 m. The results of simulation of movement along the route towards the disturbing mine airflow showed that the control system allowed the UAV with payload of 0.6 kg to withstand the oncoming flow up to 8 m/s. It was obtained that with payload mass of 0.6 kg, the braking distance does not exceed 6 m if the UAV had a speed of 6 m/s, and the braking distance does not exceed 12 m at the speed of 10 m/s. The performed simulation studies confirmed the operating capability of the developed system for automatic motion control.

Keywords

mine workings, mine conditions, accidents, unmanned aerial vehicle, drone, mathematical model, control, coordinates, simulation

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ЭНЕРГЕТИКА, АВТОМАТИЗАЦИЯ И ЭНЕРГОЭФФЕКТИВНОСТЬ

Научная статья

Разработка системы автоматического управления движением БПЛА с учетом шахтных условий

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Аннотация

Ведение подземных горных работ сопряжено со значительными рисками техногенных аварий, которые могут носить катастрофический характер. Снижение последствий таких явлений напрямую зависит от достоверности и оперативности информации о состоянии параметров многих технологических процессов, горных выработок и объектов, в них расположенных. При выходе из строя штатных систем производственной телеметрии в условиях подземных горных работ создание новых информационных каналов и мест измерения информации становится практически невозможным при аварийном развитии ситуации, что предопределяет необходимость использования принципиально новых систем сбора и передачи информации, основанных на роботизированных автономных комплексах. Задача получения достоверной информации об обстановке в аварийной горной выработке с помощью беспилотных летательных аппаратов с целью принятия рациональных решений при ведении спасательной операции является актуальной. Целью статьи является разработка системы автоматического управления движением беспилотного летательного аппарата (БПЛА) в условиях ограниченного пространства горной выработки, при значительных возмущениях шахтного воздушного потока. Обоснована математическая модель движения БПЛА в шахтных условиях, основанная на углах Эйлера или кватернионах. Основным методом, позволяющим определять текущее положение летательного аппарата, принимается метод позиционирования с использованием радиомаяков путем триангуляции. Задачу синтеза системы автоматического управления движением беспилотного летательного аппарата предлагается решать с использованием иерархической многоконтурной системы управления. Алгоритм планирования маршрута сформирован на основе алгоритма Дейкстры. Для этой цели выполняется дискретизация пространства будущего движения, строится помеченный связный граф, на котором весами дуг являются расстояния между точками маршрута. Реализован модельный эксперимент. Среднее отклонение от запланированной траектории при полете на скорости 10 м/с при массе полезной нагрузки до 0,6 кг не превышает 1 м, а максимальное отклонение – недопустимо большое. При полете на скорости 6 м/с при массе полезной нагрузки до 0,6 кг среднее отклонение не превышает 0,3 м, а максимальное отклонение – 1,2 м. Результаты моделирования движения по маршруту навстречу возмущающему шахтному воздушному потоку показали, что система управления позволяет БПЛА с полезной нагрузкой 0,6 кг выдерживать встречный поток до 8 м/с. Получено, что при массе полезной нагрузки 0,6 кг тормозной путь не превышает 6 м, если летательный аппарат имел скорость 6 м/с, и тормозной путь не более 12 м при скорости движения 10 м/с. Проведенные модельные исследования подтверждают работоспособность разработанной системы автоматического управления движением.

Ключевые слова

горные выработки, шахтные условия, аварии, беспилотный летательный аппарат, дрон, математическая модель, управление, координаты, моделирование

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Kim M. L., Pevzner L. D., Temkin I. O. Development of automatic system for Unmanned Aerial Vehicle (UAV) motion control for mine conditions. *Mining Science and Technology (Russia)*. 2021;6(3):203–210. https://doi.org/10.17073/2500-0632-2021-3-203-210

Introduction

Conducting underground mining operations is connected with significant risks of technogenic accidents, which can be catastrophic. Mitigating the consequences of such phenomena directly depends on the reliability and efficiency of information about the state of parameters of many technological processes, mine workings and facilities located in them. At failure of standard systems of industrial telemetry in conditions of underground mining the creation of new information channels and places of information measurement becomes practically impossible in case of emergency situation development. This predetermines necessity of use of essentially new systems of gathering and transfer of the information, based on robotized autonomous complexes [1–4]. The task of acquiring reliable information about the situation in an emergency mine working with the help of drones (unmanned aerial vehicles or UAV) in order to make rational decisions in the course of the rescue operation is quite relevant. The aim of the paper was to develop a system of automatic control of an unmanned aerial vehicle (UAV) movement in confined space of a mine working, with significant perturbations of the mine air flow, and taking into account other factors.

Technological solutions related to UAV operation in underground mine workings exist [5, 6]. They envisage performing a number of specific functions: determining the condition of mine workings, measuring parameters of the mine atmosphere, delivery of life-support equipment to miners who find themselves in an emergency situation. These developments also envisage creation of autonomous control systems for UAVs based on artificial intelligence algorithms, allowing to perform assigned tasks taking into account the mine plan, condition of mine workings, technological processes and mine atmosphere. The solution of such a class of problems requires more detailed elaboration in terms of formalization of systems for automatic control of UAV movement in mine conditions.

1. Mathematical model of UAV movement in mine conditions

The mathematical model of UAV motion as a control object was formed on the basis of the computational scheme (Fig. 1), in which a four-engine unmanned aerial vehicle was considered as a solid body with known [7] aerodynamic properties and assumptions.

The spatial positioning of the UAV was performed in a stationary $\{W\}=\{O_0,\ X_0,\ Y_0,\ Z_0\}$ – inertial coordinate system associated with some fixed point of the mine workings.

The moving coordinate system $\{S\}=\{O,\ X,\ Y,\ Z\}$ – centered in the center of mass of the UAV, shifted by vector $\vec{V}$ relative to the global coordinate system, has axes pointing along the main axes of the UAV as a solid body. This coordinate system has an inertia tensor in the form of a diagonal matrix.

To describe an UAV movement in space, the coordinates $\{x, y, z\}$ in the stationary system and the Euler angles $\{\varphi, \theta, \psi\}$ — roll, pitch and yaw, respectively, in the moving coordinate system were used.

The description of the UAV motion mathematical model, performed in the reference frame S, takes the form [8]:

$$\begin{pmatrix} J_1\dot{\omega}_1 \\ J_2\dot{\omega}_2 \\ J_3\dot{\omega}_3 \end{pmatrix} = \begin{pmatrix} (J_2-J_3)\omega_2\omega_3+M_1 \\ (J_3-J_1)\omega_1\omega_3+M_2 \\ (J_1-J_2)\omega_2\omega_1+M_3 \end{pmatrix}, \qquad (1)$$

where J_i – axial moments of inertia; ω_i – angular velocities; M_i – momentum of forces acting on the UAV along the axes $i=\overline{1,3}$.

The UAV has 6 degrees of freedom: three angles $(\varphi,\ \theta,\ \psi)$ and three coordinates of the center of mass (x, y, z). Thus, the UAV motion is determined by 12 states:

$$\left\{\begin{matrix} D=(x,\ y,\ z),\ E=(\varphi,\ \theta,\ \psi), \\ \dot{D}=(\dot{x},\ \dot{y},\ \dot{z}),\ \dot{E}=(\dot{\varphi},\ \dot{\theta},\ \dot{\psi}) \end{matrix}\right\}.$$

The motion of a solid can be described by three methods: rotation matrices, Euler angles, or quaternions.

The quaternion method is the most efficient: it is compact, because it uses four parameters instead of nine for the matrix method, it has no singularities and no trigonometric functions when converting vectors (unlike the Euler method).

The kinematics of the angular coordinates of an UAV in terms of quaternions Q is described by the following equation:

$$\dot{Q}=0{,}5\begin{pmatrix} 0 & -\omega_1 & -\omega_2 & -\omega_3 \\ \omega_1 & 0 & \omega_3 & -\omega_2 \\ \omega_2 & -\omega_3 & 0 & \omega_1 \\ \omega_3 & \omega_2 & -\omega_1 & 0 \end{pmatrix} Q, \qquad (2)$$

$$Q=(q_0,\ q_1,\ q_2,\ q_3)^T.$$

The UAV is affected by lifting forces F_j, $j=\overline{1,4}$ from each propeller, gravity force $G = mg$, viscous frictional forces, external forces of resistance to motion $N=(N_x\ N_y\ N_z)^T$. The equations of motion according to Newton's law for a solid body in the global coordinate system take the form:

$$\begin{pmatrix} \ddot{x} \\ \ddot{y} \\ \ddot{z} \end{pmatrix} = Q\begin{pmatrix} 0 \\ 0 \\ a \end{pmatrix}Q^T + \begin{pmatrix} 0 \\ 0 \\ -g \end{pmatrix} + k_r\begin{pmatrix} x \\ y \\ z \end{pmatrix} - \frac{1}{m}\begin{pmatrix} N_x \\ N_y \\ N_z \end{pmatrix}, \qquad (3)$$

where m – mass of the UAV; g – acceleration of free fall; k_r – coefficient of proportionality of the viscous friction force; a – total thrust of four propellers; $N=(N_x\ N_y\ N_z)^T/m$ – reduced forces of resistance to motion.

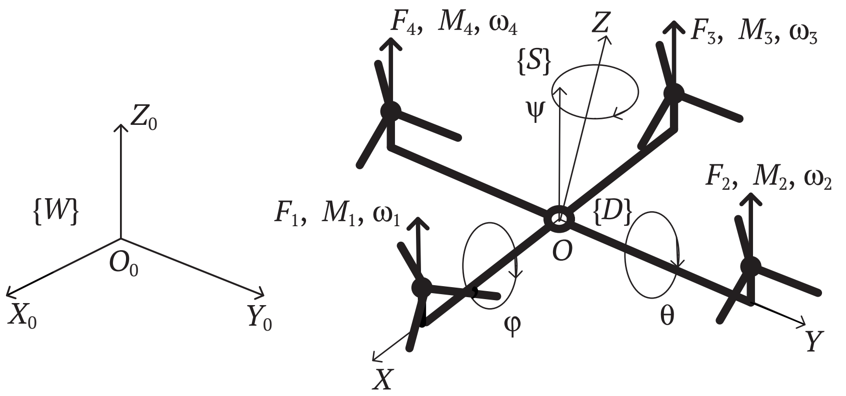


Fig. 1. Unmanned Aerial Vehicle Computational Model

The four UAV motors create an aerodynamic force, which, according to the formula of N.E. Zhukovsky [9], is defined for each j-th propeller as:

$$F_j = c_F \rho r^2 \omega_j^2 S, \quad j = \overline{1,4}, \tag{4}$$

where ω_j is the rotation frequency of the j-th propeller, c_F is the thrust factor which depends on the propeller shape; ρ is the air density; r, S are the rotor radius and the propeller disk area respectively.

The main vector of lifting force — the total thrust vector — is equal to:

$$F = \begin{pmatrix} 0 & 0 & \sum_{j=1}^{4} F_j \end{pmatrix}^T . \tag{5}$$

The axial moments of rotation — the projections of the main moment on the corresponding axes of the moving coordinate system — are determined from the expressions:

$$\begin{aligned} M_X &= (F_2 - F_4)L, \\ M_Y &= (F_3 - F_1)L, \\ M_Z &= M_1 + M_3 - M_2 - M_4, \end{aligned} \tag{6}$$

where L is the distance between the center of mass and the propeller axes.

The thrust vector and axial moments are the control actions to form the UAV motion:

$$u_1 = F, \; u_2 = M_X, \; u_3 = M_Y, \; u_4 = M_Z. \tag{7}$$

The control actions proportional to the square of the propeller rotation frequency are formed by brushless direct current electric motors of independent excitation, the mathematical model of which is represented as a linear dynamic element of the second order.

An integral part of the mathematical model of the UAV motion is the model of the external environment, the main factors of which are constraints in the spatial movement of the UAV and external disturbing influences.

To implement spatial constraints, the method of penalty functions in the form of an exponential "barrier" is used, which sets the potential acting on the object, whose value in the area where movements are allowed is zero, and in the forbidden zone exponentially increases depending on the distance to the constraining boundary, for example, according to the relation

$$F_c(d) = F_0(e^{\alpha d^2} - 1),$$

where d – distance from the UAV to the forbidden area boundary; α – tunable parameter.

When simulating the motion of an unmanned aerial vehicle, it was assumed that its position in space was known accurately enough. The task of current positioning of an UAV can be solved by optical odometry, tag-based positioning, local positioning system based on RFID, audio tag-based positioning, and *SLAM* method, used in mobile autonomous vehicles to build a map in unknown space with simultaneous control of current position and distance covered.

In mine conditions, especially in post-accident conditions, when visibility deteriorates significantly, the use of visual methods becomes impossible. Positioning using ultrasonic triangulation is also unsuitable for this task, since the mine workings have a nontrivial topology.

The method of positioning through triangulation with the use of radio beacons was accepted as the basic method that allowed to determine the current position of an UAV. This method makes it possible to achieve an accuracy of up to a few centimeters, which is an order of magnitude smaller than the size of the UAV. Thus, the position measurement error can be neglected and the exact position of the UAV can be used in the simulation process.

2. Automatic control system for UAV routing

The current position of an UAV in space is determined by six coordinates: three spatial coordinates of the moving center of mass $(x(t), y(t), z(t))$ in the stationary coordinate system, and three angular coordinates $(\varphi(t), \theta(t), \psi(t))$ in the moving coordinate system. The trajectory to be set for the forthcoming motion of the UAV is formed as a function of time in the natural coordinates:

$$r(t) = R(x_d(t), y_d(t), z_d(t)). \tag{8}$$

It was proposed to solve the problem of creation of the automatic system for an unmanned aerial vehicle movement control, according to [10, 11], with the use of a hierarchical multiloop control system with the structure shown in Fig. 2.

In the presented system, block 1 is the route planner designed to form a trajectory of the route to the target at any given moment of time and adjust it with the appearance of obstacles.

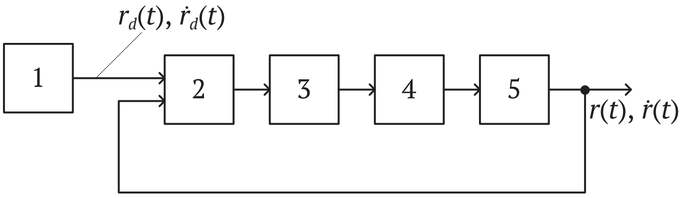


Fig. 2. Structure of automatic UAV trajectory control system

Block 2 of the structure is the block of control of the UAV position in space; it controls the UAV linear coordinates in order to keep it along the planned trajectory of movement.

Block 3 of the structure is the block for controlling the angular position of the UAV; it allows to maintain the required values of quaternions.

Block 4 of the structure is a motor control block, which forms the control actions on the motor windings and allows the formation of the required thrust force of the UAV.

Each block of the system structure, with the exception of the planner block, is linear with a feedback loop and digital *PID* controller [12]. The control system structure turns out to be in a sense similar to that of a slave control system.

Noises and delays of signals arising in real conditions in the "sensors" for determination of angular orientation and location in the fixed coordinate system, used to form feedbacks in the control system, were not taken into account in the simulation.

External disturbing influences — forces of resistance to motion, are generated by air flows, the vector of which can be directed counter, orthogonal or along the direction. In the mathematical model of an UAV motion, these disturbing influences are specified in the form of known functions of time in the right-hand sides of the equations of motion.

The necessary changes in the angular position of the UAV are achieved by changing the rotation frequency of the propellers and, as a consequence, by changing the lifting forces and moments. The task of controlling the angular position of an UAV is to maintain the required angular position of the UAV.

The structure of block 3, where the problem of controlling the angular position of the UAV is solved, is shown in Fig. 3. The structure contains: 3.1 – gyroscope unit; 3.2 – accelerometer unit; 3.3 – gyrostabilizer IMU unit; 3.4 – converter of quaternions into Euler angles; 3.5 – unit of desired angles of UAV position with input from block 2; 3.6 – PID controller; 3.7 – converter of angular accelerations into motor speed; 4 – motor control block.

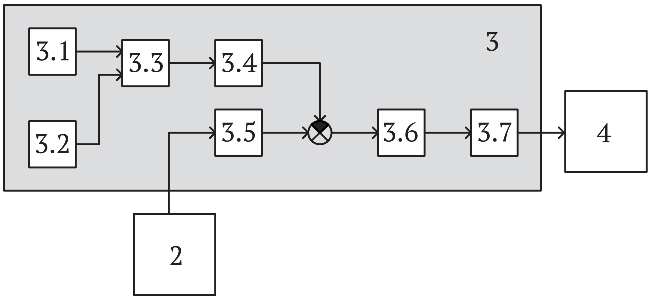


Fig. 3. Structure of UAV angular position control block

The IMU software module is used to determine the angular position of an UAV [13]. It is known that the description of gyroscope and accelerometer operation is more convenient in the quaternionic representation. The IMU module allows UAV to position itself in space without the use of external sensors. The output signal from this module is a four-dimensional vector of angular position of the UAV in quaternion representation. Block 3.4 converts the quaternions into Euler angles [14].

The resulting three-dimensional vector, which characterizes the current angular position of the UAV, allows to determine the angular position error by comparing it with the required angular position of the UAV, which is dictated by the trajectory planner from block 1. The error signal goes to the *PID* controller, where the control signal is formed and sent to the conversion unit 3.7, where the motor rotary speed setting signals are formed.

In the UAV angular position control block, a linearized model of equations (6) was used, in the form of:

$$\begin{pmatrix} J_1\dot{\omega}_1 \\ J_2\dot{\omega}_2 \\ J_3\dot{\omega}_3 \end{pmatrix} = \begin{pmatrix} (F_2 - F_4)L \\ (F_3 - F_1)L \\ (F_1 - F_2 + F_3 - F_4)\gamma \end{pmatrix}, \qquad (9)$$

where L is the distance from the center of mass of the UAV to the motor axes; γ is the conversion coefficient of the motor force into the rotor angular momentum.

Equations (9) represent three equations relative to the four unknown lifting forces F_i, so to unambiguously determine all components of the thrust vector of the motors, we should use the equation of dynamics along the z-axis:

$$m\ddot{r}_3(t) = \sum_{i=1}^{4} F_i - mg\cos\alpha,$$

where α is the angle of inclination of the UAV relative to the vector of gravity.

Denoting

$$B = \frac{m\ddot{r}_3(t) + mg\cos\alpha}{4},$$

we obtained, by transforming equations (8), the basic relations on the required thrust of the engines:

$$\begin{aligned} F_1 &= B + \dot{\omega}_1 + \dot{\omega}_2 + \dot{\omega}_3, \\ F_2 &= B + \dot{\omega}_1 - \dot{\omega}_2 - \dot{\omega}_3, \\ F_3 &= B - \dot{\omega}_1 - \dot{\omega}_2 + \dot{\omega}_3, \\ F_4 &= B - \dot{\omega}_1 + \dot{\omega}_2 - \dot{\omega}_3. \end{aligned} \qquad (10)$$

3. Control of UAV spatial position

The problem of controlling an UAV position in space is solved by determining $\ddot{r}_d(t)$, the required inclination angle of the UAV. In order to keep the UAV altitude at the required level, the thrust of all motors must be equal to

$$\sum_{i=1}^{4} F_i = F.$$

To move in the horizontal plane, it is necessary to set such φ, θ (pitch and roll angles) so that the projection of the thrust vector F on the horizontal plane is directed to the target point. Thus, at each moment of time, knowing the required angular acceleration $\ddot{r}_d(t)$, the required pitch and roll angles $\varphi_d(t)$, $\theta_d(t)$ should be determined.

To keep the UAV position in the vicinity of the desired angles, a linear *PD* controller was synthesized, which formed a control vector based on ε_φ, ε_θ, ε_ψ – the current angular errors of the UAV position.

The *PD*-controller parameters were determined by minimizing the quadratic functional from the angular errors and their derivatives.

Planning UAV trajectory route

The route planning algorithm implemented in Block 1, the route planner of the system, is based on the Dijkstra algorithm [15]. For this purpose, discretization of the future motion space was performed, a labeled connected graph was constructed, on which the arc weights were the distances between the route points. The target point of movement is set by mobile stations *SBGPSMaster-06-WRM* (base stations “GRANCH”), located along the route of UAV movement every 100 m; in this case several variants of routing to the target point are possible.

The found route is a piecewise linear continuous curve, which does not have the required smoothness in conjunctions. The required smoothness property is provided by spline approximation of the curve.

After generating the splines and setting the current speed, time dependences of the required coordinates, speeds and accelerations are generated. These functions are sent to the coordinate controller, where the relations (8) are implemented.

4. The motion control system simulation studies

Experimental computer simulation of the flight control process along the given trajectory, taking into account spatial constraints in mine workings and the mine airflow disturbances, was performed using SimInTech software [16].

For the computer simulation, parameters of the UAV with a span of 0.36 m, a mass of 0.65 kg, a matrix of axial moments of inertia $J = diag(0.002352\ 0.002352\ 0.004704)$ kgm/s^2 were used.

The drives of the propellers are DC motors X2212 KV980 with a linear characteristic of the frequency of rotation $\omega_i = 102{,}6u_{M_i}$, $i = \overline{1,4}$ as a function of the supply voltage *uM*, which takes values from 0 to 15.4 V. The UAV storage batteries of 330W have capacity of 4000 mAh. The developed mathematical model of an UAV as a controllable object is represented by a system of ordinary nonlinear differential equations with restrictions on the permissible range of motion, which reflects the spatial limitations of motion in the conditions of mine workings, and perturbations reflecting the impact of mine air flows [17].

At the first stage of the numerical simulation, the optimum parameters of the *PD*-controller of the the UAV spatial position control block and the UAV angular position control block were selected. Since a linear algorithm is used in these blocks, linearization of the equations was performed to select the coefficients of the algorithm, followed by the calculation of the coefficients of the controller algorithm, optimal in terms of the quadratic functional of the tracking error and its derivative over time, by numerical solution of the Riccati matrix equation.

At the second stage of the research experiment, simulation of the guided motion of UAV with different weight load in the presence of disturbances in the form of mine air flows and various obstacles was carried out.

To form the motion route, a fragment of mine workings system of "Mine named after V.D. Yalevsky" of JSC SUEK-Kuzbass was used, shown in Fig. 4, *a*, where the main markers of the supposed route of the UAV, the motion direction, the temperature at the nodal points, as well as the location of "GRANCH" base stations [4] are shown.

Based on the route prototype, a test polygon for the trajectory planner, shown in Fig. 4, *b*, which contains the main features of the actual fragment of the topology of mine workings. When simulating, the velocity of the mine airflow was varied;

The UAV flight speed in the absence of disturbing air flow was taken constant of 6 m/s and 10 m/s.

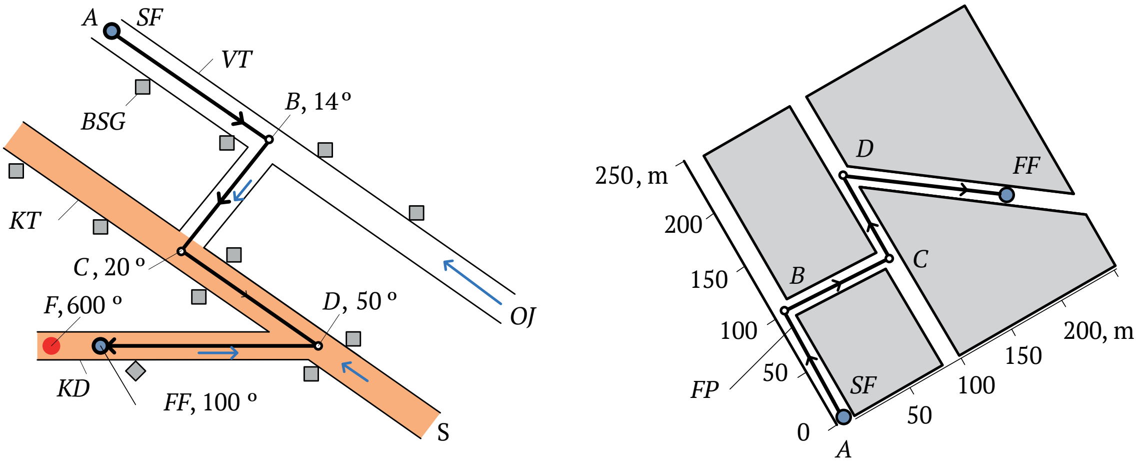
Under these conditions, the following key simulation experiments were conducted:

- a flight over the test polygon without load, disturbances and obstacles;
- a flight without load, disturbance, but with obstacles;
- a flight without load, but with counter mine air flow;
- a flight with payload without air disturbances;
- a flight with at payload with counter mine air flow;
- an emergency flight braking with different payload.

In each of the experiments, the deviations of the actual trajectory from the planned trajectory were quantitatively recorded along all three axes, by which the mean deviation of the actual trajectory from the planned one and the maximum deviation of the actual trajectory from the planned one during the flight period were calculated.

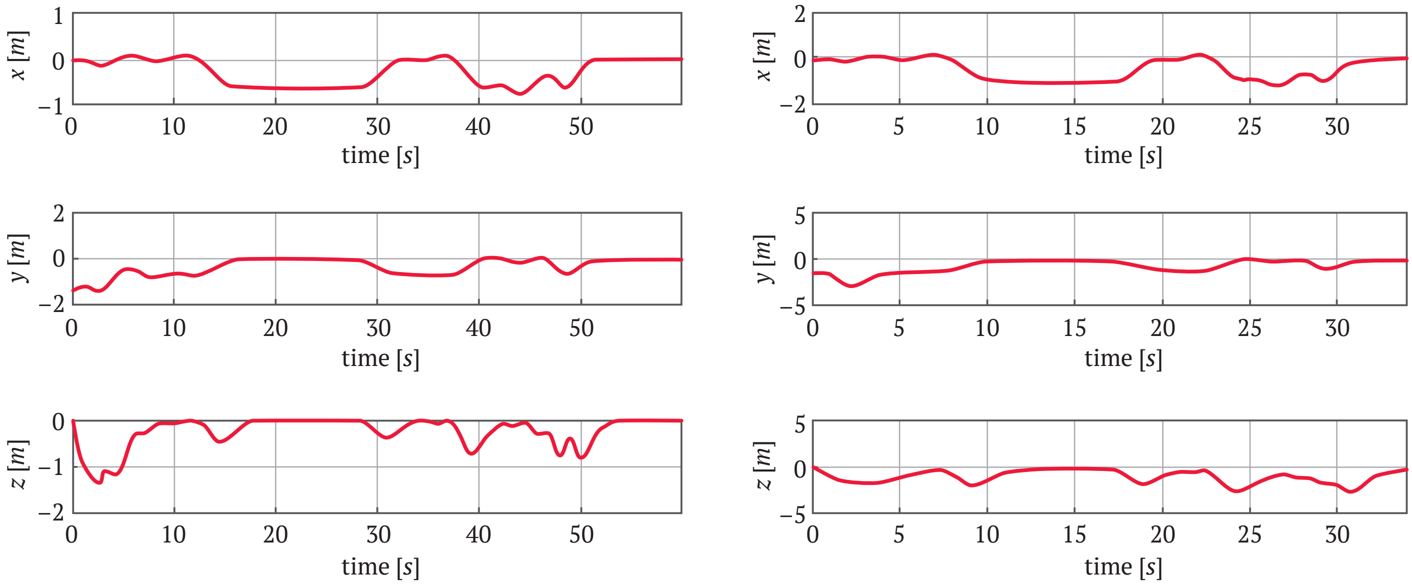
5. The simulation study findings

Fig. 5, *a, b* presents the experimental curves reflecting the quality of control when flying the route, when the UAV was loaded or not loaded, when it flied at its maximum speed or at 60 % of it. The simulation of the motion in projection on the horizontal plane revealed undershooting in the *x, y, z* coordinates.



a *b*

Fig. 4. a – fragment of mine workings of the Mine named after V.D. Yalevsky of JSC "SUEK-Kuzbass", *b* – test polygon for the trajectory planner:
SF – flight start; *FF* – flight finish; *F* – fire; *S* – smoke; *OJ* – outgoing jet; *VT* – ventilation shaft; *KT* – conveyor shaft; *KD* – conveyor drift; *BSG* – base stations "GRANCH"; *A, B, C, D* – points of trajectory bend and air temperature values in them



a *b*

Fig. 5. Coordinate deviations from the trajectory when moving:
a – at a speed of 10 m/s without payload; *b* – at a speed of 6m/s with payload

The maximum magnitude of deviation in case *a* was slightly more than 0.5 m for the *x*-coordinate, up to 1.0 m for the *y*-coordinate, up to 1.2 m for the *z*-coordinate. In case *b*, the maximum deviation value was up to 1.0 m for the *x* coordinate, up to 1.5 m for the *y* coordinate, up to 1.3 m for the *z* coordinate.

Fig. 6 shows the change in the modulus magnitude of the current deviation vector of the actual trajectory from the preset one at the UAV speed of 6 m/s with payload. The maximum deviation value was just over 1 m in the first seconds of flight, and later on the deviation did not exceed 0.3 m.

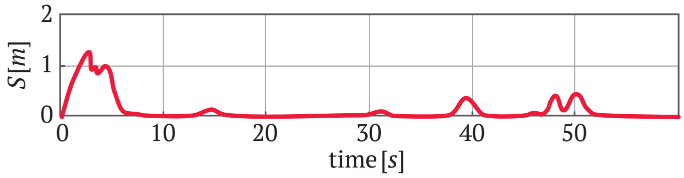


Fig. 6. Time dependence of modulus of the current deviation vector of the actual trajectory from the preset one at UAV speed of 6 m/s with payload

The average deviation from the planned trajectory when flying at a speed of 10 m/s with the payload mass up to 0.6 kg did not exceed 1 m, and the maximum deviation was unacceptably large. When flying at 6 m/s with payload mass up to 0.6 kg the average deviation did not exceed 0.3 m, and the maximum deviation, 1.2 m.

5.1. A flight with mine air disturbances

The task of the simulation experiment was to check the sensitivity of the motion control system to the disturbance in the form of mine air flow. The maximum deviation of the UAV from the preset trajectory depending on the velocity of the disturbing air flow, the limiting value of the disturbing air flow velocity, which caused unacceptable UAV deviation from the trajectory for two cases, with load and without load, were determined.

The findings of the simulation of the routing against the disturbing mine air flow showed that the control system allowed the UAV with payload of 0.6 kg to withstand the counter air flow up to 8 m/s.

5.2. An emergency braking

Studies of unmanned aerial vehicle motion in a mine at post-accident mine workings conditions require testing not only the maneuverability when flying around various obstacles, but, also in case of impossibility of flying around, testing for emergency braking.

It was found that with payload mass of 0.6 kg the braking length did not exceed 6 m at the UAV speed of 6 m/s and did not exceed 12 m at the speed of 10 m/s.

Conclusion

The implemented research allowed to create the structure of the automatic control system for autonomous motion of an UAV based on the mathematical simulation of the UAV motion adapted to mine conditions, in which the shortest motion trajectory was formed algorithmically as a function of the target, coordinate transformations were performed, and control actions on the UAV drives were formed to perform motion along the planned route.

The performed computer simulation of UAV target trajectory motion allowed to determine the following: controlling quality indicators — coordinate deviations, average and maximum estimates of these deviations, which did not exceed permissible values under the mine conditions, limiting values of the UAV payload, range of external disturbing mine air flow velocity values. The study findings confirmed the operating capability of the developed system for automatic motion control.

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ЭНЕРГЕТИКА, АВТОМАТИЗАЦИЯ И ЭНЕРГОЭФФЕКТИВНОСТЬ

Научная статья

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Оценка эксплуатационной надежности электроснабжения развивающихся участков добычи руд на высокогорном руднике

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Аннотация

Приоритетными направлениями развития добычи вольфрамо-молибденового сырья являются освоение новых месторождений и модернизация существующих горнодобывающих предприятий с целью повышения их эффективности. При модернизации горных производств внимание уделяется всем технологическим процессам и инженерным системам, трансформация которых должна быть направлена на повышение экономических показателей, в том числе и надежности. Исследования систем электроснабжения горных предприятий и режимов их работы являются актуальной задачей, так как этот класс инженерного обеспечения горных работ, вспомогательных процессов и объектов определяет конкурентоспособность предприятия в целом. В ходе исследований проведена оценка эксплуатационной надежности электроснабжения развивающихся участков добычи руд на высокогорном руднике Тырныаузского месторождения (Кабардино-Балкарская Республика). Установлено, что при существующей схеме раздельного питания сетей напряжением 6 кВ от подстанций полные токи замыкания на землю для этих сетей соответственно равны: I_{030} = 17,5 А и I_{0116} = 12,2 А и достигают предельных значений, при которых возможно их отключение разъединителями (предельный ток составляет 20 А). Рекомендована формула полного тока замыкания на землю в подземных сетях напряжением 6 кВ. Показано, что надежность электроснабжения на указанном руднике, а также уровень опасности поражения электрическим током в электроустановках в значительной степени зависят от количества однофазных замыканий на землю. На основе теоретических и экспериментальных исследований разработаны решения по улучшению эксплуатации электрических сетей, учитывающих структуру и режимы работы высокогорного рудника.

Ключевые слова

горнодобывающие предприятия, рудники, электроснабжение, режим работы, эффективность, надежность, безопасность, замыкания, изоляция, проводимость, Тырныаузское месторождение, Кабардино-Балкарская Республика

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POWER ENGINEERING, AUTOMATION, AND ENERGY PERFORMANCE

Research article

Assessment of operational reliability of power supply to developing ore mining areas at a high-altitude mine

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Abstract

Development of new deposits and modernization of existing mining enterprises in order to improve their efficiency are the priorities for the development of tungsten-molybdenum resource sector. In the modernization of mining productions, attention is paid to all technological processes and engineering systems, transformation of which should be aimed at improving economic performance, including reliability. Research of power supply systems of mining enterprises and modes of their operation is an urgent task, as this type of engineering support of mining works, auxiliary processes and facilities determines competitiveness of an enterprise as a whole. In the course of research, an assessment of operational reliability of power supply of developing ore mining areas at a high-altitude mine of the Tyrnyauz deposit (Kabardino-Balkaria) was performed. It was found that under the existing scheme of separate power supply of 6 kV networks from substations, the full ground fault currents for these networks are respectively: I_{030}=17.5 A and I_{011b} =12.2 A and reach the limiting values at which it was possible to disconnect them by disconnectors (the limiting current was 20 A). A formula for total ground-fault current in 6 kV underground networks was recommended. It was shown that reliability of power supply in the above mine, as well as the level of danger of electric shock in electrical installations depended largely on the number of single-phase ground faults. On the basis of the theoretical and experimental research, the solutions to improve the operation of electrical networks, taking into account the structure and modes of operation of a high-altitude mine were developed.

Keywords

mining enterprises, mines, electric power supply, efficiency, operating regime, reliability, safety, ground faults, insulation, conductivity, Tyrnyauz deposit, Kabardino-Balkarian Republic

For citation

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Введение

Месторождения вольфрамо-молибденовых руд всегда привлекали к себе внимание, так как во многом определяют конкурентоспособность целых отраслей экономики. Приоритетными направлениями развития добычи любого минерального сырья, как правило, являются освоение новых месторождений и модернизация существующих горнодобывающих предприятий с целью повышения их эффективности. При модернизации горных производств внимание уделяется всем технологическим процессам и инженерным системам, трансформация которых должна быть направлена на повышение экономических показателей, в том числе и надежности [1].

Горные предприятия являются в значительной степени энергоемкими, так как многие технологические процессы связаны с добычей, переработкой горной массы, ее перемещением, созданием безопасных условий для ведения технологических процессов, а именно проветриванием, откачиванием рудничных вод и т.д. Поэтому исследования систем электроснабжения горных предприятий и режимов их работы являются актуальной задачей [2]. Исследования проводились на системе (системах) электроснабжения рудника Тырныаузского месторождения (Кабардино-Балкарская Республика).

Электрические сети напряжением 6 кВ рассматриваемого рудника развивались постепенно. По мере увеличения горных работ и при ожидаемом их объеме они не смогут обеспечивать надежное электроснабжение развивающихся участков подземной и открытой добычи на руднике. Поэтому повышение электропотребления, необходимость обеспечения более высокой надежности и гибкости системы электроснабжения горных работ и безопасности электрических установок в сетях различных напряжений при расширении фронта горных работ как подземным, так и открытым способами – важная научно-практическая задача, требующая эффективного решения [3–5].

Объектом исследования являются системы электроснабжения развивающихся участков добычи руд на высокогорном руднике – как подземным, так и открытым способами, в том числе при подземной добыче руд в энергонарушенных массивах [6].

Одним из самых проблемных мест является эксплуатация кабельных сетей напряжением 6 кВ на высокогорном руднике. Сложности связаны с возможным одновременным выходом из строя кабельной изоляции на различных участках кабельной сети при коротком замыкании (КЗ) на землю на одном из ее участков. Такие одновременные повреждения изоляции кабелей в нескольких местах при КЗ на землю могут быть следствием плохого ее состояния или значительных перенапряжений в двух других фазах при замыкании на землю третьей фазы.

Цель исследования – оценка эксплуатационной надежности электроснабжения развивающихся участков добычи руд на высокогорном руднике для обеспечения более высокой гибкости системы электроснабжения горных работ, а также безопасности электрических установок в сетях различных напряжений при расширении фронта горных работ как подземным, так и открытым способами.

Для достижения поставленной цели необходимо решить следующие задачи.

1. Выполнить измерение токов КЗ на землю и разработать методики их расчета, а также мероприятия по их ограничению.

2. Исследовать межфазные токи КЗ для проверки возможности селективной работы релейной защиты элементов электрических сетей.

Методология исследований включает в себя комплекс методов исследования прямого замыкания на землю для определения полной проводимости изоляции, например, метод, использующий искусственное смещение нейтрали сети относительно земли от постороннего источника тока, а также метод наложения постоянного оперативного тока на рабочую сеть высокого напряжения [7–10].

В последние годы ведется активная работа по восстановлению и вводу в эксплуатацию Тырныаузского месторождения. По оценкам «Ростеха», балансовые запасы месторождения составляют 209,5 тыс. т триоксида вольфрама (содержание – 0,432 %) и 36,6 тыс. т молибдена (содержание – 0,076 %). Проект возрождения ГОКа был включен в «Инвестиционный план первоочередных и перспективных проектов, реализуемых и планируемых к реализации в Северо-Кавказском федеральном округе на период до 2025 года». В работе приведены результаты функционирования месторождения, которые планируется использовать при вводе месторождения в опытную эксплуатацию в ближайшие годы.

Тырныаузское месторождение разрабатывается комбинированным способом: основную долю в добыче составляет подземный способ и около 20% добывается открытым способом. Технологический процесс добычи руды подземным способом состоит из операций бурения, взрывания, погрузки и доставки руды из очистных забоев, откатки к стволу шахты и выдачи руды не поверхность. Транспортировка руды на эксплуатационных горизонтах производится внутришахтным транспортом до рудоспуска электровозами в вагонетках, а на остальных горизонтах – автосамосвалами. Руда из очистных блоков транспортируется до капитальных рудоспусков и перепускается на горизонт «Штольня Главная». Основными стационарными установками рудника являются: вентиляторные, пневматические, подъемные, водоотливные. Кабельная сеть рудника выполнена кабелями типа АСБ, ААБ и СБ.

Электрические сети напряжением 6 кВ высокогорного рудника состоят из воздушных и кабельных участков. Воздушные электрические линии напряжением 6 кВ представлены двумя фидерами Ф-61 и Ф-63 от подстанции (ПСТ) «Эльбрус», Республика Северная Осетия-Алания (РСО – Алания), до ПСТ № 11б, а также отпайками от этих фидеров к ПСТ № 30 и другим подстанциям по пути следования фидеров (рис. 1). Фидеры Ф-61 и Ф-63 выполнены сталеалюминиевыми проводами марки АС сечением 95 мм^2 на деревянных опорах. На отдельных труднопроходимых участках воздушные сети выполнены кабельными вставками с соизмеримой допустимой токовой нагрузкой. Кабельная сеть рудника выполнена бронированными кабелями с бумажной пропитанной изоляцией в свинцовой оболочке или с отдельно освинцованными жилами, которые по поверхности высокогорного рудника проложены в траншеях и представлены кабелями типа АСБ, ААБ и СБ с медными и алюминиевыми жилами. Кабели в подземных выработках проложены открыто.

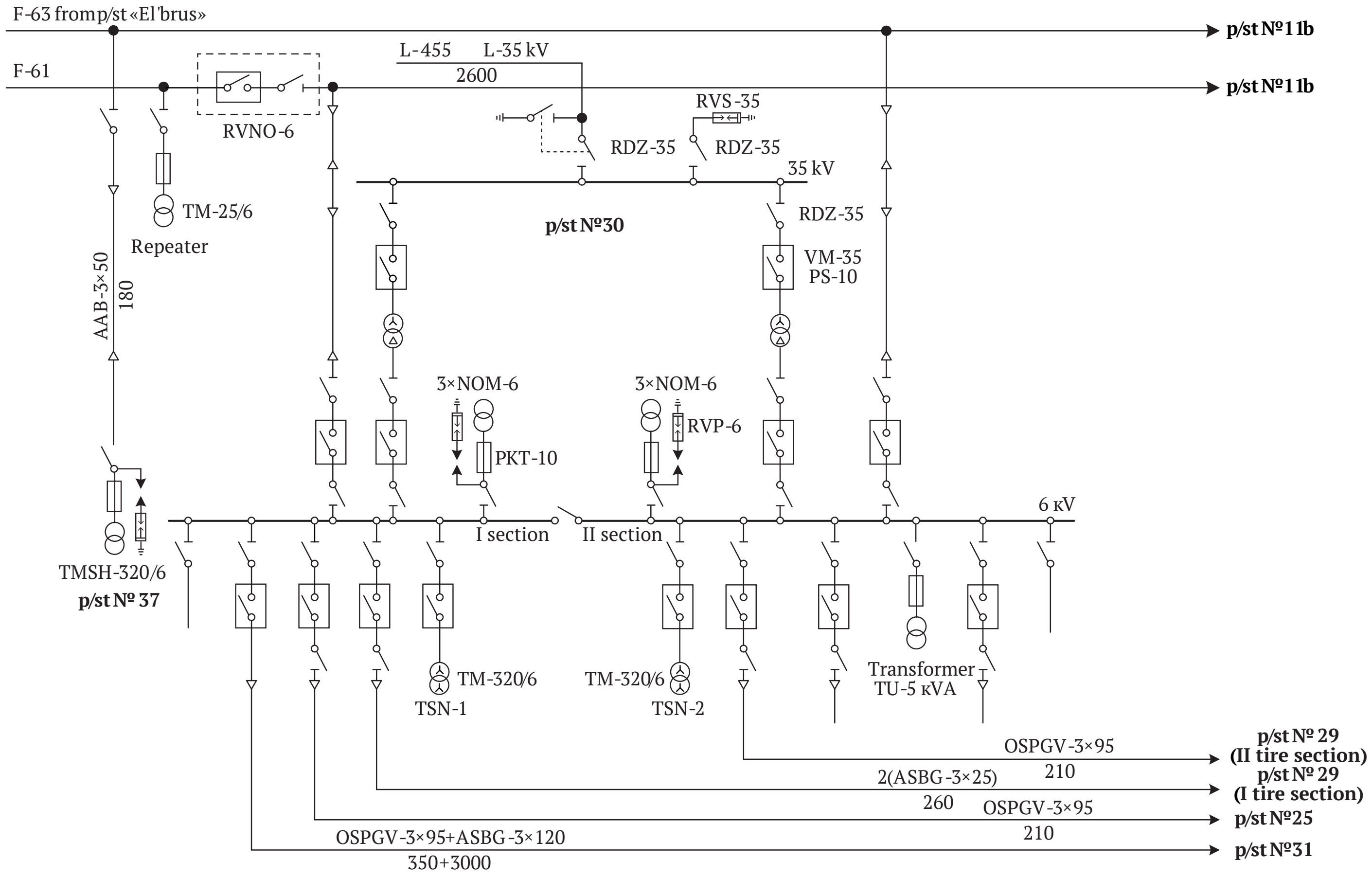

Рис. 1. Фрагмент схемы электроснабжения высокогорного рудника

Fig. 1. Fragment of power supply schematic diagram of mountain mine

Отдельные участки кабельной сети соединяются при помощи типовых и собственного изготовления переходных кабельных муфт с заливкой типовой битумной массой. Для питания сетей напряжением 6 кВ рудника используются два трансформатора ТМ-3200, установленных на ПСТ №30. Оба трансформатора подключены к одной воздушной линии напряжением 35 кВ от ПСТ «Эльбрус» – ЛЭП-455. Распределительные устройства ПСТ № 30 и 11б, а также остальных подземных и поверхностных подстанций скомплектованы из комплектных распределительных ячеек типа РВН-6 (РВНО-6) с масляными выключателями ВМБ-10 или ячеек с разъединителями и высоковольтными предохранителями для подключения трансформаторов собственных нужд подстанций и измерительных трансформаторов напряжения типа НОМ-6, НТМИ-6, НТМК-6.

Ячейки оборудованы приводами и масляными выключателями типа ПРБА со встроенными двумя токовыми реле прямого действия типа РТМ без выдержки времени или типа РТВ с выдержкой времени. На ряде ячеек ввода питающих линий на подстанцию (11б, 30, ствол, главные вентиляторы и др.) защита выполнена реле косвенного действия типа ЭТ-520 без выдержки времени и типа ИТ-80 с выдержкой времени, воздействующими на привод выключателя через отключающую катушку.

Подстанции верхних горизонтов при наличии двух вводов на подстанцию не имеют секционирования шин РУ-6 кВ. Отсутствие секционирования шин не затрудняет работу максимальной токовой защиты (МТЗ) как на питающих подстанцию линиях, так и на отходящих от них линиях. Надежность системы электроснабжения этих горизонтов при наличии двух линий питания мало отличается от надежности системы с одной линией. Ограниченная пропускная способность фидеров Ф-61 и Ф-63, а также сложная схема питания подстанций и ввода резерва затрудняют бесперебойное электроснабжение ответственных компрессорных и вентиляционных установок, преобразовательных подстанций и т.д. [11, 12].

Состояние изоляции всей сети в целом и отдельных ее элементов на руднике, как и токи КЗ и возможные перенапряжения в сети, исследованиям не подвергались. Надежность электроснабжения на руднике, а также опасность поражения электрическим током в электроустановках напряжением 6 кВ в значительной степени зависят от количества однофазных замыканий на землю (ОЗЗ) и связанных с ними перенапряжений в сети, а также возможности перехода однофазных замыканий в многофазные. ОЗЗ с длительно протекающими токами до 30 А могут вызвать на заземленном оборудовании опасный для человека потенциал от 60 В ($R_{\text{з}} = 20$ Ом) до 300 В ($R_{\text{з}} = 10$ Ом).

Решение поставленных в работе задач позволит выработать решения по улучшению эксплуатации электрических сетей, более полному использованию существующих сетей и обеспечению безопасных условий обслуживания электроустановок сетей напряжением 6 кВ высокогорного рудника.

Общая характеристика электрических сетей напряжением 6 кВ высокогорного рудника

Электрические распределительные сети напряжением 6 кВ рассматриваемого рудника получают питание от двух источников: линии 35 кВ и двух воздушных линий 6 кВ от подстанции «Эльбрус». Воздушная ЛЭП напряжением 35 кВ длиной 2,6 км выполнена проводом АС-3×95 на деревянных опорах и подходит к ПСТ №30, на которой установлены два трансформатора мощностью 3200 кВА каждый, питающих две секции шин напряжением 6 кВ. Секции 6 кВ ПСТ №30 секционированы разъединителем.

Воздушные ЛЭП напряжением 6 кВ от ПСТ «Эльбрус»: Ф-61 и Ф-63 – подходят к ПСТ №11б, которые на отдельных участках имеют кабельные вставки одинаковой с воздушными линиями пропускной способностью, от обеих ЛЭП имеются отпайки к шинам 6 кВ ПСТ №30. Шины 6 кВ ПСТ №11б не секционированы. Сеть рудника имеет большую разветвленность и значительную протяженность (общая протяженность кабельной сети 6 кВ $l_{cable} = 45$ км и воздушной lair = 11 км, число подключенных подстанций равно 32). Распределительная сеть 6 кВ проходит по поверхности рудника, по штольням и вертикальным выработкам. Кабельная сеть выполнена кабелем с алюминиевыми и медными жилами, в свинцовой и алюминиевой оболочках с бумажной пропитанной изоляцией на горизонтальных участках и с осушенной изоляцией на вертикальных участках.

Методы исследования состояния изоляции электрических сетей напряжением 6 кВ

Полная проводимость изоляции сети определяет полный ток ОЗЗ и наоборот согласно выражениям [9, 10]:

$$y_0 = \frac{I_0}{U_{ph}}; \quad I_0 = U_{ph} y_0, \tag{1}$$

где y_0 – полная проводимость изоляции сети, Ом$^{-1}$; I_0 – полный ток замыкания на землю, А; U_{ph} – напряжение фазы (phase) сети относительно земли, В.

Полная проводимость изоляции представляется двумя составляющими: активной – g_0 и реактивной – b_0:

$$y_0 = \sqrt{g_0^2 + b_0^2}. \tag{2}$$

Соотношение между g_0 и b_0 зависит от качества и состояния изоляционного материала кабеля, т.е. определяется тангенсом угла диэлектрических потерь:

$$\operatorname{tg} \delta_0 = \frac{b_0}{g_0}.$$

Метод прямого замыкания на землю

Полная проводимость изоляции y_0 наиболее просто и точно определяется через ток ОЗЗ I_0 и напряжение фазы сети относительно земли U_{ph}. *Значение тока ОЗЗ* I_0 определяется при замыкании одной из фаз сети на землю через амперметр (или трансформатор тока) по схеме (рис. 2) [9, 10]. Такой способ измерения прост, требует минимального времени на выполнение измерений, простейшую измерительную аппаратуру (амперметр, трансформатор тока).

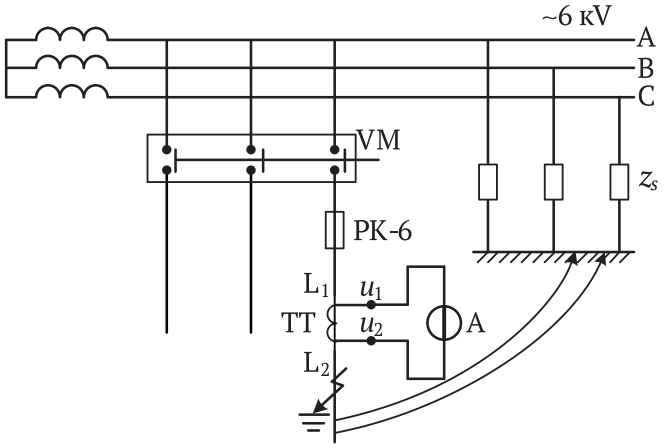

Рис. 2. Метод прямого замыкания на землю
Fig. 2. Method of direct ground fault

Основным недостатком этого метода является повышение напряжения на двух неповрежденных фазах относительно земли и возможность перехода однофазного замыкания в многофазное замыкание на землю, что негативно сказывается на электробезопасности для условий горных предприятий высокогорья, где необходимо быстрое отключение повреждений для недопущения прекращения электроснабжения потребителей в труднодоступных горных районах.

Для обеспечения безопасности при измерении тока ОЗЗ замыкание производится через предохранитель типа ПК-6 (РК-6), а установка максимальной токовой защиты на масляном выключателе ячейки, на которой производится замыкание, частично или полностью выводится. Как показывает опыт аналогичных исследований, наличие в замыкаемой цепи предохранителя указанного типа обеспечивает безопасность проведения измерения тока замыкания на землю – вследствие быстродействия его срабатывания. При возможном многофазном замыкании отключение «земли» производится предохранителем раньше, чем максимальной токовой защитой при полностью выведенной выдержке времени. Максимальная токовая защита при измерении резервирует работу предохранителя [9, 10, 13–15].

Общая протяженность электрически связанной сети 6 кВ высокогорного рудника превышает 56 км, на которых более 45 км кабельной и 11 км воздушной сети. Ожидаемый ток ОЗЗ при параллельной работе всей сети и трансформаторов можно определить по эмпирической формуле:

$$I_0 = U_l\left(\frac{l_{air}}{390} + \frac{l_{cable}}{10} + \frac{N_{el}}{500}\right), \tag{3}$$

где U_l – линейное (line) напряжение сети, кВ; l_{air}, l_{cable} – протяженность воздушной (air) и кабельной (cable) сети, км; N_{el} – число подключенных к сети напряжением 6 кВ отдельных элементов (elements): трансформаторных подстанций, двигателей 6 кВ, ячеек 6 кВ и т.д.

При измерении тока ОЗЗ по схеме, приведенной на рис. 2, необходимо: установить предохранитель ПК-6, I_{in} = 15 А; полностью вывести выдержку времени максимальной токовой защиты; установку максимальной токовой защиты по первичному току принять равной 30 А. Для измерения полной проводимости изоляции в сетях 6 кВ высокогорного рудника могут быть использованы и другие методы [13–15].

Косвенные методы определения полной проводимости включали: а) метод замыкания фазы на землю через большое сопротивление, дроссель; б) метод искусственного смещения нейтрали при помощи постороннего источника тока. Оба указанных метода измерения полной проводимости сети основываются на методе, предложенном проф. Л. И. Сиротинским [13, 14]. Согласно первому методу одна из фаз сети замыкается на землю через сопротивление (дроссельную катушку). Величина возникшего при этом смещения нейтрали измеряется вольтметром, включенным между землей и искусственно созданной нулевой точкой сети, полученной путем включения в сеть 6 кВ трансформатора напряжения типа НТМИ-6, НТМК-6. Одновременно измеряется ток, протекающий через дроссельную катушку. По величине напряжения смещения нейтрали и величине тока через дроссельную катушку определяется ток ОЗЗ по формуле:

$$I_0 = I_{cc}\frac{U_{ph}}{U_{nbv}}, \tag{4}$$

где I_0 – ток ОЗЗ, А; I_{cc} – ток, протекающий через дроссельную катушку, А; U_{nbv} – напряжение смещения нейтрали, вызванное включением дроссельной катушки, кВ.

Метод проф. Л.И. Сиротинского предполагает, что система, в которой производится измерение тока ОЗЗ, симметрична; векторы фазных напряжений равны между собой и сдвинуты на 120°, естественное напряжение смещения нейтрали сети относительно земли U_0 = 0. При замыкании на землю одной из фаз сети через дроссельную катушку происходит смещение нейтрали сети, величина которого зависит от фазного напряжения сети и параметров дроссельной катушки. Однако в практике эксплуатации электрических сетей из-за разной проводимости изоляции фаз относительно земли симметрия фазных напряжений нарушается, что приводит к появлению естественного смещения нейтрали сети U_0 относительно земли. Включение дополнительной проводимости – дроссельной катушки – в одну из фаз сети приводит к изменению величины U_0 до нового значения, обусловленного сложением векторных значений величин $\bar{U}_0$ и $\bar{U}_{nbv}$ – напряжения смещения нейтрали сети относительно земли, вызываемого включением дросселя. Понимание процессов определения электрических величин представляется на векторной диаграмме [13, 14].

Исследования показали, что метод проф. Л. И. Сиротинского применим и при наличии естественного смещения нейтрали, если в знаменатель формулы (4) подставлять только ту часть результирующего напряжения смещения нейтрали $\bar{U}_{nbv}$, которая обусловлена включением дросселя во все фазы сети. Если естественное смещение нейтрали по результатам измерений равно U_0, а после поочередного включения дросселя в разные фазы сети оказалось соответственно равным $\bar{U}_{01}$, $\bar{U}_{02}$, $\bar{U}_{03}$, то величина напряжения смещения нейтрали сети, обусловленная включением дросселя $\bar{U}_{nbvi}$, будет равна:

$$\bar{U}_{nbvi} = \bar{U}_{0i} - \bar{U}_0. \qquad (5)$$

Если принять за средние значения величины:

$$I_{nbv.average} = \frac{I_{nbv1} + I_{nbv2} + I_{nbv3}}{3}, \text{ А};$$

$$U_{ph} = \frac{U_{ph1} + U_{ph2} + U_{ph3}}{3}, \text{ кВ};$$

$$U_{nbv.average} = \frac{U_{nbv1} + U_{nbv2} + U_{nbv3}}{3}, \text{ кВ},$$

то формула (4) будет записана в виде:

$$I_0 = I_{nbv.average} \frac{U_{ph}}{U_{nbv.average}}, \text{ А}. \qquad (6)$$

При отсутствии естественного смещения нейтрали в сети, $U_0 = 0$, на основании формул (4), (5): $U_{nbvi} = U_{0i} = U_{nbv}$, выражение (6) превращается в формулу (4). Очевидная сложность работы с векторными величинами и громоздкость графических построений затрудняют практическое применение этого метода в представленном виде. На практике находит применение несколько упрощенный метод расчета, значительно сокращающий объем и сложность измерений, позволяющий вести вычисления не с векторами, а со скалярными величинами. Несколько повышенная погрешность расчетов (до 15 %) не оказывает существенного влияния на практические вычисления [16]. При расчете тока замыкания на землю I_0 по упрощенному методу вместо величины напряжения смещения нейтрали $U_{nbv.average}$ в формулу (6) подставляется среднеарифметическое значение напряжения смещения $U_{0average}$, замеренное при включении дросселя поочередно в три фазы сети, кВ:

$$U_{0average} = \frac{U_{01} + U_{02} + U_{03}}{3}.$$

Относительная ошибка расчетов β равна:

$$U_{0average} = \frac{U_{0average} - U_{nbv.average}}{U_{nbv.average}} \cdot 100\%. \qquad (7)$$

При $U_0/U_{0average} \leqslant 0{,}65$ β не превышает 15%.

При замене $U_{nbv.average} = U_{0average}$ зависит от отношения: $U_0/U_{0average}$ и при $U_0/U_{0average} \leqslant 0{,}65$ не превышает 15 %. График зависимости $\beta = f(U_0/U_{0average})$ представлен на рис. 3.

Измеряется фазное напряжение сети U_{ph} относительно земли по показаниям вольтметров V_1, V_2 и V_3 при полной обмотке трансформаторов напряжения НТМИ-6 (НТМК-6). Измеряется естественное смещение нейтрали U_0 при помощи измерительного трансформатора типа НОМ-6 и вольтметра V_5, включенного во вторичную обмотку НОМ-6. Кроме того, значение можно измерить с помощью вольтметра V_5 или V_4, включенного в обмотку по схеме открытого треугольника. Дроссель поочередно включается между фазами 1, 2, 3 и землей, одновременно измеряется напряжение смещения нейтрали U_{01}, U_{02}, U_{03} (V_5), и по показанию амперметра (А) определяется значение тока в дросселе I_{cc1}, I_{cc2} и I_{cc3}. Находятся величины $U_{0average}$ и $I_{nbv.average}$ и отношение $U_0/U_{0average}$.

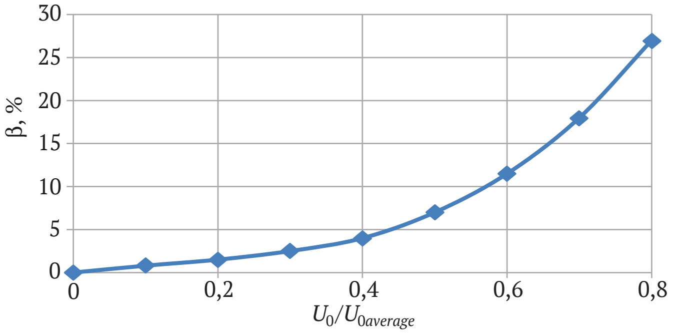

Рис. 3. График зависимости $\beta = f(U_0/U_{0average})$

Fig. 3. Dependence graph $\beta = f(U_0/U_{0average})$

По результатам представленной зависимости находится относительная погрешность расчетов β исходя из отношения $U_0/U_{0averag}$ (см. кривая, рис. 3) и вычисляется величина смещения нейтрали, обусловленная только включением дросселя, кВ:

$$U_{nbv.average} = \frac{U_{0average}}{1 + \dfrac{\beta\%}{100}}. \qquad (8)$$

Рассчитывается величина полного тока ОЗЗ и полной проводимости изоляции фаз сети относительно земли:

$$I_0 = I_{nbv.average} \frac{U_{ph}}{U_{nbv.average}}; \quad y_0 = \frac{I_0}{U_{ph}} = \frac{I_{nbv.average}}{U_{nbv.average}}.$$

Для определения тока ОЗЗ и полной проводимости изоляции фаз сети относительно земли предлагается следующий порядок проведения измерений (рис. 4).

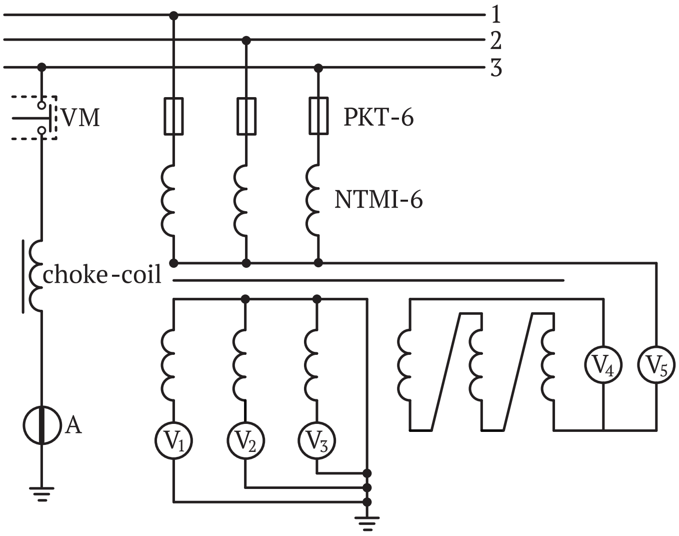

Рис. 4. Порядок проведения измерений для определения тока ОЗЗ и полной проводимости изоляции фаз сети относительно земли

Fig. 4. Measurement procedure for determining the single phase-to-ground fault current and total conductivity of insulation of network phases relative to the ground

В процессе эксплуатации электрических сетей значительный интерес представляет регистрация уровня изоляции сети в течение определенного времени. В условиях разветвленной сети рассматриваемого рудника при большом количестве подстанций несомненный интерес представляет характер изменения проводимости сети в течение времени при различного рода оперативных переключениях, изменении схемы и нагрузки в сети. Регистрацию изменения проводимости изоляции сети во времени можно осуществить методом, использующим искусственное смещение нейтрали сети относительно земли от постороннего источника тока. По этому методу полная проводимость изоляции сети вычисляется с использованием результатов двух измерений, выполненных в данной сети по схеме (рис. 5), по формуле, Ом$^{-1}$:

$$y_0 = \frac{I_0' - I_0''}{E_0' - E_0''}. \tag{9}$$

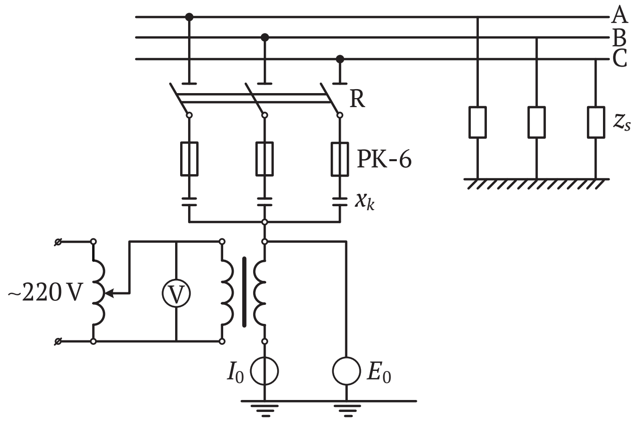

Рис. 5. Метод измерения проводимости изоляции, использующий искусственное смещение нейтрали сети относительно земли от постороннего источника тока

Fig. 5. Method for measuring insulation conductivity using artificial displacement of network neutral relative to ground from an external supply source of current

Для получения достоверных результатов y_0, с относительной погрешностью до 5 %, необходимо, чтобы величина меньшей ЭДС постороннего источника питания E_0'' была больше 60 % напряжения естественного смещения нейтрали сети U_0.

Второе измерение в сети производится по той же схеме, что и при значении ЭДС постороннего источника $E_0' = 1{,}5U_0$, что дает возможность более точно зафиксировать разность токов $I_0' - I_0''$, полученных соответственно при первом и втором измерениях. В связи с тем что внутреннее сопротивление источника смещения нейтрали мало, значения E_0' и E_0'' при измерении можно заменить соответственно напряжениями U_0' и U_0'', замеряемыми с помощью приборов.

По графику, представленному на рис. 6, определяется относительная ошибка σ, %, при соответствующих значениях коэффициента K и угла α. Коэффициент K показывает отношение тока искусственного смещения I_0' к току естественного смещения I_0, который можно измерить замыканием нулевой точки системы через амперметр на землю или рассчитать приближенно по формуле [14], А:

$$I_0 \approx U_0 \cdot y_0'. \tag{10}$$

Уточненное значение проводимости изоляции y_0 определяется по выражению:

$$y_0 = \frac{y_0'}{1 - \frac{\sigma\%}{100}}. \tag{11}$$

Ток замыкания на землю определяется по формуле, А:

$$I_0 = U_{ph} y_0. \tag{12}$$

Таким образом, надежность работы отдельных участков и всей сети в целом определяется следующими факторами:

– состоянием изоляции электрической (преимущественно кабельной) сети, наличием постоянного

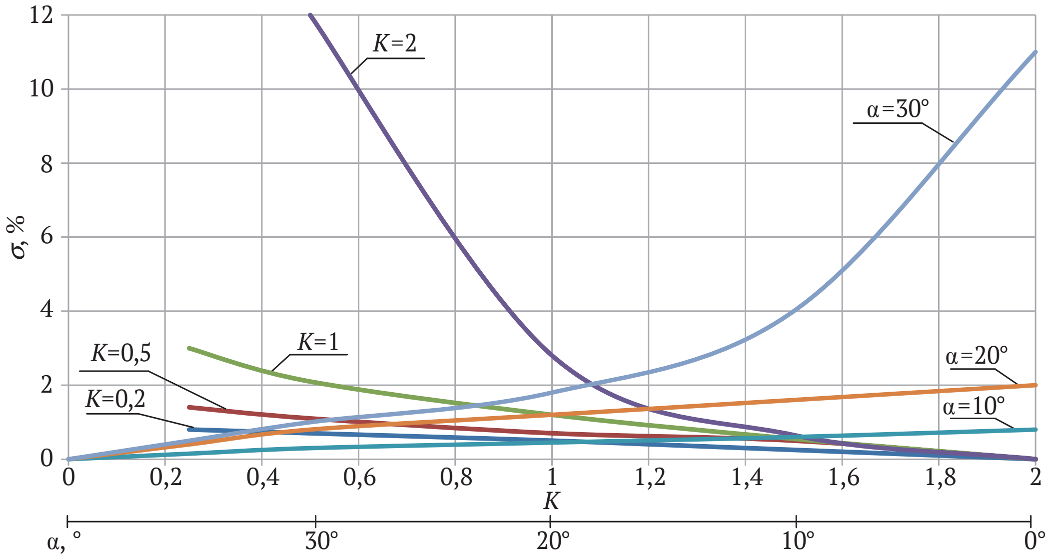

Рис. 6. График определения относительной ошибки σ, %, при соответствующих значениях коэффициента K и угла α

Fig. 6. Graph for determining the relative error σ, %, at the corresponding values of K coefficient and angle α

автоматического контроля изоляции, способов определения мест повреждения и быстрого восстановления электрической прочности изоляции;

– существующей схемой электроснабжения и общей протяженностью сети, питающейся от одного источника питания, определяющей величину тока ОЗЗ и внутренних перенапряжений в сети и вероятность их перехода в многофазные КЗ;

– величинами многофазных КЗ в сети, наличием на подстанциях надежной коммутационной отключающей аппаратуры в сочетании с быстродействующей селективной защитой;

– резервированием отдельных элементов сети, определяемым схемой электроснабжения рудника.

Заключение

В результате проведенных исследований дана оценка эксплуатационной надежности электроснабжения развивающихся участков добычи руд на высокогорном руднике для обеспечения более высокой гибкости системы электроснабжения горных работ, а также повышения надежности и безопасности электрических установок в сетях различных напряжений при расширении фронта горных работ как подземным, так и открытым способами. Представленные результаты исследований имеют важное значение при реализации образовательных программ подготовки горных инженеров [17].

Выводы:

1. Установлено, что при существующей схеме раздельного питания сетей напряжением 6 кВ от подстанций полные токи замыкания на землю для этих сетей соответственно равны: I_{030} = 17,5 А и $I_{011\text{б}}$ = 12,2 А и достигают предельных значений, при которых возможно их отключение разъединителями (предельный ток составляет 20 А).

2. Определены полные сопротивления изоляции сетей, которые равны: z_{030} = 198 Ом и $z_{011\text{б}}$ = 284 Ом и являются низкими для сетей напряжением 6 кВ. При этом омическое сопротивление изоляции трех фаз в среднем не превышает: $R_{0.average} = 343 \cdot 10^3$ и составляет около 100 Ом/В или на одну фазу 300 Ом/В, что значительно ниже рекомендуемой для электрооборудования величины 1000 Ом/В.

3. Обосновано, что при полной проводимости сети $y_0 = 5050 \cdot 10^{-6}$ Ом$^{-1}$ и $y_0 = 3520 \cdot 10^{-6}$ Ом$^{-1}$ на три порядка выше, чем омическая проводимость $g_0 = 2{,}8 \cdot 10^{-6}$ Ом$^{-1}$, которой можно пренебречь и считать, что емкостная проводимость b_0 равна полной проводимости изоляции y_0: $b_0 = y_0 = 1/x_0 = \omega \cdot C_e \cdot l_{cable}$ (здесь C_e – удельная емкость сети, мкФ/км).

4. Рекомендована формула полного тока замыкания на землю в подземных сетях напряжением 6 кВ и улучшенным электрическим коэффициентом для отдельных элементов сети $K_{\text{эл}}$ = 1700 при погрешности расчета не более 7 %.

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ОПЫТ РЕАЛИЗАЦИИ ПРОЕКТОВ В ГОРНОПРОМЫШЛЕННОМ СЕКТОРЕ ЭКОНОМИКИ

Научная статья

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PEST-анализ развития Нивенского ГОКа на территории Калининградской области

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Аннотация
Сегодня, в реалиях экономической неопределенности и непредсказуемости, важно своевременно реагировать на новые вызовы, используя актуальные инструменты для принятия эффективных управленческих решений. В мировой практике одним из таких инструментов для эффективного управления проектами промышленных предприятий является PEST-анализ, функциональные возможности которого рассмотрены в данной работе. Цель исследования заключается в проведении PEST-анализа, рассмотрении и анализе факторов, оказывающих влияние на развитие Нивенского ГОКа, расположенного на территории Калининградской области, – экспортно-ориентированного инвестиционного проекта строительства нового промышленного предприятия с основным производством премиального минерального удобрения – сульфата калия. Посредством проведенного анализа полученных результатов в статье предложен ряд возможных превентивных мероприятий для нивелирования негативных факторов, а также для развития сильных сторон компании-оператора Нивенского ГОКа – ООО «К-Поташ Сервис». Таким образом, PEST-анализ ключевых микро- и макроэкономических факторов позволил комплексно оценить положение Нивенского ГОКа в создаваемой горно-химической промышленности Калининградской области. В работе установлена степень влияния на компанию и других внутренних и внешних факторов, включая возможности внедрения новых инновационных технологий в работу промышленного предприятия, тенденции развития рынка минеральных удобрений с учетом геополитического контекста. Выводы, рекомендации и предложения позволяют оценить компанию со стороны, а полученные результаты учитывать возможности развития промышленного предприятия при принятии стратегических и долгосрочных решений по развитию проекта. Основные результаты проведенного исследования могут быть использованы в качестве эффективных мер развития проекта нового промышленного предприятия – Нивенского ГОКа.

Ключевые слова
горная промышленность, горно-обогатительные предприятия, экономика, управление проектами, макроэкономические факторы, PEST-анализ, сульфат калия, добыча, макросреда, инфраструктура, Нивенский ГОК, Калининградская область, Россия

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EXPERIENCE OF MINING PROJECT IMPLEMENTATION

Research article

PEST-analysis of Nivensky Mining and Concentration Complex development in Kaliningrad region

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Abstract
Today, in the times of economic uncertainty and unpredictability, it is important to respond to new challenges in a timely manner, using relevant tools to make effective management decisions. In the world practice, one of such tools for effective management of an industrial enterprise project is PEST-analysis, the functionality of which is considered in this paper. The study purpose is to conduct PEST-analysis, review and analyze the factors

affecting the development of the Nivensky GOK (Mining and Concentration Complex), located in the Kaliningrad region – an export-oriented investment project to build a new industrial enterprise with core production of a premium mineral fertilizer, potassium sulfate. Based on the analysis, the paper proposes a number of feasible preventive measures for mitigating effects of negative factors, as well as for the development of strengths of the Nivensky GOK's company-operator, "K-Potash Service" LLC. Thus, PEST-analysis of key micro- and macroeconomic factors allowed to comprehensively assess the position of Nivensky GOK in the emerging mining-and-chemical industry of Kaliningrad region. The study established the degree of influence of other internal and external factors on the company as well, including the possibilities of introducing new innovative techniques in the industrial enterprise operation, trends in the development of the mineral fertilizer market, taking into account the geopolitical context. Conclusions, recommendations and proposals allow evaluating the company and the possibilities of its development for making strategic and long-term decisions on the project development. The main results of the study can be used as effective tools for optimizing development of the new industrial enterprise project, Nivensky GOK.

Keywords
mining, mining and concentration complexes, economy, project management, macroeconomic factors, PEST-analysis, potassium sulfate, extraction, macro environment, infrastructure, Nivensky GOK, Kaliningrad region, Russia

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Введение

Исторически сложилось, что Калининградская область развивается в сложных полуэксклавных условиях – это формирует неустойчивость её социально-экономической системы к действующим глобальным экономическим и политическим тенденциям. Однако и сам регион выступает макросредой для локальных субъектов рынка Калининградской области. Такие факторы, как изоляция от основной территории Российской Федерации, близость к экономически развитым странам Европы, статус свободной экономической зоны, предопределяют специфичность и типы развития бизнес-структур региона, а также систему управления проектами промышленных предприятий [1]. В связи с этим обязательным условием для эффективного развития экономики субъекта является постоянный мониторинг и комплексный анализ внешних и внутренних социально-экономических показателей, оказывающих влияние на все сферы деятельности.

Сегодня одним из популярных и практико-ориентированных инструментов планирования и формирования стратегических направлений деятельности предприятия является PEST-анализ, целью использования которого выступает мониторинг изменений макросреды по четырем основным направлениям, а именно исследование политических, экономических, социальных и технологических факторов внешней макросреды [2].

Практика применения PEST-анализа и его развитие начались в шестидесятых годах прошлого столетия. Плюсами использования PEST-анализа, несомненно, являются детальный и комплексный анализ внешней среды, формирование общего понимания рынка и перспектив развития исследуемого предприятия. В свою очередь, недостатками анализа выступают следующие факторы: неточность результатов анализа, значительное сокращение данных и трудность учета наиболее значимых факторов. В связи с этим для точности расчетов необходимо четко придерживаться методологии анализа, состоящего из различных этапов проведения комплексного исследования.

Первоначально необходимо провести исследование факторов по каждому из направлений, способных повлиять на прибыльность предприятия в долгосрочной перспективе, – в промежутке от 3 до 5 лет. После этого следует консолидировать и комплексно изучить информацию о каждом факторе с целью прогнозирования его динамики.

Далее необходимо оценить значимость выбранных факторов, направлений и их влияние на деятельность компании. Стоит отметить, что сила влияния фактора оценивается по шкале от 1 до 3. Следующим этапом является оценка вероятности изменения факторов по 5-балльной шкале, где 1 означает минимальную вероятность изменения фактора внешней среды, а 5 – максимальную. Для достоверности оценка степени влияния и изменения факторов должна проводиться экспертами в своей профессиональной области [3].

После проведения оценки в итоговой таблице PEST-анализа записывается среднее арифметическое результатов по каждому фактору, что позволяет сформировать вероятность колебания характеристик внешней среды. Завершающий этап анализа – приведение всех расчетов в матричный вид, где все факторы в порядке убывания своей важности размещаются в таблице.

Появление Нивенского ГОКа в структуре производственного сектора Калининградской области

К 2021 г. в региональной промышленности Калининградской области сформировался ряд направлений, являющихся локомотивами экономического развития. К таким направлениям относятся: автомобиле- и судостроение, обрабатывающие производства, добыча полезных ископаемых, обеспечение электроэнергией, газом и паром, а также другие виды экономической деятельности[1]. Однако освоение Нивенского месторождения на базе разведанных запасов калийно-магниевых солей Нивенской впадины изменит повестку стратегического развития Калининградской области, включив в реальную перспективу появление горно-химической промышленности[2].

Сегодня Нивенский ГОК для Калининградской области – это крупнейший инвестиционный проект строительства нового промышленного предприятия с основным производством премиального минерального удобрения – сульфата калия или Sulfate of Potash (далее – SOP)[3].

Стоит отметить, что SOP – это нишевый продукт, и объем его потребления ограничен, что определяет высокую цену на мировых рынках. Премиальное удобрение содержит два питательных компонента: калий и серу, и не содержит хлор. В мировом сельском хозяйстве это удобрение используется для высокоценных культур – картофеля, сахарной свеклы, цитрусовых, тепличных и других растений [4].

Безусловно, появление крупного экспортно-ориентированного промышленного предприятия – это вызов существующей инфраструктуре Калининградской области, включая транспортно-логистическую систему, социальные институты, смежные народно-хозяйственные отрасли региона. Стоит заметить, что на 2020-й год запас прочности и гибкость перечисленных структур не отвечают требованиям будущего производства [5].

Отсюда следует, что необходимость диверсифицировать экономику региона и появление Нивенского ГОКа – два фактора, определяющих актуальность проведения в настоящий момент комплексного анализа условий работы предприятия.

PEST-анализ макроэкономических факторов влияния на Нивенском ГОКе

Сегодня PEST-анализ незаменим для исследования условий бизнеса, на которые нельзя повлиять напрямую. Собственно, аббревиатура PEST совмещает в себе Political (политические), Economical (экономические), Social (социальные) и Technical (технологические) факторы, которые при помощи данного метода стремятся оценить управленцы, маркетологи, эксперты, чтобы понять не только их влияние, но и вероятные варианты развития ситуации, а также то, насколько «жизнеспособным» в целом является рассматриваемый проект [3, 6 ,7].

Базируясь на принципах оценки PEST-анализа, в ходе работы авторы сформировали перечень факторов и параметров макросреды, которые воздействуют на развитие компании ООО «К-Поташ Сервис», выступающей компанией-оператором проекта Нивенского ГОКа, и провели анализ возможностей развития горно-химической промышленности на территории региона. Весь комплекс макроэкономических факторов, оказывающих влияние на развитие Нивенского ГОКа, сгруппирован в соответствии с подходом PEST-анализа по четырем основным категориям, что отражено на рис. 1.

Стоит отметить, что для оценки степени влияния установленных факторов макросреды на компанию ООО «К-Поташ Сервис» были привлечены эксперты, компетенции которых соответствуют виду деятельности исследуемой компании, что позволило максимально точно оценить степень влияния факторов на предприятие. Также в работе использованы доступные источники информации в целях получения более полной и достоверной картины о внешней среде: внутренняя информация, обзоры рынков, статьи в научных журналах, эмпирические исследования независимых экспертов.

Определившись с методом макроанализа и перечнем факторов для его проведения, эксперты рассмотрели и оценили внешнюю среду компании ООО «К-Поташ Сервис» с разных сторон. Таким образом, в ходе проведения PEST-анализа был составлен авторский подробный список PEST-факторов, оказывающих влияние на развитие компании. После чего экспертами была проведена оценка влияния данных факторов на развитие Нивенского ГОКа в сложившихся макроэкономических условиях [15, 16].

Первоначально экспертами индивидуально была установлена степень влияния факторов на реализацию проекта Нивенского ГОКа – от 1 (min) до 3 (max). Стоит отметить, что используемые для оценки факторы, представленные на рис. 1, определяют стратегию развития проекта Нивенского ГОК, оказывая непосредственное влияние на среду и прибыль предприятия, финансово-хозяйственную деятельность, продажи, распределение ресурсов, технологическое развитие, которые часто являются причинами изменений и потери места на рынке.

Базируясь на консолидации результатов оценки экспертов по каждому фактору в заданных параметрах, с помощью функции среднего значения в программе Microsoft Excel, авторы рассчитали среднее арифметическое по оценкам. Таким образом, консолидировав результаты оценки экспертов, рассчитали среднее значение по каждому фактору, что позволило выявить те из них, которые оказывают наибольшее воздействие.

Далее была произведена оценка вероятности изменения установленных факторов по шкале от 1 (min) от 5 (max), результаты которой представлены в табл. 1. В результате проведенных экспертно-оценочных процедур была оценена реальная значимость каждого фактора, что позволило выяснить, насколько следует контролировать тот или иной фактор при дальнейшей реализации проекта. Для оценки фактора с поправкой на вес среднее значение экспертной оценки умножается на степень влияния фактора, а результат делится на сумму средних баллов экспертных оценок.

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[1] Отчет о деятельности Правительства Калининградской области за 2019 год, в том числе по вопросам, поставленным Калининградской областной Думой. URL: https://gov39.ru/working/ekonomy/strategy/ezhegodnye-otchyety/ (Дата доступа: 18.10.2020).

[2] Справка о состоянии и перспективах использования минерально-сырьевой базы Калининградской области на 15.06.2020 год. URL: https://www.rosnedra.gov.ru/data/Fast/Files/202011/d56faa9eb7abd9a280781c366e04f3d5.pdf (Дата доступа: 18.10.2020).

[3] Официальный сайт ООО «К-Поташ Сервис» URL: http://k-potash.ru/ (Дата доступа: 18.10.2020).

Р-факторы (политические)

- законодательство по охране окружающей среды;
- политика РФ по отношению к соседним государствам и миру;
- частные инвестиции в промышленность [8];
- налоговая политика (тарифы и льготы);
- готовность региона к реализации проекта Нивенского ГОКа;
- государственные дотации на развитие инфраструктуры и бизнеса;
- будущее и текущее законодательство, регулирующее правила работы в отрасли;
- количественные и качественные ограничения на импорт, торговая политика;
- тенденции к регулированию или дерегулированию горнодобывающей отрасли;
- стремление к протекционизму отрасли, наличие государственных компаний в отрасли;
- стратегия развития Калининградской области;
- устойчивость политической власти РФ;
- степень защиты интеллектуальной собственности и закон об авторском праве и др. [9];

Е-факторы (экономические)

- дистрибуция, логистика [10];
- инвестиционный климат в исследуемой области;
- зависимость от регуляторов по предоставлению энергоносителей в области;
- степень глобализации и открытости экономики [11];
- курсы основных валют;
- платежеспособность потребителей;
- кредитно-денежная и налогово-бюджетная политика области в Калининградской области;
- экономическая стабильность: ВРП Калининградской области, ВВП, темпы инфляции;
- зависимость от поставщиков сырья и оборудования;
- темпы роста экономики;
- уровень инфляции и процентные ставки;
- специфика производства сульфата калия в аспекте реализации проекта Нивенского ГОКа;
- уровень развития предпринимательства и бизнес-среды в Калининградской области;
- уровень располагаемых доходов населения в Калининградской области и др.

S-факторы (социальные)

- бренд, репутация компании, имидж используемой технологии [12];
- негативное отношение жителей региона к реализации проекта Нивенского ГОКа;
- отношение к натуральным и экологически чистым продуктам;
- уровень миграции и иммиграционные настроения;
- требования к качеству продукции и уровню сервиса;
- потребительские предпочтения;
- отношение к работе, карьере, досугу и выходу на пенсию;
- отношение к импортным товарам и услугам;
- уровень здравоохранения и образования в Калининградской области;
- темпы роста населения в Калининградской области и др.

Т-факторы (технологические)

- развитие конкурентных технологий [13];
- степень использования, внедрения и передачи технологий;
- ключевые технологические изменения;
- потенциал инноваций [14];
- доступ к новейшим технологиям;
- зрелость технологий;
- законодательство по технологиям;
- уровень инноваций и технологического развития отрасли;
- расходы на исследования и разработки;
- в целом технологическая скорость реагирования среды;
- производственная емкость, уровень;
- развитие и проникновение Интернета, развитие мобильных устройств;
- законодательство в области технологического оснащения отрасли и др.

Рис. 1. Комплекс макроэкономических факторов, оказывающих влияние на развитие Нивенского ГОКа (ООО «К-Поташ Сервис») (*составлено авторами*)

Fig. 1. A set of macroeconomic factors affecting the Nivensky GOK (K-Potash Service LLC) development (*compiled by the authors*)

Таблица 1 / Table 1

Выявление ключевых факторов в результате проведения PEST–анализа компании ООО «К-Поташ Сервис»

Identification of key factors from PEST analysis of "K-Potash Service" LLC

Факторы	**Экспертная оценка с поправкой на вес**
Р-фактор (политические)	
Законодательство по охране окружающей среды	0,0919
Политика РФ по отношению к соседним государствам и миру	0,0744
Частные инвестиции в промышленность	0,0626
Налоговая политика (тарифы и льготы)	0,0548
Готовность региона к реализации проекта Нивенского ГОКа	0,0477
Государственные дотации на развитие инфраструктуры и бизнеса	0,0443
Будущее и текущее законодательство, регулирующее правила работы в отрасли	0,0443
Количественные и качественные ограничения на импорт, торговая политика	0,0431
Тенденции к регулированию или дерегулированию горнодобывающей отрасли	0,0376
Стремление к протекционизму, наличие государственных компаний в отрасли	0,0374
Стратегия развития Калининградской области	0,0352
Устойчивость политической власти РФ (федеральной, местной)	0,0352
Бюрократизация и уровень коррупции	0,0318

Окончание табл. 1

Факторы	**Экспертная оценка с поправкой на вес**
Свобода информации и независимость СМИ	0,0204
Антимонопольное и трудовое законодательство	0,0204
Степень защиты интеллектуальной собственности и закон об авторском праве	0,0136
Вероятность развития военных действий в РФ	0,0102
Е-фактор (экономические)	
Дистрибуция, логистика	0,0827
Инвестиционный климат в исследуемой области	0,0744
Зависимость от регуляторов по предоставлению энергоносителей в области	0,0735
Степень глобализации и открытости экономики	0,0666
Курсы основных валют	0,0647
Платежеспособность потребителей	0,0597
Кредитно-денежная и налогово-бюджетная политика области в Калининградской области	0,0511
Общие проблемы налогообложения (Калининградская область – ОЭЗ)	0,0477
Экономическая стабильность: ВРП Калининградской области, ВВП, темпы инфляции	0,0470
Зависимость от поставщиков сырья и оборудования	0,0434
Зависимость от импорта при реализации товарной продукции	0,0409
Темпы роста экономики	0,0340
Уровень инфляции и процентные ставки	0,0340
Уровень безработицы, размер и условия оплаты труда в Калининградской области	0,0310
Специфика производства сульфата калия в аспекте реализации проекта Нивенского ГОК	0,0306
Уровень развития предпринимательства и бизнес-среды в Калининградской области	0,0221
Уровень располагаемых доходов населения в Калининградской области	0,0187
Сезонность/ влияние погоды	0,0102
S-факторы (социальные)	
Бренд, репутация компании, имидж используемой технологии	0,0822
Негативное отношение жителей региона к реализации проекта Нивенского ГОКа	0,0822
Отношение к натуральным и экологически чистым продуктам	0,0647
Уровень миграции и иммиграционные настроения	0,0626
Требования к качеству продукции и уровню сервиса	0,0509
Потребительские предпочтения	0,0409
Отношение к работе, карьере, досугу и выходу на пенсию	0,0306
Отношение к импортным товарам и услугам	0,0289
Уровень здравоохранения и образования в Калининградской области	0,0221
Темпы роста населения в Калининградской области	0,0221
Демография, плотность населения в Калининградской области	0,0170
Образ жизни и привычки потребления	0,0145
Развитие религии и прочих верований	0,0085
Т-факторы (технологические)	
Развитие конкурентных технологий	0,0919
Степень использования, внедрения и передачи технологий	0,0666
Ключевые технологические изменения	0,0626
Потенциал инноваций	0,0579
Доступ к новейшим технологиям	0,0477
Зрелость технологий	0,0460
Законодательство по технологиям	0,0434
Уровень инноваций и технологического развития отрасли	0,0374
Расходы на исследования и разработки	0,0332
В целом технологическая скорость реагирования среды	0,0318
Производственная емкость, уровень	0,0306
Развитие и проникновение Интернета, развитие мобильных устройств	0,0306
Законодательство в области технологического оснащения отрасли	0,0289

Источник: составлено авторами.

Чем выше оценка с поправкой на вес, показатели которой определяют реальную значимость каждого фактора, тем больше внимания следует уделять нейтрализации данного фактора. Исходя из полученных результатов действия компании на возможности и риски внешней среды разработаны для всех факторов, показатель которых выше значения 0,005.

После проведения оценки влияния каждого фактора были выделены те из них, которые оказывают максимальное влияние на развитие ООО «К-Поташ Сервис». В частности, в качестве ключевых Р-факторов можно выделить законодательство по охране окружающей среды с результатом экспертной оценки 0,0919, а также политику РФ по отношению к соседним государствам и миру (0,0744) и частные инвестиции в промышленность (0,0626). Среди Е-факторов были выделены: дистрибуция и логистика (0,0827), инвестиционный климат в исследуемой области (0,0744) и др.; S-факторы: бренд, репутация компании, имидж используемой технологии (0,0822), негативное отношение жителей региона к реализации проекта Нивенского ГОКа (0,0822) и др.; Т-факторы: развитие конкурентных технологий (0,0919), степень использования, внедрения и передачи технологий (0,666) и др.[4]

В данном случае полученная реальная значимость фактора позволяет оценить, насколько компании следует обращать внимание и контролировать факторы изменения внешний среды, и рассчитывается, как вероятность изменения фактора, умноженная на силу влияния этого фактора, отражается на деятельности компании.

Каждый из ключевых факторов был проанализирован с точки зрения его воздействия на экономику региона в целом и компании в частности для разработки и внедрения мер по снижению негативного влияния (если фактор негативный) на деятельность компании и максимальному раскрытию положительных факторов.

Таким образом, качественный анализ ключевых факторов внешней среды, оказывающих влияние на развитие ООО «К-Поташ Сервис», выявил ряд возможных ответных мер для разрешения вопросов, сдерживающих эффективное развитие проекта промышленного предприятия на территории Калининградской области.

Так, например, постоянный мониторинг готовящихся изменений для превентивного реагирования и участие в лоббизме законопроектов, а также совершенствование политики в части развития экспортно-ориентированной дистрибуции позволят усовершенствовать работу компании в области политической сферы (Р-факторов). Что касается остальных факторов, то экспертами было предложено: кооперация органов власти и частного сектора для развития инфраструктуры региона и совершенствование инвестиционного климата региона на законодательном уровне (Е-факторов), активизация PR-служб органов власти и частного сектора и постоянное информирование граждан, участие в социальной, политической, культурной жизни области (S-факторы); участие в качестве венчурного инвестора в создании новых технологий и привлечение независимых консалтинговых агентств для постоянного аудита работы всех служб предприятия в сфере развития технологий (Т-факторы)[5].

Таким образом, внешняя среда, которая воздействует на компанию ООО «К-Поташ Сервис», может оказать на нее значительное негативное влияние в основном из-за нестабильности существующих экономических и правовых систем области, отсутствия необходимых транспортно-логистических мощностей в регионе. Значительным фактором в эффективном развитии компании является проработка вопроса, связанного с настроением местных жителей Калининградской области, в части разъяснений и проведения мероприятий на тему безопасности будущего производства Нивенского ГОКа.

При этом ООО «К-Поташ Сервис» для преодоления негативного воздействия внешней среды следует опираться на свое положение как системообразующего предприятия для Калининградской области, а также превентивно использовать предложенные в статье меры с целью противодействия внешним угрозам.

Заключение

Проведенное исследование текущего внешнего влияния на развитие Нивенского ГОКа, экспортно-ориентированного инвестиционного проекта строительства нового промышленного предприятия с основным производством премиального минерального удобрения – сульфата калия, посредством PEST-анализа, позволило определить стратегическое направление развития данного проекта, а также установить факторы, которые могут измениться в будущем.

Таким образом, PEST-анализ ключевых макроэкономических факторов позволил комплексно оценить положение Нивенского ГОКа в создаваемой горно-химической промышленности Калининградской области, а также выявить степень влияния на компанию других внешних факторов. Проделанная работа позволяет оценить компанию со стороны, а полученные результаты учитывать при принятии кратко- и долгосрочных решений по развитию проекта и формировании системы административно-управленческой деятельности в развитии проекта промышленного предприятия Нивенского ГОКа.

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[4] Федеральный закон «Об охране окружающей среды» от 10.01.2002 № 7-ФЗ (с изменениями на 31 июля 2020 года). URL: http://docs.cntd.ru/document/901808297 (Дата доступа: 18.10.2020); Налоговый кодекс Российской Федерации (НК РФ). URL: http://base.garant.ru/10900200/ (Дата доступа: 18.10.2020).

[5] Курсы ЦБ РФ обмена валют URL: https://www.vbr.ru/banki/kurs-valut/cbrf/ (Дата доступа: 18.10.2020); KGD.RU: Госэкспертиза выдала положительное заключение на проект по разработке месторождения солей в Нивенском. URL: https://kgd.ru/news/society/item/90630-gosjekspertiza-vydala-polozhitelnoe-zaklyuchenie-na-proekt-po-razrabotke-mestorozhdeniya-solej-v-nivenskom (Дата доступа: 18.10.2020); КАЛИНИНГРАДСТАТ: Миграция населения Калининградской области в январе-июле 2019 года. URL: http://kaliningrad.old.gks.ru/wps/wc-m/connect/rosstat_ts/kaliningrad/resources/c2098e804ee2773fbfe5bf3467c8ff84/%D0%9C%D0%B8%D0%B3%D1%80%D0%B0%D1%86%D0%B8%D1%8F-07.pdf (Дата доступа: 18.10.2020).

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