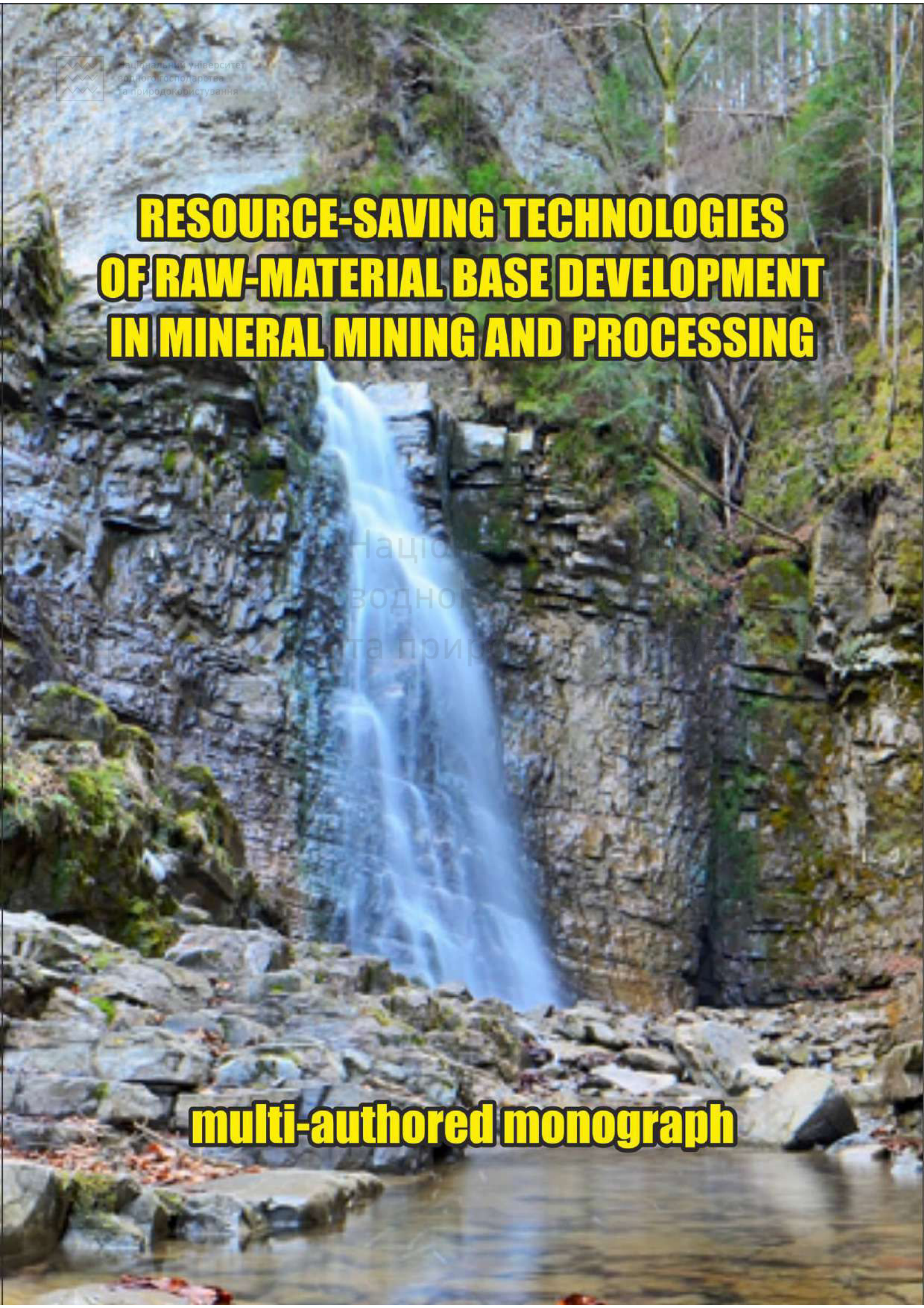
RESOURCE-SAVING TECHNOLOGIES OF RAW-MATERIAL BASE DEVELOPMENT IN MINERAL MINING AND PROCESSING



multi-authored monograph

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Reviewers: ***Mihaela TODERAS***, Ph.D.Habil.Eng., Professor,
Vice-Dean Faculty of Mines University of Petroșani, Romania

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The monograph considers potential technological development of ore mining and processing industries through updating mining machines and technologies
The book is intended for a broad mining audience of scholars, practitioners, postgraduates and students.

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PREFACE

We are glad to present the monograph "Resource-saving technologies of raw-material base development in mineral mining and processing".

The monograph considers some specific features of technological advance in the mining and processing industries including the necessity of updating current enterprises and developing mineral deposits in various regions. The stages of forming foundations of developing deposits and parameters of mining enterprises are revealed. The components of the technological advance forecast are under analysis. Global trends in developing the mining and processing industries and forecast of basic development trends of these industries are presented. New efficient technologies applied to mineral mining and processing are analyzed in detail.

Co - editors,

Zinovii MALANCHUK - Doctor of Sciences (Engineering), Professor, Director Institute of Postgraduate Education, National University of Water and Environmental Engineering, Ukraine.

Roland MORARU, Ph.D.Habil.Eng., Professor, Research Vice-Rector University of Petroșani, Romania.

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JUSTIFICATION OF THE PROSPECTS FOR INNOVATIVE DEVELOPMENT OF THE ENTERPRISE FOR THE EXTRACTION OF COPPER-CONTAINING BASALTS

Malanchuk Z.R.
National University of Water and Environmental Engineering (NUWEE), Doctor of Technical Sciences, Professor, Department of Development of Deposits and Mining, Ukraine

Abstract
Attracting investments for the development of natural resources enterprise needs deep justification of their efficiency. The conducted analysis of basalt processing technologies is indicative of fairly wide possibilities of its processing. Physical and mechanical properties and chemical composition of basalts offer an opportunity to use them in different industries.

The main problem is the presence of native copper in the deposits of the basalt quarry and the lack of technology for its extraction and processing. The development of this technology can provide a prospective and sustainable development of the enterprise in the future.

1. Mining-and-geological and technological analysis of the structures of deposits of copper-bearing basalts

Volyn basalts attract researchers with their unique properties as per mineralogical and chemical composition. Their isotopic age according to the potassium-argon method is 510-598 million years. They are represented by two modifications. Aphanitic basalts are black and dark-grey aphanites. Basically it is palagonite basalt. Its mineral composition is as follows: plagioclase - 36%, pyroxene - 33%, glass - 19%, palagonite - 6%, ore mineral - 6%. These basalts come out in the quarries of the villages of Berestovtsi, Yanova Dolyna, Ivanchi, Polytsi, etc. Amygdaloidal basalts are greenish-grey fine-grained rock with a large number of amygdules up to 15 mm in size. Mineral composition is the following: plagioclase, ore mineral (magnetite, titanic iron ore), apatite, rock glass. The main eruptions are in the Styr basin. Rock density is ρ=2.65 g/cm^3. The chemical composition of the basic basalts is presented in the table 1 (weight content in percentage) [1-10].

In order to destroy the ore body and subsequently to extract the mineral product it is necessary to expend a significant amount of energy. The development of new technologies for processing and preparing of the rock mass involves the use of the properties of rocks inherent in nature to reduce energy consumption for their destruction, increase liberation of ores and extraction of mineral products. Therefore, it is relevant to study the natural jointing of various rocks in a basalt mass, which can significantly change the approaches and principles of technological impact on them for further destruction and extraction of mineral products.

Geologists at the end of nineteenth and the beginning of the twentieth century took interest and began to study structures in rocks in the form of joints at the macro and mega levels, designated as autonomous blocks in the earth's crust by cracks and faults of different sizes. Generally split horizontal layers on the platform were detected beside vertical ones. Therewith the cracks had the regular arrangement - vertical and each of their systems had sustained stretching over a large territory, which is indicative of directed tectonical forces that are transmitted through the platform [9,10].

Currently this factor is mainly not taken into account in the existing crushing plants. The grinding and preparation equipment and the processes accompanying them are aimed only at release of grains of the extracted mineral, not taking into account the structure and texture of the bearing rocks, so the task is to find the optimal size of the ore (or rock) constituent to obtain maximum ore release, maximum convergence of the size of the technological product and natural consistency of its structure.

The solution of the set task requires conduct of additional researches and will allow to form the physical process of mining recovery using new technologies and principles of ore processing and extraction of commercial components (obtaining quarrystone, break stone and chips of the correct geometric shape, reducing the number of splices in concentrate during enrichment, removing precious metal pick-ups without their destruction, etc.).

According to our empirical investigations the destruction of rock mass in joints occurs most vivid in centrifugal crushers and mills. For example, centrifugal crushers are used to produce breakstone of a cubic shape at relatively low energy consumption. The destruction of the ore body rock that contains metal pick-ups occurs without grinding of metal inclusions.

Table 1

Chemical composition of basic basalts by quarries

Coming out places	Chemical elements composition						
	SiO_2	TiO_2	Al_2O_3	Fe_2O_3	FeO	MnO	MgO
	1	**2**	**3**	**4**	**5**	**6**	**7**
village Khodosy, village Hutvyn, village Yanova Dolyna, village Mydsk	45,04	2,54	14,30	6,03	6,46	0,4	8,47
village Berestovtsi, village Yanova Dolyna (quarry No.2)	49,05	2,85	12,79	3,36	10,63	0,21	6,19

Continuation of table. 1

Chemical composition of basic basalts by quarries

Coming out places	Chemical elements composition							
	CaO	Na_2O	K_2O	P_2O_5	SO_3	CuO	H_2O	Other
	8	**9**	**10**	**11**	**12**	**13**	**14**	**15**
village Khodosy, village Hutvyn, village Yanova Dolyna, village Mydsk	6,58	2,42	0,48	0,17	0,3	0,03-1,2	0,72	1,88
village Berestovtsi, village Yanova Dolyna (quarry No.2)	9,38	2,78	2,05	0,57	0,2	0,22	0,8	2,41

Apparently the destruction in this case occurs along the so-called reticular planes, which is characterized by the number of nodes (atoms, ions) of the plane lattice per unit of its area (according to the Bravais theory). Apparently this rule applies not only to crystals, but also to rocks in general. The obtained results need additional applied and theoretical researches. The brief review of the theoretical aspects and analysis of existing ideas about the jointing of rocks that is carried out with the participation of the author indicates on the topicality and prospects of such researches. Recent works show the efficiency of accounting of the structure of deposits when grounding the parameters of their extraction and processing. The main laws of development of natural jointing of igneous rock deposits on the territory of Zhytomyr region are determined in the work, a relationship between the orientation of tectonic faults and cracks of natural joints, as well as between the location of faults and deposits of trim stone is revealed. It is established that the law of development of natural fractures of joints is tributary to the normal law. Account of the peculiarities of the mass structure and the anisotropism of the trim stone made it possible to improve the sawing of blocks and to reduce its losses.

In the work [11-13] on the basis of established connection between the mechanism of creation of the oriented structure of bearing rocks and their peculiarities is recommended the character of the mechanical impact at break. It was established for the first time that at static load the fracture pattern of the intrusive rock is determined by the peculiarities of the microstructure – the orientation and degree of orientation of microcracks, and under dynamic load the nature of destruction is influenced by the macrostructure of the mass – orientation of microcracks and the degree of their opening. Based on the results of the investigations the technology of severance of granite monoliths from the mass was improved and the percentage of blocks output was increased.

Based on the positions of blocky structure and cleavage the studies of basalt coming out under the conditions of the Rafalivskyi and Berestovetskyi quarries showed their high cleavage (working height is 15-20 m), clear cleavages, limited by vertical and horizontal cracks. Currently the extraction of basalt is done for break stone. The technological scheme of extraction includes open cut mining (the thickness of the sand-chalk layer of opening is 2-5 m), drilling of wells in accordance with the blasting pattern, undercut of stope that is followed by excavator loading in road transport for delivery to the crushing-and-sorting site. Salable material in the form of break stone in three classes of fineness (10; 10-20; 20-40 mm) is obtained using pin breakers, gyratory cone breakers, centrifugal crushers and vibratory screens.

In the process of analysis of mined and crushed rock, especially in basalts and lava-breccias it is possible visual selection of barrel-copper in the form of different size chips. Pick-ups in general have branched structure and weigh from several to hundreds of grams. On a massive scale are found chips of copper with the weight of 10-40 grams. The biggest pick-up was found in Rafalivskyi quarry and it weighs 1.4 kg [14, 15].

2. State of exploitation maturity of the process of extraction and processing of copper-bearing basalts

Over the last years are carried intensive researches and commercial exploitation of basalt deposits in Rivne-Volyn district due to detected valuable raw material types. Trappean structure of basalt deposits, diversity of mineralization forms and mineral rock composition even at the state of art of exploration maturity are indicative of the necessity of complex approach in the deposits development, moreover it refers to the mining and processing technology and to the usage of final product. For example, multilayer structure of basalt mass that contains directly basalts, tuffs of different mineral composition, lava-breccias with intensive inclusions of barrel-copper and ore layers with inclusions of precious metals in bulks, call in doubt the rationale of basalt extraction only for production of break stone and face slabs.

Multifactority of the composition, different cover thickness and character of bearing rocks need complex technology of extraction (open-cut or shaft mining, hydraulic mining by boreholes). In such a way the diversity of mineral deposits requires complex of the applied influence means on the deposits and familiarization of different processes of mining industry. Suffice it to say that the territorial location of deposits in protected areas leads to reduction in the open-cut mining development method from an environmental point of view. The best in this case is the wellfield method based on the transfer of a mineral deposit into a mobile state in situ. In this case compared with traditional (open-cut and deep-mining methods) wellfield methods of extraction of mineral deposits have two fundamental differences:

- release, preparation and development of the deposit is done through well drilling and as a rule the extraction is done from the earth surface;
- mineral deposit is extracted to the surface preatreated by one of the processing methods, for example, by hydraulic washing-out. In this case the features of the ore body are used during elaboration of technology of its processing in the place of bedding.

At the same time the work is planned to manage the rock mass over the worked out space and are taken into account the processes that take place in the mass. According to the complex of works under consideration in Rivne-Volyn region a scientific base has been developed for hydraulic mining by boreholes technology for the extraction of heavy metals from placer accumulations and legacy placers. In the stated works the implementation of the concept of exploration, commercial preparation and development of mining of valuable minerals in Rivne region is described, the directions for increasing the degree of their concentration at the places of accumulation with leaving soft bearing rocks in underground conditions are developed.

During the former investigations of Rivne mining party it was established not only diversity and volume of commercially valuable mineral deposits but also the stated conditions of their bedding, thickness of beds and percentage content of each mineral deposit, pattern of their occurrence in the thickness of basalt deposit [14, 15]. Therefore the feasibility of a comprehensive not only production, but also processing is determined by the capabilities of the deposit and modern technologies, the requirements of the ecological situation in the region and the relevance for the national economy of deposit derivative products.

For the accuracy of the concept of complex conduct of mining and processing works in Volyn region it is necessary in the first instance to conduct short analysis of the deposit capabilities.

Researchers have a profound interest to the natural minerals zeolites, since there is an experience of their practical use in the world. Japanese scientists, and subsequently industrialists of the USA, China, Russia, Bulgaria and Ukraine, where the zeolite deposits were discovered, have established a positive effect on crop yields when applied to the soil, as well as feed additives for animals and poultry. Great prospects have been established for the use of zeolite-smectite tuffs to combat technogenic pollution of the environment with radionuclides and heavy metals. Taking to account the scale of the contaminated territories, especially in the industrially developed regions of Ukraine, large areas require preventive treatment, so the extraction of zeolites is relevant. Tuff reserves in the basalt quarries of Rivne region are estimated at about 20 million tons. They underlay at a depth of 10 meters or more, so they can be mined by the open-cut mining and considering the solubility of some types of tuff downhole technology is also possible.

According to the studies of the mineral composition of zeolite-smectite cinerites of Rivne region the main minerals in their composition are zeolites, smectites, iron-bearing dispersed minerals and aluminosilicates. Most of them have a crystalline lattice of the plate-sheet type, which determines the high specific surface of individual particles that have an exposed cavity. Therefore minerals actively interact with water and are successfully used for its purification [16, 17, 18].

When interacting with water smectite tuff minerals that have an expensive crystalline lattice increase or decrease in volume by up to 20 times, when dehydrated the volume of the mineral decreases sharply, the plastic property of wet minerals increases, therefore, a hydraulic mining by boreholes technology can be used.

The important component of trappean structure of basalts in Volyn is lava clastic breccia that underlay in the bottom of basalt layers. The geologists of Rivne regional geological office among the lava clastic breccia of Rafalivka ore cluster opened perspective deposits of barrel-copper with different percentage content. It has to be mentioned that alongside with the main basalt mass in the content of rock the significant role play aqueous and pyroclastic rocks that occupy around 50% of its stratigraphic framework. Altogether they compile (from the bottom up) four rock formations: Horbashi (gritstone sandy with admixture of pyroclastics); Zabolotsi basalt with tuff interburdens; Babyn tuff and Ratne basalt with layers of lava clastic breccia, tuff bedding rocks, tuffites and tuff glomerations. Copper bearing lava-breccia are cropping out in the range of quarries of basalt extraction. Lava-breccia underlay in the tuffs of Babyn rock formations, at the core of the first (from the bottom) basalt flow of Ratne rock formations in the form of bedding rock with the efficiency of 1.0-0.7 m. The content of copper in lava-breccias changes from 0.04 to 5.0%. The copper is in the form of pick-ups of different coarseness and form. The detected pick-ups have weight of more than 700 grams. Shallow bedding of productive strata in the quarries and borehole method cause high prospects for the detection of deep fields of barrel-copper and precious metals. Beside copper in the content of ore bodies were detected silver, gold, palladium, rhodium, platinum that show economic interest and however currently not being under development need careful examination for preindustrial preparation for complex field development.

The study of the peculiarities of deposits of barrel-copper in

Zhyrychi and Rafalivka allowed establishing that copper mineralization has several morphotypes with different localization conditions. The classification of barrel-copper of Volyn is given in the work and the main places of its location in ore bodies are defined. So diffuse-porphyry and laminated-porphyry barrel-copper is localized mainly in flow-laminated, slightly changed phaneric basalts in the low part of powerful flows. Its concentration hardly ever exceeds 0.3 % in basalts and in the basaltic tuffs of Babyn rock formation were found copper concentrations up to 1.0 % in the ore interval of 1.5-2.0 m.

Vein-disseminated barrel-copper is spread in the cracks and veins of hydrothermal mineralization of basalts and tuffs. Its concentration is very uneven and increases in the places where stringer-porphyry mineralization is put upon the bedrocks with dispersed disseminated copper. A high copper content was found in the relatively thin bedrocks of the lava clastic breccias at the base of the lower basalt flow of the Ratne rock formation. The copper content in them varies from 0.04 to 5.0%. Further studies showed that finer inclusions of barrel copper make up the bulk in the ore bodies of amygdaloidal basalts, tuffs and tuff breccias, in the quartz reefs of the Zhyrychi and Rafalivka volcanic rocks [2-4, 10, 17].

Studies with the participation of the author of this work as a result of spectral analysis of rocks of the Rafalivka basalt quarry established that in the aphanitic basalt with stringer-porphyry barrel-copper mineralization the copper concentration is in the range of 0.43%. In lava-breccia of the basalt content the percentage of copper is in the range of 4.0-0.174%, and in basalt tuffs with dispersed disseminated and veined porphyry mineralization the percentage of copper is about 0.7%. In this case there is an uneven content of copper in all three rocks. Taking to account the variety of morphotypes of barrel-copper mineralization there is a need for their separate study to search for processing and extraction methods. Previous studies of the authors have shown that each type of mineralization requires an individual approach. In the case of coarsely disseminated copper (more than 3-5 mm) it is advisable to use the method of crushing and, especially, grinding without violating the integrity of the pick-ups. In this case an incomplete cycle of their enrichment is possible, which eliminates such technologically complex operations as precipitation and flotation, since it is possible to replace them with fine classification and electric separation.

Finely disseminated mineralization needs fine crushing of bearing rocks with the following methods of flotation concentration or combination of fine crushing, grinding with the following fine classification (less than 50 microns) for separation of finely ground bearing rock from small copper pick-ups. Therewith the important requirement is the exclusion of regrinding of disseminated copper.

Taking to account the abovementioned peculiarities of the technology in the Institute of Geotechnical Mechanics of the National Academy of Sciences of Ukraine was developed internally rolling conical mill that allows limiting the fragmentation size of finely disseminated copper. It conducts its preliminary preparation and the task of this process is assurance of output coarseness for grinding on the specially developed centrifugal mill where the metal inclusions of barrel-copper are not ground but only deformed, while bearing rocks are ground until micron level releasing finely disseminated copper inclusions. Loading out of the centrifugal mill is done on the vibrating sieve of fine classification with the aim of separation of the pick-ups (oversieve product) from dead ground (undersieve product). A further process operation is to separate the pick-ups from the splices by electric separation or metallurgical extraction. Crushed basalt after screening at a screen is a valuable chemical raw material (basalt casting, thermal insulation wool, etc.) [14, 15].

It should be noted that the abovementioned basic operations of the technology for the extraction of barrel-copper have been tested on lava breccias, however, for brown tuffs with high mineralization with titano-magnetites and finely disseminated barrel-copper they can be used for dry enrichment of these metals from tuff. After fine classification tuff screening products are also valuable chemical raw material for the production of fertilizers, feed additives and water purification filters.

The significant experience has been gained in the quarry mining of ores, which makes it possible to borrow the most effective technical solutions by the technology for extraction and usage of necessary equipment. The ore preparation processes for ores of different mineralization and percentage composition require an individual solution, and the issue of the extraction of barrel-copper in the basalt deposit on an industrial scale in Ukraine is being solved for the first time, and, nevertheless, the available experience allows us to solve the problem brought up in the work [19-26].

Thus the complex processing and usage of basalt raw material is technically feasible. The process requires pre-industry verification, proving, study of features, selection of equipment and development of recommendations for its industrial development.

3. Barrel-copper extraction as the direction of innovative development of the Private Joint-Stock Company "Rafalivka basalt quarry"

One of the main factors in the development of the mining industry enterprise in the long term is the implementation of innovations. Precisely thanks to innovations that such important tasks are solved: in the interests of business entities - the competitiveness of the enterprise, its flexibility to changes of external environment increases, financial stability enhances; for consumers - the number of purchase alternatives is increasing and the circle of satisfied needs is expanding.

The topicality and importance of solving of these tasks is due to the great variability of the business environment and, accordingly, excessive risks hinder the process of innovation implementation in the enterprise.

The main directions of innovations implementation at an industrial enterprise in the strategic context of its activities are product diversification, its improvement and market diversification, as well as the introduction of a differentiation strategy.

The implementation of innovations based on product diversification involves the release of new products or products that are manufactured on the basis of the application of new technologies not previously used by the enterprise, and market diversification involves the entry by the enterprise on the new markets or new segments of the market where it acted up till now.

Attracting investments for the development of an enterprise of mining industry requires a deep justification for their efficiency. The problem of investment efficiency has always occupied an important place among the urgent problems of economic science. Interest in it arises at different levels of economic management - from owners of business entities to state leaders. Efficiency belongs to the most general economic categories peculiar to any branch of industry.

First of all it is necessary to clearly define what “efficiency” is, since this concept as the main goal of the functioning of any activity is rarely subject to theoretical consideration precisely because of the ambiguity of perception. Also often in modern literature such concepts as "efficiency" and "performance" cause a lot of inconsistencies. In order to be successful over extended periods, to survive and achieve own goals, the organization has to be both efficient and effective.

Thus, by its main attribute, efficiency is the result of activity obtained per unit of used (spent) resources. Herewith the obtained result can have a different expression, since different resources can be used to achieve the effect. However whatever the result and resources are taken, the efficiency always reflects their correlation, since its essence ultimately manifests itself through the dynamics of this ratio and the consistency of the latter.

Economic justification of investments in the given research is conducted in conventional units with the aim to show its efficiency and to reflect traditional for Ukraine correlations of different types of expenses by the example of one of mining enterprises of Ukraine that is located in Rivne region – Private Joint-Stock Company "Rafalivka basalt quarry".

According to conducted analysis of external environment we can arrive at decision that the enterprise is insufficiently using the existing manufacturing resources. Beside the market where it sells own products is narrowing. That is why the enterprise has to strengthen own positions and to decrease the dependence on changes in sales of the main types of products. In such case it is reasonably to apply innovative approaches in the enterprise in order to ensure survival and development in long-term perspective.

In order to improve market position of the enterprise it is necessary to analyse innovative capabilities of the enterprise. According to the results of SWOT analysis in the Private Joint-Stock Company "Rafalivka basalt quarry" were detected unused capabilities of the enterprise, namely the presence of copper in the quarry deposits and absence of technology of its extraction, also to the strong points of the enterprise it is possible to refer physical and mechanical peculiarities and chemical composition of basalts, which gives the possibility to expand the range of products.

Basalt quarry

cho-pping	grin-ding	fusion (low temperature)			fusion (high temperature)	fusion (with extraction)		
break stone	pow-der	fou-ndry production	coarse basalt fiber	basalt scale	STBF	continuous fiber		
road du-sting	pow-der metallurgy	casting products	fibro-basalt-cement technologies	anticorrosion coating	insulator	fabric manufacturing techni-ques	rope produc-tion	basalt plastic techno-logies
	wa-shers	foa-med basalt	tubes, roofing mate-rials (asbestos slate, roofing tile)		wired mats	geotex-tiles	for river and sea vehicles	tubes of different industrial designa-tion
	isola-ting device		insula-tor		filters	refractory fabric	for cargo fixation	oil, supply, main, sewage pipes
					vehic-le air silen-cers, brak eshoes	suits for firemen	for lifting mechanisms	electroin-sulating materials
					fracti-onal mate-rials	finishing materials for saloons, automo-bile interiors, cabins, boats, plane cabins		industrial and domestic electric heaters
						knitted textile shells for reinfor-cing of tunnel space		tanks for radioactive waste manage-ment

Fig. 1. Scheme of technological capabilities of basalt usage

The analysis of basalt processing technologies demonstrates that there are wide enough opportunities for its processing. Physical and mechanical characteristics and chemical composition of basalts enable their application in various economy branches (fig.1).

Usually the basalt is crushed to crushed stone and is mainly used for road paving. That is, the processes of crushing or shredding are carried out.

But outside of Ukraine wide development receive technologies on basalt melt on manufacturing of thermal insulating materials, pipes, a basalt thread, a tile and other.

Conditionally, basalt casting can be divided into three types:

1. Fusion (low temperatures):

1.1. Foundry production (casting products, foam basalt);

1.2. Coarse basalt fiber (fiber basalt-cement technology-pipes, roofing materials (cipher, roof tile), insulation);

1.3. Basalt scale (anticorrosive coating).

2. Fusion (high temperatures):

2.1.Super-thin basalt fiber (STBF) (a sheet of super-thin basalt fiber), insulant, filters, vehicle sound dampers;

2.2. Fiber basalt plastic technologies - pipes for various purposes, fractional materials, brake pads for vehicles and railway transport, for lifting mechanisms of various industries.

3. Fusion with extraction (continuous fiber):

3.1. Textile manufacturing technology (geotextile materials, refractory fabrics: firefighter suits, covering materials for car showrooms, ship cabins, aircraft showrooms, living quarters, offices, knitted textile shells for strengthening the tunnel goaf);

3.2. Manufacture of ropes (for sea and river transport, lifting mechanisms of various purposes, for fastening of cargoes during transportation);

3.3. Basalt-plastic technologies (pipes of different purpose: oil-gathering, drainage, pipeline, sewerage, water supply; electrical insulating materials: stamping plates, domestic and industrial electric heaters, cylinders and containers for various purposes, containers for disposal of radioactive waste, basalt cement reinforced constructions).

A lot of basalt is exported to Poland, where it is used for basalt casting instead of on highway “pillows” laying. The calculation is simple: finished basalt products cost at least several hundred times more expensive than raw materials. The most economical is the production of high-tech products. Thus, the production cost of one kilogram of basalt crumb, taking into account all the expenses is equal to USD 0.01, while the same amount of basalt thread costs USD 1.3, and one square meter of thermal protective fabric - USD 6-8.

Thus, physicomechanical and chemical properties of basalts of PJSC “Rafalivka basalt quarry” allows its usage for organization of new productions and production types that will enable the company to take a competitive position in the market.

We have analyzed the opportunities for native copper producing. The first findings of native copper were found in Volyn basalts in the 30s of the last century. Since then, their unique origin has been intensively studied, exploration work on copper ores has been carried out and areas of rich copper mineralization have been identified. One of the most promising is the Rafalivka ore cluster in the Rivne region, established by the Rivne geological expedition.

Technologists at Rafalivka basalt quarry carried out research work and concluded that the basalt deposits have a trappean structure, the main components of which are layers of basalt, lava breccias and tuff. And the highest native copper content is observed namely in lava breccia (Fig. 2). In basalts and tuffs, copper is present in the form of veins and inclusions. When chopping, it can be separated from the enclosing rocks.

During the research, copper pick-ups from 0.1 to 1.4 kg were found. In basalt and tuff, copper contains in the form of veins and inclusions.

Copper holds one of the leading positions both in production volume among metals and in industrial consumption volume among industries of developed countries. At the current stage of the world economy development, there is a tendency for increasing the rates of copper consumption, which is also peculiar for Ukraine (metallurgical complex, mechanical engineering, electrical engineering, communications, electronic industry, etc.). The annual demand of the Ukrainian economy for copper is huge and, according to the Program of Non-ferrous Metallurgy Development of Ukraine, is up to 200 thousand tons, the cost of which according to various estimates can be UAH 30 billion per year. Consumption of copper in Ukraine is based on imported and domestic secondary raw materials.

Fig. 2. Modified basalt fragment with a leaf-shaped copper pick-up (on the left) from the Rafalivka basalt quarry

Now Ukraine has no explored copper deposits, but the prospect of creating its own mineral resource base of this metal is quite significant, which is associated with deposits of native copper at PJSC "Rafalivka basalt quarry".

Therefore, for PJSC "Rafalivka basalt quarry" this area of development is promising for ensuring competitive advantages in the long term. After all, copper in Ukraine is a unique product.

Copper is extremely important for electro technical and other industrial sectors. At present time, the deficit of copper in Ukraine is sorely felt, that is why the works on searching for this important nonferrous metal have been activated. The considerable needs of copper in Ukraine are met by its imports (currently about 250 thousand tons per year). Copper is not mined in Ukraine - only geological exploration works are carried out. Within the country, there are enterprises that are withdrawing copper from secondary resources. This refers to electrical, radio and electronic equipment or parts thereof that have reached the end of their working life. The cost of copper in this case is high due to the labor intensity of its extraction. In addition, the chemical purity of this copper is low, and in this case the matter is more about alloys. Copper is widely used in the manufacture of products for various purposes: pipelines, chemical equipment, electrical devices, etc. Copper is resistant to oxidation, bacteria and viruses and it detritions slowly. Copper pipes began to be used more actively due to their mechanical properties, low density of pipe walls and simplicity of installation. The copper pipes connection is simple and reliable. It is believed that copper is perfectly resists corrosion. In terms of its electromechanical properties it is the same as noble metals. Copper pipes also prevent the bacterial growth in water. This is extremely important, especially for systems that supply hot drinking water. Application of copper is advisable in systems supplying both cold and hot drinking water, heating systems, solar energy storage systems, gas and liquid fuel supply systems, refrigeration industry. This means the multiversity of copper use, which is associated with its special physical properties.

There is no environmental threat from native copper mining. In fact, quite a simple technological scheme of its processing is used: shredding, washing and gravity treatment, during which heavier metal particles settle, as shown in figure 3.

The tuff of quarry mining, being in the refuses, has a particle size up to 200 ÷ 300 mm, and after the softening - 50 ÷ 100 mm. For further processing it needs to be crushed and shredded. On the basis of the analysis of tuff particle size composition after crushing and shredding, results of softening, there was developed a scheme of ore preparing of tuff for complex processing taking into account its strength, mineral composition and necessary particle size composition. Scheme shown in the figure 3, consists of a chain of successive devices: receiver cone 1, vibrating feeder 2, jaw crusher 3, belt loader 4, roll crusher 5, vibrating screen 6, belt loader 7, roller mill 8, fine-shredding vibrating screen 9, belt conveyor 10 magnetic separator 11 and electric separator 12.

The selection of equipment parameters is made on the basis of the obtained research results. For example, based on the maximum magnetic susceptibility of tuff (1.3 tesla), the magnetic separator should provide this strength of the field. Roller mill and vibrating screen should ensure a particle size of 0.3 ÷ 0.1 mm for effective magnetic and electrical separation. The design values of the roller crusher (diameter of the rollers, clearance between them, speed) are determined depending on the established original tuff size. The parameters of the jaw crusher are also selected based on the same considerations. The efficiency of the scheme has been tested in laboratory conditions. In the course of research it was determined that as a roller mill 8, it is advisable to use an intra-roller cone vibratory mill designed by the IGTM NAS of Ukraine, since it will prevent significant fragmentation of tuff (less than 0.1 mm) by selecting the winding pitch and height of the cone. As it has been established earlier, such fragmentation sharply reduces the efficiency of magnetic and electrical separation [30].

For operations of small and fine classification of shredded tuff to the necessary for effective magnetic and electrical separation in the laboratory conditions of modeling the technological line were used vibrating screens with dynamically active screening surface and additional metal net with the necessary size of cells. The screen of the IGTM NAS of Ukraine design showed efficiency up to 85-90% in the class of particle size of 100 microns, and it is recommended for equipping the presented technological scheme.

Associated products of copper extraction are **basalt** (basalt chips, sifting) and **tuff**.

For a long time tuffs were not in practical demand and during the exploitation of basalt deposits they were moved to dumps. At this time, it was found that due to the significant content of sorbents (50-60%) tuffs demonstrate important sorption, selective and cationic exchange capacities and can be used as:

- impurities in the feed concentrates for the purpose of sorption of artificial radionuclides, removal of toxins from animals' bodies, supplementary feeding of feed ration with microelements deficient for the Polissia zone (copper, vanadium, chrome, manganese, etc.);
- as natural inorganic sorbent - ameliorant during decontamination of soils in the zone of radioactive contamination;
- as a natural dye - filler in the manufacture of oil and silicate brown paints.

For practical use, tuff raw materials do not require pre-equipment. The near-surface conditions of occurrence allow surface tuff mining, first of all, from the bottom of the pit without expenses for overburden removing.

The availability of access roads and mining infrastructure at the existing quarry also makes tuff exploitation cheaper.

Thus, the Rafalivka basalt quarry has quite unique resource deposits of native copper, both in economic and environmental aspects.

If to pay attention to the fact that copper is not mined in Ukraine at all (the country's demand is met by exports), then such activity will provide the company with the future development prospects.

Mastering of the copper mining technology, which lies in the deposits of the quarry, is extremely important to ensure the financial stability of the enterprise, and obtaining competitive advantages and development in the longer term.

The main problem is the presence of native copper in the basalt deposits of the quarry and the lack of technology for its mining and processing.

The mastering of this technology will be able to provide perspective stable development of the enterprise in the future.

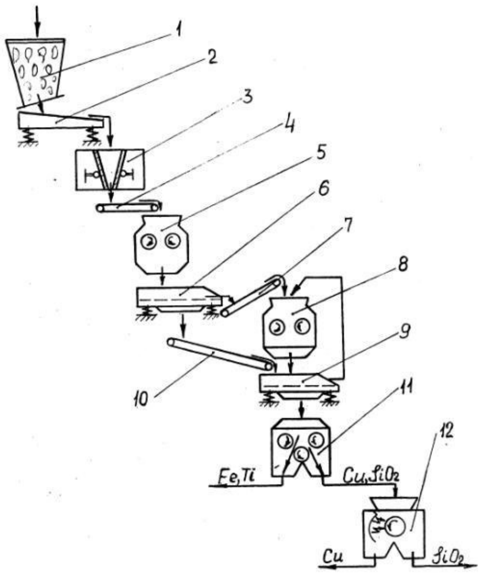

Fig. 3. Tuff ore preparation scheme

Consider several alternatives of mastering native copper mining technology.

- **Proprietary solution of native copper mining technology**

For mastering technology for the mining of native copper and related minerals, the enterprise can begin to develop such technology on its own by conducting research and development and trial manufacturing. After all, during the research work experts and geologists justified a quite simple technological scheme of its processing: shredding, washing and gravity treatment, during which heavier metal particles settle.

Advantage: application of this alternative will provide the company with its own technology.

Disadvantages: low professional-qualification level of specialists of the enterprise in this direction, high expenses for staff training and research and development, high risk of failure to obtain the desirable results.

- **Purchase of a patent for the native copper mining**

In Ukraine there are no patented technologies for copper mining, there are only geological exploration works, according to which it was established that deposits of native copper were found in Volyn and Polissia.

There are only two known deposits of such copper in the world - near the Great Lakes in the USA and in China, respectively, there exists such technology, developed and patented. Therefore, it is possible to purchase a copper mining patent from these manufacturers.

Advantages: Purchase of the patent will save the time required to carry out research and development and will enable to start copper mining immediately, to train employees of the quarry with involvement of highly professional specialists from abroad.

Disadvantages: such an alternative would require large investment of funds - attraction of significant credit resources. Today in the conditions of economic instability, it is quite risky for the enterprise to take big credits, since the enterprise does not have own funds necessary for realization of such variant of actions. Purchase of such a patent and personnel training will cost the company a lot.

But there is a great risk associated with the fact that the technologies in the USA and China were developed in accordance with their quality characteristics of basalt lava. For example, the copper content in ore concentrate in the USA is 99.5% and in the Rafalivka basalt quarry is only between 0.2 and 5%. Therefore, the applied technology may not be suitable for the mining conditions at PJSC "Rafalivka basalt quarry".

At the time of technology development for the native copper mining were issued Ukrainian patents and certificates of authorship (declaration patent of invention No. 59882A. Ukraine. Gravitational magnetic separator MKI 7B 03C 1/04, declaration patent of invention No. 50098A. Ukraine. Screen. MKI 7B 07B 1/40, declaration patent of invention No. 70428A. Ukraine. Vibrating screen. MKI 7B 07B 1/46). In recent years, scientists of IGTM NAS of Ukraine and NUWEE together with employees of PJSC "Rafalivka basalt quarry" were searching for investors to engage in the manufacturing process.

The investment required for the realization of this project is 10 million common units. In this case, the undistributed profit of PJSC "Rafalivka basalt quarry" may amount to 1.767 million c.u., of which

1.5 million c.u. may be attracted to the project.

Thus 17% of received profit from realization of investment can be credited to the account of PJSC "Rafalivka basalt quarry", thus the administration of quarry will undertake organizational and legal issues on realization of production. Preliminary expenses for implementation of the future investment proposal are given in Table 2.

Table 2

Cost estimates

Ser. No	Name of the expense component	1	2	3	4	5	Total
1	Expenses, in total	9813,75	3615,94	2419,60	2391,82	2378,50	20620
2	incl. equipment purchase	5250					5250
3	equipment installation and manufacturing process setup	900	200				1100
4	research and development, preparation of pilot production	1500	950				2450
5	Operating expenses	1908,75	2085,94	2039,60	2011,82	1998,50	10045
6	Non-production expenses	255,00	380,00	380,00	380,00	380,00	1775

The equipment needs for the establishment of the production process are given in Table 3.

Table 3

Equipment needs

Ser. No	Name	Measurement unit	Number per project	Price per unit, thousand c.u.	Value, thousand c.u.
1	2	3	4	5	6
1	Jaw crusher	pcs	2	150	300
2	Centrifugal crusher	pcs	2	150	300
3	Vibrating screen	pcs	8	70	560
4	Conveyer	pcs	5	30	150
5	Mechanical loader	pcs	2	150	300
6	Electrostatic separator	pcs	4	100	400
7	Wash box	pcs	5	25	125
8	Mill	pcs	4	100	400

Continuation of table. 3

1	2	3	4	5	6
9	Truck	pcs	2	300	600
10	Equipment kit for equipment analysis	pcs	2	200	400
11	Overpasses for equipment and foundation	pcs	18	65	1170
12	Bin	pcs	14	10	140
13	Blasting lead	pcs	15	17	255
	Total				5250

The total capital investment is 10000 thousands c.u. The total capital investment is 10000 thousands c.u. Of which the equipment purchase amounts to 5250 thousand c.u., its installation and adjustment of the production process amounts to 900 thousands c.u. and 200 thousand c.u. respectively in the first and second year of the project implementation. Expenses for the implementation of research and development works and preparation of trial production in the first year will amount to 1500 thousand c.u. (investments of the quarry), and in the second year - 950 thousand c.u.

Operating and non-production expenses of innovative products manufacturing are given in Table 4.

Table 4

Operating and non-production expenses of innovative products manufacturing

Ser. No.	Name of the expense component	1	2	3	4	5	Total
1	**Expenses, in total**	**2163,75**	**2465,94**	**2419,60**	**2391,82**	**2378,50**	**11819,6**
	including						
2	**Operating expenses**						
2.1	Wages	540,00	720,00	720,00	720,00	720,00	3420
2.2	Social contributions	202,50	270,00	270,00	270,00	270,00	1282,5
2.3	Water and Wastewater Treatment Plant	10,00	10,00	10,00	10,00	10,00	50
2.4	Electricity, heating	200,00	250,00	300,00	350,00	400,00	1500
2.5	Drilling and blasting services	200,00	200,00	200,00	200,00	200,00	1000
2.6	Preliminary overburden removing	100,00	100,00	100,00	100,00	100,00	500
2.7	Depreciation of fixed assets	656,25	535,94	439,60	361,82	298,50	2292,11

Continuation of table.4

2.8	**Total Operating expenses**	**1908,75**	**2085,94**	**2039,60**	**2011,82**	**1998,50**	**10044,6**
3	**Non-production expenses**						
3.1	Administrative expenses	145,00	145,00	145,00	145,00	145,00	725
3.2	Advertising expenses, marketing		125,00	125,00	125,00	125,00	500
3.3	Staff training	95,00	95,00	95,00	95,00	95,00	475
3.4	Environmental Protection	15,00	15,00	15,00	15,00	15,00	75
	Total non-production expenses	**255,00**	**380,00**	**380,00**	**380,00**	**380,00**	**1775**

Estimated production and sales volumes of innovative products in natural indicators are given in Table 5.

Table 5

Expected production and sales volumes of innovative products in natural indicators

No	Innovative product name	Measurement unit	1 year	2 year	3 year	4 year	5 year	1-5 years
1	Basalt	ton	4000	6000	8500	8500	8500	22000
2	Native copper	ton	150	200	250	300	350	1100
3	Tuff	ton	900	1500	2000	2000	2000	8000

We accept that the following price is formed in the market: basalt - 60 c.u./t, copper - 60000 c.u./t, tuff - 40 c.u./t. The expected volume in cash equivalents is calculated in Table 6.

Table 6

The expected sales volume of innovative products

Years	Number of units, t.	Price per unit, thousand c.u.	Value of products, thousand c.u.
Basalt			
0	4000	0,06	240
1	6000	0,06	360
2	8500	0,06	510
3	8500	0,06	510
4	8500	0,06	510
0-4	34500	0,06	2070
Native copper			
0	150	60	9000
1	200	60	12000
2	250	60	15000

Continuation of table.6

3	300	60	18000
4	350	60	21000
0-4	1100	60	66000
Tuff			
0	900	0,04	36
1	1500	0,04	60
2	2000	0,04	80
3	2000	0,04	80
4	2000	0,04	80
0-4	8400	0,04	336

Having the planned product sales volumes and current prices in the market it is possible to predict the expected income of the technology park and Rafalivka basalt quarry (Table 7) and calculate the economic feasibility of the project (Table 8 and Table 9).

Table 7

Expected income from sales of innovative products, thousands c.u.

Elements	1	2	3	4	5	Total
Expected income	9276	12420	15590	18590	21590	74466,0
Investments into the project	6500	3500				
including credit	5000	3500				
investments of PJSC "Rafalivka basalt quarry"	1500					
Manufacturing charges	2163,7	2465,9	2419,60	2391,82	2378,50	11819,6
Credit repayment by technology park	1000,0	1875,0	1875,00	1875,00	1875,00	8500,00
credit %	1050	1575	1181,25	787,5	393,75	4987,50
Total credit payments	2050,0	3450,0	3056,25	2662,50	2268,75	13487,5
Income after % repayment	5062,2	6504,1	10114,15	13535,7	16942,8	52158,9
Profit of PJSC "Rafalivka basalt quarry" (17%)	860,58	1105,7	1719,41	2301,07	2880,27	8867,01

Table 8

Expected present income for technology park

No	Elements	Measurement unit	Years					Total
			1	2	3	4	5	
1	Project income	thousands c.u.	4201,67	5398,37	8394,75	11234,61	14062,48	43291,88
2	Project expenses	thousands c.u.	5000,00	3500,00				8500,00

Continuation of table.8

3	Coefficient of discounting (20%)	coefficient	1,00	0,69	0,58	0,48	0,40	3,15
4	Discounted income	thousands c.u.	4201,67	3724,88	4868,95	5392,61	5624,99	23813,10
5	Discounted expenses	thousands c.u.	5000,00	2415,00				7415,00
6	Net present value	thousands c.u.	-798,33	1309,88	4868,95	5392,61	5624,99	16398,10

So, the net present value of the project for the technology park is 16398.1 thousand c.u.

The calculation of net present value (NVP) of the project for PJSC "Rafalivka basalt quarry" is given in Table 9.

Table 9

The calculation of NVP of the project for PJSC "Rafalivka basalt quarry"

No	Elements	Measurement unit	Years					Total
			2011	2012	2013	2014	2015	
1	Project income	thousands c.u.	860,58	1105,69	1719,41	2301,07	2880,27	8867,01
2	Project expenses	thousands c.u.	1500,00					1500,00
3	Coefficient of discounting (20%)	coefficient	1,00	0,69	0,58	0,48	0,40	3,15
4	Discounted income	thousands c.u.	860,58	762,93	997,26	1104,51	1152,11	4877,38
5	Discounted expenses	thousands c.u.	1500,00					1500,00
6	Net present value	thousands c.u.	-639,42	762,93	997,26	1104,51	1152,11	3377,38

Fig. 4 shows the dynamics of net present value for the PJSC "Rafalivka basalt quarry"

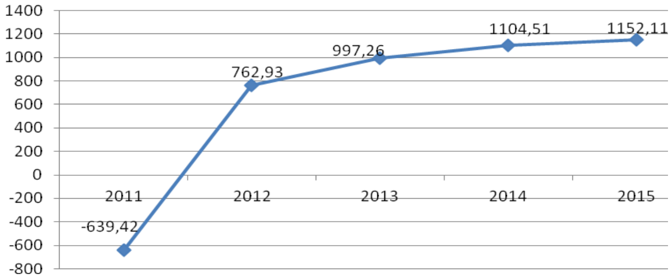

Fig. 4. The NVP dynamics of PJSC “Rafalivka basalt quarry” project

Net present value of the project is more than zero, the profitability index of the project is 3.2, which is more than 1, and the payback time is 1.9 years, which indicates the feasibility of the project.

After the termination of the agreement, the terms of the cooperation may be reviewed. By that time, it is possible to buy the technology and continue in-house production, or to continue joint ventures on the basis of the interest receiving from the net profit, depending on the assigned responsibilities.

Thus, the accompanying mining and sales of copper will allow the company to expand its activities and minimize the long-term risk, which will significantly improve the economic condition of the company, and Ukraine will receive additional copper concentrate in addition to basalt products.

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TECHNOLOGY OF PRODUCTION AND PROCESSING OF PEAT AT ENTERPRISES OF THE RIVNE REGION

Moshynskyi V.S.
National University of Water and Environmental Engineering (NUWEE), Professor, Doctor of Sciences (Agricultural), Professor, Department Of Land, Cadaster, Land Monitoring And Geoinformatics, Ukraine

Solvar L.M.
National University of Water and Environmental Engineering (NUWEE), Senior Lecturer, Institute of Postgraduate Education, Ukraine

Semeniuk V.V.
National University of Water and Environmental Engineering (NUWEE), Senior Lecturer,
Department of Development of Deposits and Mining, Ukraine

Kucheruk M.O.
National University of Water and Environmental Engineering (NUWEE), Assistant, Department of Development of Deposits and Mining, Ukraine

Summary
This section analyzes the development of Ukraine's peat industry. Attention is drawn directly to the prospects of the peat industry of Volyn, in particular the Rivne region. The technologies of peat extraction and processing at the enterprises of Rivne region are described in detail, their advantages and disadvantages are considered, taking into account losses in the process of development on the example of peat deposits of Rivne region. An alternative path and recommendations for more rational development of industrial reserves of peat deposit with reduced economic costs have been developed.

Introduction

The peat industry of Ukraine was started in 1919. Until the mid-1960s, peat was used as fuel for power plants, sugar and alcohol factories, schools, hospitals, and in part for the general population. With the development of gasification and electrification, peat began to lose its position as an energy resource. But peat briquette factories continued to be built, as this type of fuel was extremely popular with non-gas consumers [1, 2].

Ukraine's peat deposits are represented mainly by lowland type (96% of all resources). There are also small (1.8%), transitional (1.6%) and mixed (0.6%) types of peat deposits. By the nature of peat accumulation and features of peat deposits in the territory of Ukraine, the following peat regions are distinguished: Polissia, Małopoliska, Lisostepova, Stepova, Carpathian. The most favorable conditions for peat accumulation occurred in the post-glacial period in Polissia, where numerous and diverse peat bogs were formed. The peat reserves and resources of this region (Volyn, Rivne, Chernihiv administrative regions) make up almost half of all peat stock of Ukraine [3].

According to geological-geomorphological features, 11 peat-marsh regions are distinguished within the peat-marsh regions, each of which is characterized by a certain degree of wetland and peat land. The most swampy (10.1%) and peaty (7.3%) districts of Western Polissia in the peat-swamp region of Polissia [4].

In Ukraine, most swamps are peat lands. The latter term is often used for drained marshes; sometimes under peat land understand the peat bog, especially when developing it. In the marshes of Ukraine lowland peat deposits prevail, in the Western Polissia and Carpathians mixed transitional types occur. Top and mixed top types of deposits are known in Western and Central Polissia, in the Carpathians. The most widespread peat deposits are in Rivne, Volyn, Chernihiv, Zhytomyr Kyiv, Lviv regions. Tariffs of Rivne and Volyn regions reach 6.5%, while in Ternopil, Khmelnitsky, Vinnitsa, Cherkasy, Poltava, Sumy and Kharkiv regions it does not exceed 1.9% of the whole territory. Even less common are peat deposits in Mykolaiv, Zaporizhia, Dnipropetrovsk, Transcarpathian, Ivano-Frankivsk regions, where the degree of peat land does not exceed 0.1% [4].

The Ukrtorf concern produces up to 600,000 tons of peat annually, the vast majority of which (62%) is converted to fuel briquettes - a fairly efficient solid fuel with a lower combustion heat of about 15 MJ / kg, humidity up to 20% and ash content up to 23 %.

About 50% of Ukraine's peat reserves are concentrated in three oblasts - Volyn, Rivne, and Chernihiv. The geological peat reserves in Ukraine are estimated at 2.2 billion tons. The total area of the fields is about one million hectares, including 0.6 million hectares within industrial depth; Balance of peat reserves exceeds 838 million tons. Peat reserves in industrial fields are 22.6 million tons, and the prepared industrial capacity for peat extraction is 2.1 million tons (for the production of peat briquettes - 0.7 million tons).

In general, the largest number of balance reserves is in the Volyn region (165 million tons), which is about 21% of all industrial deposits of Ukraine, the second place is taken by the Rivne region with reserves over 133 million tons, or 17% from the national ones (Fig. 1). .

The most promising region for the construction of mining and peat processing enterprises is Polissia. Deposits operated by the Ukrainian Peat Industry Concerns of Ukrtorf concentrate 15% of explored reserves, 28% of reserves, prospects for exploration - 18%, security (those located within protected areas) - 11%, and drained ones - 17%, for saline (Ac> 35%) - 5%, for small deposits - 6% [5].

The most reserve and prospecting for peat deposits is in the Rivne region. According to the State Geomology, 2168 peat deposits were discovered and explored in the territory of Ukraine. About 100 thousand hectares of peat lands belong to the objects of the nature reserve fund.

The state concern Ukrtorf is engaged in the extraction and processing of peat in the country, in the structure of which there are the following state enterprises: "Volintorf", "Zhytomyrtorf", "Kyivtorf" (operates in the territory of three oblasts - Kyiv, Poltava and Cherkassy - one peat plant), Podillatorf (Khmelnytskyi, Vinnytsia and Ternopil region), Rivnetorf (one plant for the production of Smigatorf peat briquettes), Sumy and Irvantsivsky [6].

During the formation of peat deposits, peat deposits, depending on water and mineral nutrition, vegetation, climate and when they change, may undergo certain stages associated with the accumulation of different types of peat. Therefore, in nature there is a great variety of stratigraphic features of peat deposits.

The primary unit of classification is the type of peat [7]. Currently, 150 species of peat have been identified, of which 65 are lowland, 41 are transitional, 44 are upland. All species are grouped into 6 groups - woody, herbaceous, mossy, grassy, grassy-mossy, mossy.

These six groups are grouped into three subtypes - forest, forest-drab, and driftwood. And the subtypes form 4 types of peat deposits - lowland, upland, transitional and mixed.

Peat is a layering of peat species, currently there are 90 varieties of peat deposits.

The lowland type of deposits is deposits with soil nutrition, so the content of mineral salts in the peat. Ash content is different: from 15-25, up to 30-40%.

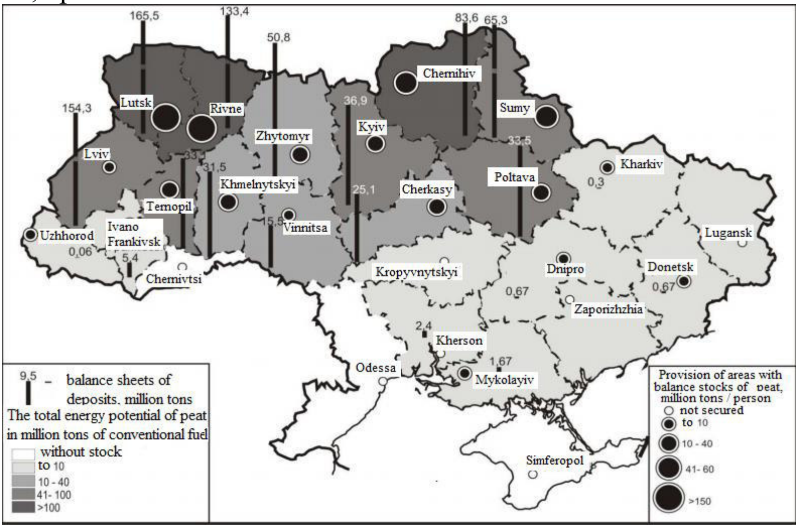


Fig. 1. Distribution of peat resources in the territory of Ukraine

High-ash lowland fields after hydro melioration are more appropriate to use in agriculture as forage or arable land. In its composition lowland peat contains lime and peat vivanite, which promotes high fertility of agricultural crops. Lowland peat with high ash content, carbonate and phosphorus content and naturally low acidity is also a valuable fertilizer or component for the production of plant growth stimulants, soil mixtures for the cultivation of flowers, vegetables, mushrooms, the basis for the manufacture of composts and the like.

Lowland peat deposits include deposits formed wholly or largely from lowland peat.

Horse fields are fed mainly by precipitation, so the peat bogs have low ash content, are rich in bitumen (9 to 15%) and have high acidity. Horse-type peat is used in animal husbandry for litter; for the production of coke, semi-coke, wax, fodder yeast, are important components for the production of light soil mixtures and substrates for many types of ornamental, vegetable, flower, seedling, seedling, plant growth stimulants, etc.

Horse-type peat deposits are widespread only in the West Polissia peat area and in the Carpathians. The main stocks of horse peat have been discovered at the deposits of Rivne region [8].

On peat deposits of transitional type of nutrition surface-sewage, so the deposit is composed of different types of peat, the bottom - lowland species, and the lowland floor - the top. Deposits with transitional peat deposits are usually found in small and shallow basins and valleys.

At mixed-type deposits, the deposit is represented by a layer of different types of peat - lowland and transitional, covered by a top, with a capacity of not less than 0.5 m.

Territorial congestion is one of the most important indicators. It characterizes not only the share of peat covered area, but also the relative value of peat reserves that occur and spread, and therefore the relative abundance of peat. And if the price of the territory of Ukraine does not exceed 1.7%, the figure for the Volyn, Rivne and Chernihiv regions is 8.25, respectively; 9.30; and 3.72%, and in the Polissia districts of these regions and Zhytomyr region it reaches 15%.

Most types of peat found in the territory of Ukraine are suitable for the production of organic fertilizers, mainly peat-based composts, animal bedding, plant growth stimulants, soil mixtures, peat cups for seedlings and more.

Given the limited stock of peat in Ukraine, as well as the situation on the fuel market in our country, we can agree with the statement that the main use of peat in modern conditions is the production of peat fuel (peat briquettes and lump peat) for household consumption. It is necessary to study in more detail the feasibility of introducing small peat-fired power plants, based on the experience of Finland, where about 15% of electricity is generated by its combustion [9].

Unfortunately, in the presence of such diverse peat raw materials, over 80% of Ukraine's peat, regardless of its properties, is burned, while in the world more than 80% of peat is used in agriculture as a source of humus.

The Rivne region ranks second among all regions of Ukraine in terms of the number of peat deposits and reserves (Fig. 2) [10]. With a total area of 20.1 thousand square km, the area of peat deposits within the limits of industrial depth of deposits is 1.33 thousand square km or

6.5% of the total. The peat fund of the region consists of 332 peat deposits with a total area within the limits of industrial depth of 133.6 thousand hectares and geological reserves of peat 376.9 million tons or 17% of the total in Ukraine, of which: balance reserves amount to 188.7 million tons, on-balance sheet - 173.2 million tons [11].

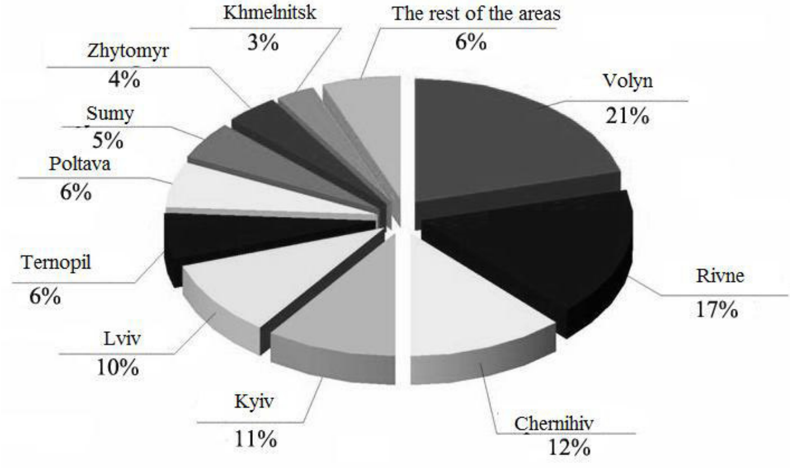


Fig. 2. Distribution of peat reserves by regions of Ukraine

By administrative districts, the distribution of peat reserves and the peat land are uneven. In Rokytne district, peat resources make up 67.3 thousand tons, their area of distribution is 28470 hectares, the territory's congestion is 12.5%; in the Dubrovytskyi district, geological reserves amount to 63066 thousand tonnes, area - 25046 hectares, territorial congestion - 13.5%. In the southern regions, for example, in Radyvylivskyi, the total area of peat deposits is 544 ha, reserves - 2467 thousand tons, and peat land - 0.6%; in Koretskyi, the total area of peat deposits is 141 hectares, the peat reserves are 198 thousand tons, and the territory is 0.1%.

More than 50% of the discovered and explored peat stock is deposits with an area less than 100 hectares, but the main reserves are concentrated on 49 peat deposits with an area over 500 hectares [1, 10, 11].

More than 80% of the peat fund explored is low-lying peat reserves. But at the same time, a considerable share is made of peat deposits of transitional and transitional types, which occur on 25 peat deposits and cover an area of about 15.0 thousand hectares, and reserves of peat type -

more than 50 million tons. These peat deposits are grouped in northwestern part of the region. The most significant of them are the fields: Morochno I, which has an area within the limits of industrial depth of 5.1 thousand hectares and geological reserves - 15.86 million tons; Morochno II - with an area of 4.7 thousand hectares and geological reserves of 12.6 million tons.

The peat reserves of the region by types of deposits are distributed as follows: lowland - 80%, high - 9.7%, transitional - 6.2%, mixed - 4.1%.

According to the degree of study of peat reserves are divided into exploration categories A, B, C1 and previously explored - categories C2.

The degree of exploration of the peat stock of the region is low. Of the total peat stock of the region, 25% of the fields have been explored in detail and the reserves have been approved under category A.

The remaining 75% of the fields have been explored by preliminary and route exploration, as well as surveyed; reserves - approved under categories C1, C2.

The largest peat deposits by area are explored in detail, they have 168.3 million tons of peat reserves, which is 44.8% of the total reserves of the region.

Peat deposits are spread throughout the region. Their greatest concentration is observed in the northern part of the region, in the valleys of the rivers Styr, Goryn, Sluch, Sviga, and others - in the right tributaries of the Pripyat River, which represent the western part of the Ukrainian Polissia.

For further development, the first three groups of deposits are the most promising: operational, reserve and prospecting for exploration, which are concentrated in 80 deposits (Table 1).

Table 1

Summary table of peat reserves of Rivne region on peat deposits with area over 10 hectares

The degree of industrial development	Number of deposits	Area within industrial depth, ha	Peat reserves, thousand tons	
			$A+B+C_1+C_2$	Off-balance sheet
1. Operating	46	21410	61527	5588
2. Backup	21	16324	48059	3304
3. Prospective for exploration	13	16682	30900	13189
4. Rest	252	73469	47261	151315
4.1. Security	15	29892	-	77933
4.2. Dried	127	19027	-	65154
4.3.For the greens	16	1898	-	5761
4.4. Shallow	94	22652	47261	2467
Total	332	127885	187747	173396

The region's peat reserve fund consists mainly of a number of large peat deposits of industrial importance.

Peat mining distinguishes between the following methods of open-cast mining: layer-by-surface, layer-by-slot and quarrying.

At present, the layer-by-surface or milling method of extraction is dominant in the region.

In this method, the final production is a milling crumb, characterized by the following indicators: type of deposit; the degree of decomposition of the deposit layer; moisture content; ash content; specific heat of combustion; clogging with wood, scrapes and other extraneous inclusions; bulk density; content of small fraction up to 1 mm; the average diameter of the particles.

Milling peat is used as a raw material for the production of the following products: fuel; fertilizer; litter; pots and packing material; activated carbon; yeast; mountain wax.

When choosing a type of product, the following indicators are taken into account: type of deposit, degree of decomposition (R) and ash content (Ac) (Table 2).

Depending on the type of products they change. When choosing a product type, it should be borne in mind that the characteristics of the deposit vary both in area and capacity [12].

Table 2

Requirements for peat as a raw material for the manufacture of various products [12]

№	Type of products	Lowland peat	Horse and transitional peat
1	Fertilizer	R – from 15% A^c – to 35% CaO – from 10% P_2O_5 – from 1%	R – from 15% A^c – from 35%
2	Fuel	R – 10% and above A^c – to 35%	R – from 20% A^c – to 23%
3	Mountain wax	All types of peat A^c – to 10%, R – from 30%, the main criteria are the content of gasoline bitumen – 5%	
4	Yeast	-	R – from 35% A^c – to 6%
5	Activated carbon	-	R – from 35% A^c – to 6%
6	Litter	All types of peat R – from 15% - 20%, A^c – to 15%	
7	Pots and packing material	All types of peat R – from 10% to 25%, A^c – to 15%	

Currently in the area of milling peat mainly produce fuels and fertilizers.

Peat production is carried out by the State Enterprise Rivnetorf of the Ukrtorf State Concern of the Ministry of Fuel and Energy of Ukraine.

SE “Rivnetorf” in different years consisted of structural units that worked on the following raw materials bases: “Volodymyretstorf” - Gryada; Mokwintorf - Sand; Bereznetorf - Dike Halo; Chemernetorf - Korabelskoe; "Smigatorf" - Starniks; "Klesivtorf" - Kreminne, "Verbatorf" - Willow I.

As of 1 January 2020, only the Smigatorf unit, which performs mining operations at the Starniki peat field, remained in the structure of Rivnetorf SE. For all other reasons, all other units are liquidated or are in the process of being liquidated, but in the future the extraction works can be resumed in the listed deposits.

Two types of milling peat are produced at Rivnetorf: fuel and agriculture. From the fuel milling peat produce semi-briquettes, which are used as municipal fuel.

In the milling method of peat extraction, products are obtained in the form of a loose mixture of small particles, different in size, but not more than 20-25 mm. The milling method derives its name from the initial milling operation. Milling - the process of processing peat deposits with mills. The cutter is a tool with cutting elements (knives); milling cutters used in metal and woodworking industries are an example. Rotating around their own axis and sinking into the stock at a progressive stroke, they remove a small thickness of the layer, turning it into a crumb.

In the milling method, the formation of the reservoir is performed from the surface by horizontal layers, so it is also called a layer-by-surface method of development. Drying of the milled crumb is carried out in the same area, where milling was performed in natural open-air conditions due to the use of solar energy and heat of the air masses. This method depends on the weather and is seasonal. In order to accelerate the drying and obtain a more homogeneous moisture content of the finished products is used to stir the drying layer. Dried to the required humidity, the milling peat is collected from the spread of pre-prepared rolls in the stack of the correct triangular or trapezoidal section. Peat comes into the stack directly from the harvesting machine or from the piles at the foot of the stack created by the bunker harvesters. From the heaps to the pile, the peat moves the stacking machine and evens it on the surface with an even layer.

Production of FT is determined by the schemes of storage of finished products. Currently, there are three schemes of assembly of finished products, depending on the location of units of storage of finished products (stacks), they are divided:

I. stacks are placed perpendicular to the map and parallel to the gross channels;

II. stacks are arranged parallel to the card and perpendicular to the gross channels;

III. enlarged stacks are placed on the border or beyond the deposit [13].

According to the first scheme, as a rule, mainly used bunker harvesting machines, which collect the finished product in the hopper and take it to the storage location on the strips. Thus peat is collected directly from the spread - pneumatic, or from the rolls - mechanical method of harvesting.

According to the II scheme, transshipment machines are used, which pour peat from pre-assembled rolls (distance between rolls of 20m) from one roll to another, until the formation of a stack.

The third scheme depends on the production technology and aims to create conditions for export of finished products to the consumer, usually by motor transport. At the same time use loading machines.

Extraction of peat suitable for fertilizer and fuel within the "Maidan" section of the "Starnik" field, located near the village. Maidan in Dubinsky district of Rivne region on the area of 606,85 ha. occurs according to the scheme And with the use of harvesters with mechanical principle of collection. These products meet the requirements for raw materials for the production of semi-briquettes, which have been manufactured by SE "Rivnetorf" since 1969 at the Smiz Peat Factory.

Peat is a valuable organic substance that has long been used in agriculture for soil enrichment. In modern technologies of cultivation of various plants peat is widely used as a substitute for soil. Indispensable is peat in container technology for growing plants, in greenhouses, for industrial production of mushrooms.

The record company LLC has been operating in the peat market since 2004. Within a short period of time, the company studied and implemented a quality system of peat extraction, started packing peat in piles, developed a system of supplying peat to buyers from Ukraine, began active export activities of its own products. Today, the company "Record" offers peat horse milling in piles and plastic bags.

High-quality peat is extracted from the licensed sections of the morochno-1 and morochno-2 peat fields located in the picturesque woodland of the Dubrovytsya district, Rivne region. This deposit is characterized by stable peat quality indicators. Peat is extracted under Scheme I with the use of pneumatic harvesting harvesters.

The technology of extraction of milling peat with the use of bunker harvesting machines with pneumatic principle of collection consists of milling a layer of peat, one stirring of the obtained crumb to accelerate its drying on the surface of the map, collecting peat from the spread by air flow nozzles of the harvesting machine. The collected peat is collected in the field stacks. The stacks are placed at the ends of the cards (scale strips), as with the technology of milling peat extraction with the use of bunker harvesters with mechanical principle of collection.

The pneumatic method of collecting peat from the surface of the drying fields is carried out by suction of the milling crumb from the spreader through the nozzle into the pipeline and transporting the peat-air mixture through pipelines to the cyclone, where the peat particles are separated from the mixture and deposited in the hopper of the harvesting machine.

The technological process is carried out by a set of technological equipment, which includes a milling drum, a mixing machine, a harvesting and stacking machine. The planned cycle duration in the production of milling peat for fuel in the pneumatic method of harvesting is 1 day. The number of cycles per season is 54-56.

The site on which the flowchart operations are carried out is similar to the site with the use of hopper machines with mechanical collection principle. Peat is stored in two stacks, located at opposite ends of the cards [14].

On the basis of SE "Rivnetorf" of the "Starnik" peat field, a study of peat extraction was carried out with the use of pneumatic harvesting principle.

At present, in this peat extraction of milling peat is carried out by means of bunker harvesting machines with mechanical principle of collecting, which collect the finished products in the bunker and take it to the place of storage on the scale strips. The peat is collected directly from the rolls. The main disadvantages of this technology are the presence of the operation of rolling, cycle of work 2 days. The length of the dimensions of the site implies the size of the length of the roll so that the volume of peat in the roll is equal to the volume of the hopper of the harvesting machine. Otherwise, it will be irrational to use the performance of the harvesting machine:

- if the volume of peat in the roll is greater than the volume of the machine, the machine will not be loaded;
- If the volume of peat in the roll is greater than the volume of the hopper of the harvester, then some of the peat will remain in the roll and will interfere with the technological operations in the successor cycles.

Changing the length of the roll requires changing the width of the strip from which the peat feeds into the roll, which in turn leads to a change in the design of the roll.

On the basis of the conducted researches it is established that one of the promising ways to reduce the cycle of work up to 1 day (to increase the volumes of peat extraction) and also to exclude the operation of rolling (to reduce the staff, to reduce the costs of fuel and lubricants) is to replace the existing technology with the technology of extraction peat using bunker harvesters with pneumatic picking principle. Improving the efficiency of this method, which is an important scientific and practical task, can be ensured by the use of modern pneumatic harvesting machines.

In our opinion, the most effective for the enterprise Rivnetorf is the replacement of the MTF-44 harvester with a JIK-40DF pneumatic harvester (Fig. 3).


Fig. 3. JIK-40DF Harvester [17]

The pneumatic hopper is an efficient productive machine, especially when weather conditions are difficult. Pneumatic harvesters can be used at enterprises with different volumes of milling peat production [14-17].

To compare the technology of milling peat extraction with the use of bunker harvesting machines with mechanical principle of collection and technology of milling peat extraction with the use of harvesting machines with pneumatic principle of collection, the calculation of the main production and technical indicators.

The calculation of the main production and technical indicators for the technology of peat extraction with the use of pneumatic harvesting machines [15, 16].

1. The duration of the peat extraction season is 129 days (May 10 - September 15).

2. The duration of the cycle. The planned duration of the cycle in the production of milling peat for fuel in the pneumatic method of harvesting is 1 day.

3. Number of cycles per season. During the one-day technological cycle, their number at the Starniks peat field at the Maidan section will be 54.

4. When using pneumatic harvesting machines, the milling depth is 9 mm = 0.009 m.

5. The operating humidity of the layer milled by year for the lowland type is: first and second year - 78%, third and subsequent - 75%, conditional humidity - 52%.

6. Coefficient of collection. For the third year of operation, the collection rate at a rate of 29% is 0.65. For the second year of operation, the collection factor is reduced by 0.05, for the first year, we reduce by 0.1.

7. Calculation of cycle and seasonal fees by years of operation.

1. Theoretical cycle fees for the first, second, third and subsequent years

$$q_{cT} = \frac{10^4 h_f \gamma_e (100 - \omega_e)}{100 - \omega_y} \qquad (1)$$

$$q_{cTz} = \frac{10^4 \cdot 0{,}009 \cdot 0{,}632 \cdot (100 - 75)}{100 - 52} = 29{,}63 t / ha\,;$$

$$q_{cT2} = \frac{10^4 \cdot 0{,}009 \cdot 0{,}692 \cdot (100 - 78)}{100 - 52} = 28{,}55 t / ha\,;$$

$$q_{cT1} = \frac{10^4 \cdot 0{,}009 \cdot 0{,}692 \cdot (100 - 78)}{100 - 52} = 28{,}55 t / ha\,\cdot$$

2. Actual cycle fees for the first, second, third and subsequent years

$$q_{cf} = q_{cT} \cdot \alpha_з \qquad (2)$$

$$q_{cf3} = 29.63 \cdot 0{,}65 = 19{,}26 t / ha\,;$$

$$q_{cf2} = 28{,}55 \cdot 0{,}6 = 17{,}13 t / ha\,;$$

$$q_{cf1} = 28{,}55 \cdot 0{,}55 = 15{,}7 t / ha\,.$$

3. Theoretical seasonal fees for the first, second, third and subsequent years

$$q_{CT} = q_{cT} \cdot n_{ch} \qquad (3)$$

$$q_{CT3} = 29{,}63 \cdot 54 = 1600{,}02 t / ha\,;$$

$$q_{CT2} = 28.55 \cdot 54 = 1541{,}7 t / ha\,;$$

$$q_{CT2} = 28.55 \cdot 54 = 1541{,}7 t / ha$$

4. Actual seasonal fees for the first, second, third and subsequent years

$$q_{Cf} = q_{cf} \cdot n_{ch}\,, \qquad (4)$$

$$q_{Cf3} = 19{,}26 \cdot 54 = 1040{,}04 t / ha\,;$$

$$q_{Cf2} = 17{,}13 \cdot 54 = 925{,}02 t / ha\,;$$

$$q_{Cf1} = 15{,}7 \cdot 54 = 847{,}8 t / ha\,.$$

Similar calculations were made for current technology. Based on the obtained formula, the graphs of actual seasonal fees for the first, second and third years using current equipment and proposed (Fig. 4) [16].

The graph shows that due to the increase in cycles for the season, the actual amount of seasonal peat collection increases, almost 2 (1.88) times. This means that it is quite realistic to increase the sales of milling peat, thereby increasing the income of the enterprise.

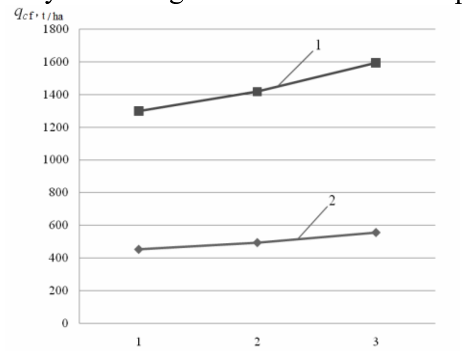


Fig. 4. Schedule of actual seasonal fees for the first, second and third years: 1 - when collecting peat with a pneumatic machine; 2 - when collecting peat with a hopper machine with mechanical principle of collection [17]

8. Seasonal program of the enterprise

$$P_{schoc}' = \frac{\beta \cdot P_p}{N_{cep}}, \tag{5}$$

$$P_p = 19013{,}25 \cdot \frac{0{,}973 \cdot (100 - 86{,}6)}{100 - 52} = 5164550\ t.$$

$$P_{schoc}' = \frac{0{,}85 \cdot 5164{,}55}{20} = 219490\ t.$$

9. The thickness of the layer triggered during the season is determined by the formula (6)

$$H = \frac{q_{SFZ} \cdot (100 - w_y) \cdot K_{BPP} \cdot K_{PGP}}{100 \cdot \gamma_{pp}(100 - w_{pp})(100 - P)}, \tag{6}$$

where q_{SFZ} - seasonal collection for the third and subsequent years, q_{SFZ} = 1040,04 t/ha

$$H = \frac{1040{,}01 \cdot (100 - 52) \cdot 0{,}85 \cdot 0{,}95}{100 \cdot 0{,}973 \cdot (100 - 86{,}6) \cdot (100 - 0)} = 0{,}3m \cdot$$

Analyzing the formula, we conclude that 0.3 m of peat layer can be worked during the season at the peat enterprise "Starniki", "Maidan" section, when replacing the current technological scheme with the technology of milling peat extraction with the use of bunker harvesters with pneumatic collecting principle.

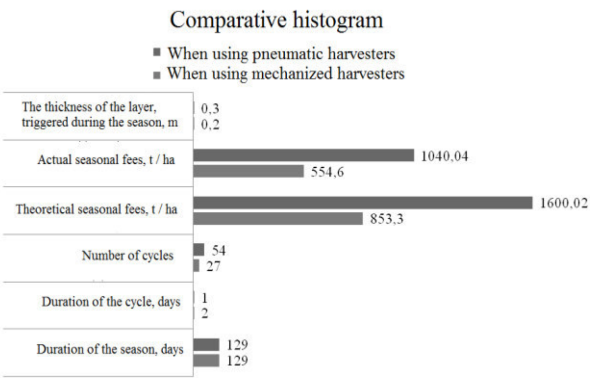
The results of calculations are summarized in Table. 3, for a visual comparison of production and technical indicators.

Table 3

Comparative characteristics of production and technical indicators [16]

№	Name of production and technical indicators	Units of measurement .	When mechanizing harvesters	Pneumatic drive of harvesting machines
1	Duration of the season	days	129	129
2	The duration of the cycle	days	2	1
3	Number of cycles	pieces	27	54
4	Theoretical seasonal fees	t/ha	853,3	1600,02
5	Actual seasonal fees	t/ha	554,6	1040,04
6	The thickness of the layer triggered during the season	m	0,2	0,3

According to the results of Table 3, a comparative histogram (Fig. 5) of production and technical indicators was used when using pneumatic harvesting machines and using mechanical harvesting machines [16].



According to the results of Table 3, a comparative histogram (Fig. 5) of production and technical indicators was used when using pneumatic harvesting machines and using mechanical harvesting machines [16].

Having analyzed this histogram, it is established that the use of bunker harvesting machines with pneumatic principle of collecting on the “Maidan” section of the “Starnik” peat bog is expedient and promising. The proposed technology is more efficient. After all, when using bunker harvesters with mechanical principle of collection, the production and technical indicators are worse than when the technology of milling peat extraction with the use of bunker harvesters with pneumatic principle of collection. This means that it is quite realistic to increase the extraction and sale of milling peat by 1.8 times, thereby increasing the enterprise's income.

Having analyzed this histogram, it is established that the use of bunker harvesting machines with the pneumatic principle of collecting on the “Maidan” section of the “Old Man” peat bog is expedient and promising. The proposed technology is more efficient. After all, when using bunker harvesters with the mechanical principle of collection, the production and technical indicators are worse than when the technology of milling peat extraction with the use of bunker harvesters with the pneumatic principle of collection. This means that it is quite realistic to increase the extraction and sale of milling peat by 1.8 times, thus increasing the enterprise's income.

Fig. 6. Machine for vacuum peat cleaning

It is the Canadian technique that allows to preserve the intact structure (fibrousness) of peat, thus retaining its high moisture retention capacity.

The record company LLC has been operating in the peat market since 2004. Within a short period of time, the company studied and implemented a quality system of peat extraction, started packing peat in piles, developed a system of supplying peat to buyers from Ukraine, began active export activities of its own products. Today, the company "Record" offers peat horse milling in piles and plastic bags.

Most of the peat is exported to Europe and Asia for further processing into the final product - peat mixes for growing a variety of plants. In 2010, Record launched its own substrate production line. This equipment allows every consumer in Ukraine to obtain a quality product for growing plants directly from the manufacturer. The stability of the raw material base, the experience of foreign partners, the extensive capabilities of the automatic production line of peat substrates together make it possible to present a series of European quality substrates at attractive prices.

Conclusions

Given the above, it can be noted that the rational extraction of milling peat depends not only on the reserves of deposits, but also on the technological equipment used in the technological scheme for the development of peat deposits, which in turn affects its economic feasibility of use in the peat enterprise as a whole, which directly affects the profit of the enterprise.

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CORRELATION OF BLASTING PERFORMANCE WITH LOADING AND CRUSHING TIME TO MINIMIZE ENERGY CONSUMPTION

Alaba, O. C.
Federal University of Technology, Akure,
Department of Mining Engineering, Nigeria
Senior Lecturer, Doctor of Mine Health, Safety and Environment,
Email: ocalaba@futa.edu.ng;
Phone No: +234 806 277 7774

Abdulraman, S.O.
Kwara State University, Malete
Department of Materials Science and Engineering, Nigeria
Senior Lecturer, Doctor of Surface Mining,
Email: sikiru.abdulraman@kwasu.edu.ng;
African University of Science and Technology, Abuja,
Department of Materials Science and Engineering, Nigeria
Visiting Faculty, Phone No: +234 803 397 9980

Abstract

Blasting performance with loading and crushing time were correlated in order to minimize energy consumption in quarry operation. A measure scaled object was placed within the muck pile of blasted rock fragments. Digital camera was used to take the photograph of the scaled object and the blasted rock fragments before the commencement of loading and crushing operations. The data collected were subjected to fragmentation analysis using split desktop App while the correlation analyses were carried with Microsoft Excel App. The fragmentation and distribution analysis showed that the fine cut off sizes are 8.75 cm, 9.55 cm, 7.58 cm, 8.58 cm and 11.75 cm while their mean size distribution are 35.99 cm, 10.25 cm, 24.86 cm, 16.78 cm and 16.10 cm respectively. The R squared values (R^2) obtained during the correlation of digging time with size of fragmentation passes (P50), mean deviation and bucket passes were given as 0.94, 0.39 and 0.63 with p-values of 0.00, 0.34 and 0.82 respectively. Consequently, the R^2 values during the correlation of crushing time with P50, mean deviation and range were specified as 0.64, 0.62 and 0.27 with p-values of 0.00, 0.11 and 0.07 respectively. The study therefore established that only the regression equation for P50 is statistically significant while statistically insignificant for mean deviation, range and number of bucket passes.

1. Introduction

In recent times, the campaign for energy efficiency in mining industry is yielding good results. Energy consumption in mining contributes to operational costs in all stages of ore recovery process: blasting, excavation, materials transfer and haulage, comminution, ventilation and dewatering [1]. In order to optimize energy in mining sector, Energy Mass Balance (EMB) has been adopted and incorporated into mining industries activities. The EMB systematically collects and analyses data on energy use and investigates where losses occur [2]. This suggests that the EMB can only be used as an energy saving strategy in mining industry but it cannot predict the outcome of any system of operations and cannot really access blast results to show energy losses during blasting.

Also, some optimization models have emerged in the mining industry that are operation specific such as loading, hauling and crushing [3,4,5]. For instance, Leighton Contractors Pty Ltd developed a series of mass and energy equations to improve the energy saving initiatives in haulage operations during mine design [6]. Fortescue Metals Group identified and quantified the energy costs associated with stopping haul trucks [7] while Downer EDI Mining developed performance indicators that use an 'equivalent flat haul' calculation to account for elevation changes on a specific mine route [8]. These models have achieved numerous results but they can only be used to evaluate the energy optimizations for haulage operations.

As mentioned earlier the major limitation that unites all the models discussed above is that they actually correct energy wastage in operations but they do not prevent the energy wastage. It does not agree with the saying that "prevention is better than cure. Fragmentation control is the best way to prevent energy wastage in the entire mining operations and it can be done through optimization of blasting performance [9]. The reason for this assertion is based on the fact that the quality of the fragmentation after blasting affects both loading, haulage and crushing. The overall cost effectiveness of the production operations is compatible with optimization of drilling and blasting parameters [10]. Kuz-Ram fragmentation model is the closest mining based model that predicts fragmentation performance [11]. It has helped blasting engineers round the world in predicting the outcome of a blast and energy used up in blasting. The basic limitation of the model is that it cannot predict the energy consumption in subsequent mining operations like: loading, haulage and crushing.

Since, most of the previous studies focused much on the correction of energy wastage in operations with little attention on the prevention of energy wastage, this study therefore provides opportunity to advance our knowledge on energy correction and prevention of energy wastage in loading and crushing operations through optimization of blasting performance

1.1 Overview of Energy Consumption in Mining and other Industrial Sectors

Energy consumption is one of the key determinants of operational cost in mining and other industrial sectors [12]. Recent investigation showed increase in daily demand of energy in mining and other industrial sectors. In 2017, the world total energy supply was 162,494 TWh of which only 113,009 TWh was finally consumed [13]. Out of this, mining sector consumed approximately 11% of the world total energy [14]. In 2012, about 2-3 % of the total energy supplied in USA was consumed by mining sector [15] while 8 % of the total energy supplied in South Africa was consumed by mining sector in 2010 [16]. The corresponding number in Canada in 2009 was 12 % [17]. The world's largest individual energy consumer is China, which also produces 40 % of the world's minerals [18].

In 2025 about 36.2 % of energy are projected to be consumed by new mining project [1]. It was also projected that about 41.1 TWh of energy would be used in 2025 by copper mining industry in Canada which is an increase of 95.5 % recorded in 2013 [19]. In all stages of mining operation, mineral processing consumes most energy worldwide and it accounts for 4-5 % of the total electrical energy produced globally [1,20]. In mineral processing operation, loading and crushing are the principal energy consumers with 40% of energy consumed in mineral processing operations and 4 % of the world's total electrical energy [21-24].

2. Material and Methods

2.1. Description of the Study Area

Francisca Muinat (FM) Quarry is the case study area and is situated within the longitudes 50 00'E and 50 17'E and latitudes 70 10'N and 70 20'N about 4 km to Akure on Aaye-Ijare road as shown in

Figure 1. The outcrop is made up of migmatite intruded by three petrological varieties of older granites under the Basement complex of South-western Nigeria. Francisca Muinat Quarry has a joint venture partnership with the Chinese. It is a commercial aggregate quarry that produces different sizes of granite aggregate for construction purposes

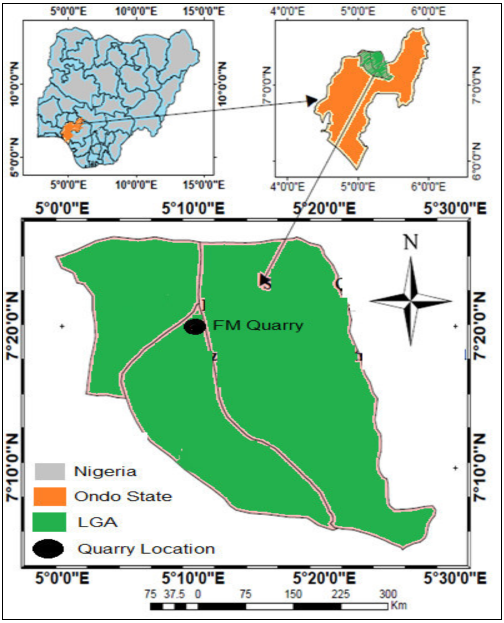

Fig. 1. Location of study area

2.2. Data Collection Procedures

(a) Loading and Haulage Procedures

(*i*) A scaled object was placed within the blasted rock fragments and digital images of the fragmented rocks with the scaled object were taken before the commencement of loading and haulage operation.

(*ii*) The time taken for the loader to dig and scoop fragmented rocks from the muck pile to the loader for each bucket was recorded.

(*iii*) The swing time to and fro (time taken in digging and loading the dump truck and then coming back to muck pile again) was recorded.

(*iv*) The number of buckets to fill the dump truck was recorded by recording the digging time for each bucket.

(*v*) The travel time of the dump truck with the loading fragments from pit to crusher hopper was recorded

(*vi*) The travel time for the dump truck to travel back to the pit empty was recorded

(*vii*) The above procedures were repeated for twelve times in order to obtain adequate data for the analysis

(b) Crushing Procedures

(*i*) A scaled object was placed within the blasted rock fragments in the crusher hopper and digital images of the run off mine fed into the jaw plates was taken before the commencement of crushing operation

(*ii*) The time to crush each of the fragmented rocks in the crusher was taken and recorded

(*iii*) The above procedures were repeated for twelve times in order to obtain adequate data for the analysis

2.3. Data Analysis

The variables P50, range, mean deviation and p-value were used to carry out correlation analysis using split desktop App and Microsoft excels application. P_{50} was used because its value can be obtained from Kuz - ram model to predict blasting performance while range and mean deviation were used to predict the size distribution of fragmented rocks. The image photographs obtained from field were uploaded on the split desktop App. The split desktop App was used to scale and delineate the images for size distributions analysis. This was used to determine the sizes at which 50 % (P_{50}) of the particular fragment passes. The mean deviation and the range were calculated while their R squared values and the P values for the models derived were observed. The variables P_{50}, range, mean deviation for all the images were plotted against time taken to dig, load and crush the fragments using Microsoft excel App. The linear regression lines obtained from Microsoft excel App and their respective R squared values with p-values were used for the correlation analysis with blasting performance. The R squared values show how well the points fit the regression lines. The R squared value indicates how well the points plotted on the graph agrees with the regression line, which informs the decision of the analyst on whether to rely on the equation or not. R squared values above 50 percent is quite reliable and the values under 50 percent is not reliable. The p-values show the level of statistical significance of the model.

3. Result and Discussion

3.1. Fragmentation and Distribution Curves Analysis

The results of the fragmentation and size distribution analyses using split desktop App were presented in Figures 2-7. The fainted red lines in the figures indicate fine particles while the bold red lines indicate fragmented rocks. The fine cut off indicates the size below the fragment that considered as fine particles. Figure 3 shows that the fine cut off of 8.75 cm with mean size of 35.99 cm while Figure 4 shows a fines cut off of 9.55 cm with mean size of 10.25 cm. Figures 5-7 show fine cut off sizes of 7.58 cm, 8.58 cm and 11.75 cm respectively while their mean sizes are 24.86 cm, 16.78 cm and 16.10 cm respectively

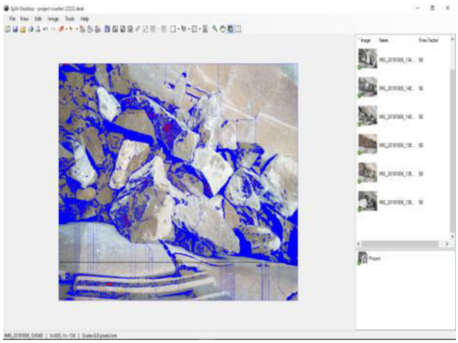

Fig. 2. Delineation picture using spilt desktop application

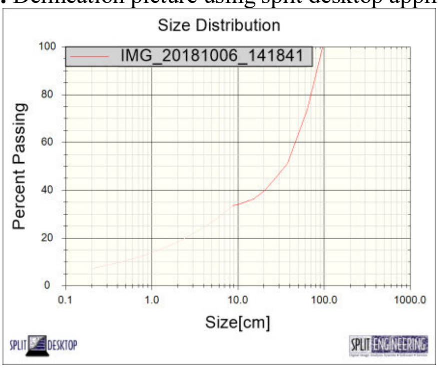

Fig. 3. Fragmentation analysis for image (1)

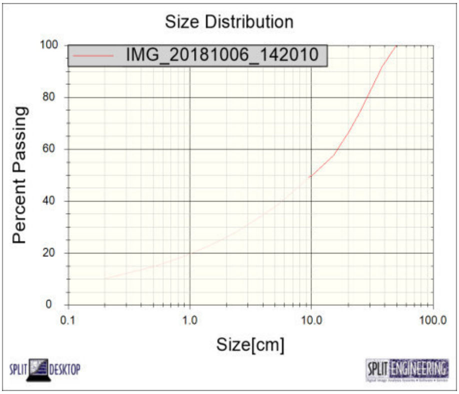

Fig. 4. Fragmentation analysis of image (2)

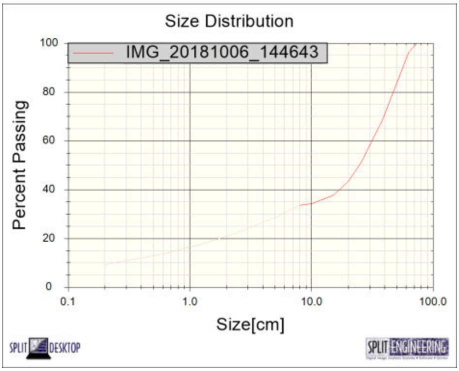

Fig. 5. Fragmentation analysis of image (3)

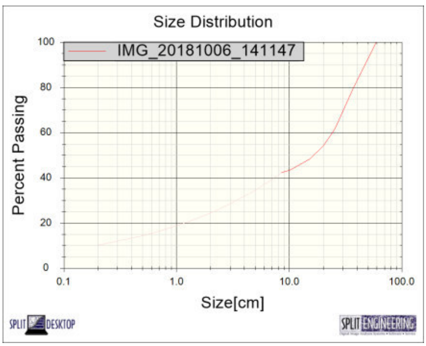

Fig. 6. Fragmentation analysis of image (4)

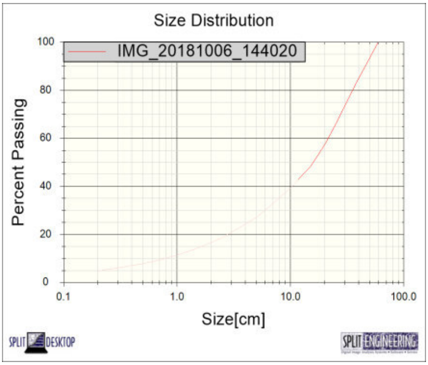

Fig. 7. Fragmentation analysis of image (5)

3.2. Correlation of loading time with fragmented size

The average time taken for digging during loading was correlated with the size at which P_{50} of the muck pile passes as presented in Figure 8. The regression equation of average digging time and P_{50} was given as y=0.0036x+0.1242 with R-square value (R^2) of 0.9403. This demonstrated that there is good correlation between the average digging time and sizes at which P50 of the particular muck pile passes as 94.0% of R squared value (R^2) of the two variables formed the regression line. The calculated p-value is given as p<0.00 which means the two variables are statistically significant.

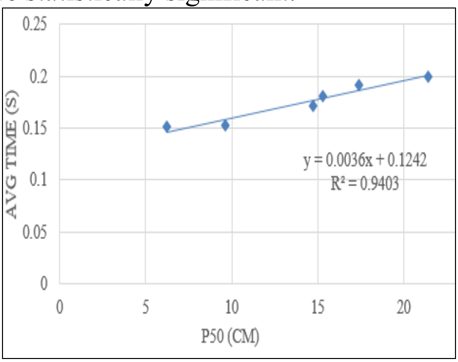

Fig. 8. Correlation between average digging time and P50

Figure 9 presents the correlation of average digging time with the mean deviation. The regression equation of average digging time and mean deviation of size distribution of the muck pile was given as

$y=0.0055x+0.106$ with $R^2=0.3932$. This substantiated that there is no good association between the digging time and size distribution of the muck pile as only 39.0% of the two variables aligned to the formation of regression line. At the same time, the calculated p-value is given as 0.34 which established that the two variables are statistically insignificant.

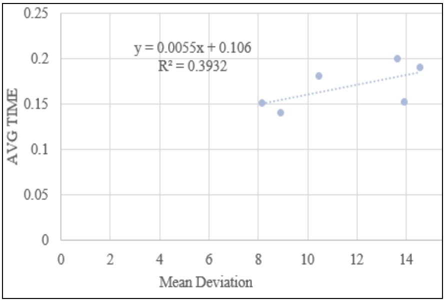

Fig. 9. Correlation between average digging time and mean deviation

The relationship between the average digging time and number of bucket passes during the loading of muck pile is presented in Figure 10. It was discovered that the regression equation between the two variables is given as $y = -0.0548x + 0.5982$ with $R^2 = 0.6364$. This affirmed that there is a good relationship between the digging time and the number of bucket passes during the loading operations as 63.0% of the two variables formulate the regression line. The estimated p-value was given as 0.82 which proved that the two variables is not statistically significant.

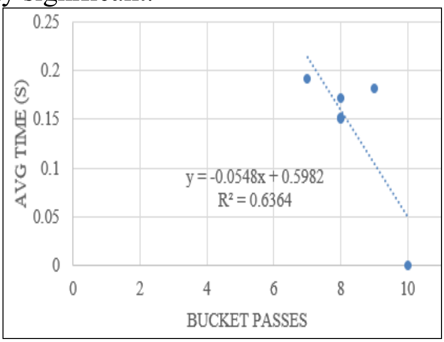

Fig. 10. Correlation between average digging time and number of bucket passes

3.3. Correlation of crushing time with fragmented size

The time taken to crush the run off mine in the crusher is plotted against P50 as shown in Figure 11. The regression equation of blasting performance from the figure was given as $y = 0.0276x + 0.3566$ with R^2 value of 0.6408. This means there is good relationship between the time taken to crush the run off mine in the crusher and sizes at which P50 of the particular fragment passes as 64.0% of R squared value (R^2) of the two variables aligned to form the regression line. The calculated p-value is given as $p<0.00$ which means the two variables are statistically significant.

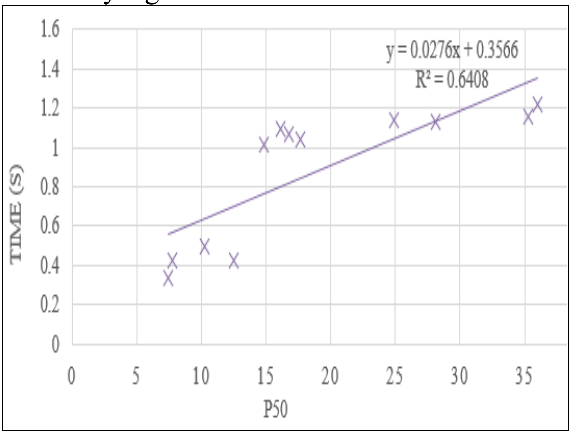

Fig. 11. Correlation between the time to crush and P_{50}

The time taken to crush each of the muck pile loaded in the crusher hopper was correlated with the mean deviation as shown in Figure 12. The regression equation of how much the various fragments deviates from the mean size (P_{50}) of the stock of materials was given as y=0.0547x+0.1345 with R^2=0.6212. This presents that there is good correlation between the time taken to crush each of the muck pile dumped in the main hopper and the fragmentations distributions of the muck pile dumped in the hopper as 62.0% of R^2 value of the two variables agreed to form the regression line. The p-value for the regression was given as 0.11 which is $p>0.05$. Therefore it can be deduced that the two variables are statistically insignificant.

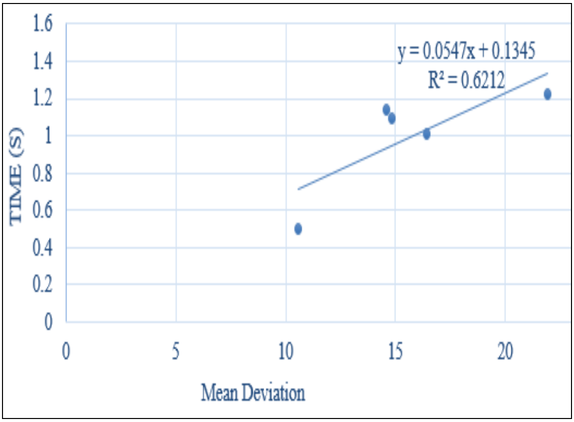

Fig. 12. Correlation between the time to crush and mean deviation

The relationship between the times taken to crush each trip of feed material and the range of sizes of the fragments were correlated as shown in Figure 13. The regression equation of range of sizes of the fragment was given as y=0.0103x+0.1763 with R^2=0.2772. This establishes that there is no correlation between the time taken to crush each trip of feed material and range of sizes of the fragments (difference between maximum and minimum sizes of the fragments) as 27.0% of R^2 value of the two variables formulate the regression line. Meanwhile, the calculated p-value is given as 0.07 which means the two variables are statistically insignificant

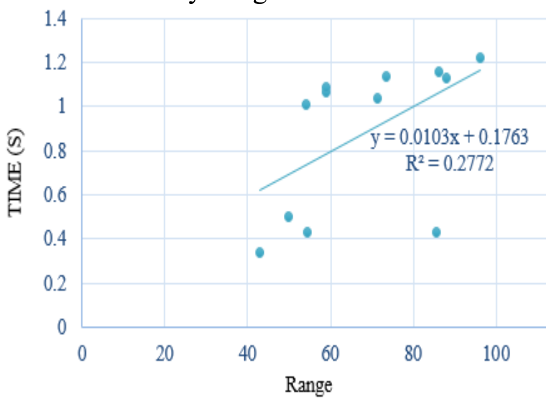

Fig. 13. Correlation between the time to crush and range

Conclusion

This study was set out to correlate blasting performance with loading and crushing time in order to minimize energy consumption in quarry industry. The study therefore concluded as follows:

(*i*) The fine cut off values of fragmentation and distribution sizes using split desktop App are given as: 8.75 cm, 9.55 cm, 7.58 cm, 8.58 cm and 11.75 cm while their mean size are 35.99 cm, 10.25 cm, 24.86 cm, 16.78 cm and 16.10 cm respectively.

The correlation of loading time with fragmented size established that the regression equation of average digging time and P_{50} had an *R*-square value (R^2) of 0.9403 with p-value of 0.00 which means the two variables are statistically significant. Meanwhile, the regression equation of average digging time and mean deviation of the muck pile had an R^2 value of 0.3932 with p-value of 0.34 which established that the two variables are statistically insignificant.

The regression equation between the average digging time and number of bucket passes during the loading of muck pile had an R^2 value of 0.6364 with p-value of 0.82 which proved that the two variables is not statistically significant.

(ii) The correlation of crushing time with fragmented size ascertained that the regression equation of blasting performance had an R^2 value of 0.6408 with p-value of 0.00 which means the two variables are statistically significant.

Consequently, the regression equation of how much the various fragments deviates from the mean size (P50) of the stock of materials had an R^2 value of 0.6212 with p-value of 0.11 which means the two variables are statistically insignificant. The regression equation of range of sizes of the fragment had an R^2 value of 0.2772 with p-value of 0.07 which means the two variables are statistically insignificant.

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RECLAMATION OF DESTRUCTED LANDS OWING TO ILLEGAL AMBER PRODUCTION IN NORTHERN REGIONS OF UKRAINE

Korniyenko V.Ya.
National University of Water and Environmental Engineering (NUWEE), Doctor of Engineering, Professor, Department of Mining and Mining, Ukraine

Chukharev S.M.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Associate Professor, Department of Mining and Mining, Ukraine

Zaiets V.V.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Associate Professor, Department of Mining and Mining, Ukraine

Vasylchuk O.Yu.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Associate Professor, Department of Mining and Mining, Ukraine

Abstract

This section analyses the state of disturbed lands as a result of the illegal extraction of amber in the Rivne region. The general characteristics of the territories and the negative consequences of illegal amber production are given. Were analysed and found possible future composition of vegetation of specific Dubrovytsky forestry area. The main stages and directions of reclamation of disturbed lands are considered. As a result of the analysis of literary sources and archival data, possible options for reclamation on the disturbed lands were suggested. It has been established that conducting a complete reclamation of sites damaged by illegal amber production is the only technically available method, given the nature and extent of the violation of these lands. It has also been found that the pilot-industrial extraction with 100% the extraction of the remaining amber is a prerequisite for preventing the repeated illegal intervention of the diggers on the reclaimed land. It is established that after the completion of the pilot-industrial extraction and technical reclamation it is necessary to

conduct additional ecological and economic assessment of the disturbed territories in order to further extract amber, forestry and use for agricultural production

Introduction

The problem of illegal extraction of amber in the forest lands of the Ukrainian Polissia (Volyn, Rivne and Zhytomyr regions) has existed for more than 15 years, but it has become a particularly urgent subject of public attention in recent years, primarily due to the efforts of the public and the media.

The illegal activities of the "diggers" cause serious economic losses to the state and society, accompanied by deterioration of the social climate, leads to the degradation of large areas of forest lands, the deterioration of the ecological situation. Since disturbed biotopes cannot be restored naturally, without human intervention, they require reclamation [1-3].

The state, represented by central and local authorities, both legislative and executive, is forced to make efforts to resolve the problem, but no significant changes have been made.

One of the areas of intervention by the state authorities and the public is the activity of practical implementation of reclamation of the affected areas. To this end, the Cabinet of Ministers approved a resolution of the Cabinet of Ministers of November 30, 2016 No. 1063, which approved the "Procedure for the implementation of a pilot project of reclamation of forestry lands damaged as a result of illegal amber production". According to this resolution, the Order of the State Forestry Agency of Ukraine No. 138 of 21.04.2017 established the "List of forest lands, within which there are parts that are damaged due to illegal amber production and need reclamation", which included 2046 disturbed areas with a total area of 30037.6 ha. Therefore, the issue of reclamation of disturbed lands due to illegal amber mining is undeniably topical.

The subject of the study is the assessment of the ecological status of forest lands disturbed by illegal amber production that require reclamation.

The object of research is the forest areas of Rivne region, which have been affected by illegal amber production.

The purpose of the research is to assess the overall ecological status of the disturbed lands and to establish acceptable areas for their reclamation.

These studies were conducted by studying, analysing and generalizing literary including archival data.

1. General characteristics of disturbed lands

The territory on which the disturbed forests are located is in the western part of the mixed forest zone, which is called the Ukrainian Polissia, of the Eastern European Plain within the Volyn Polissia region (Fig. 1).

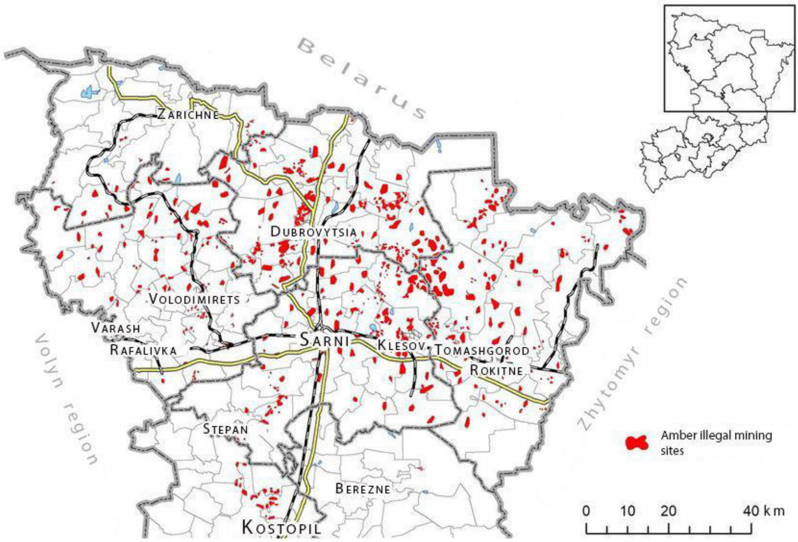

Fig. 1. Positioning of sites of illegal amber production in the territory of Rivne region

Ukrainian Polissia is a kind of physical-geographical province of the mixed forest zone of the Eastern European Plain. The surface is characterized by a slight degree of land dismemberment, a general slope to the north, considerable afforestation (pine and mixed deciduous forests), wetlands, the presence of a dense network of watercourses, with a clear division into watersheds and river valleys.

The climate is temperate continental with warm summers, mild winters and heavy rainfall. The maximum precipitation falls in June-July, the minimum in January-February. In rainy years, rainfall reaches 900 mm. The total number of days with precipitation is 160-190. The average annual relative humidity is 80%, the average annual temperature is 6.6-7°, the maximum average monthly temperature of air is 17-19° fixed in July, the minimum (-6°) - in January.

The territory is dominated by sod-podzolic, sandy and marsh soils. Wetlands occupy large areas, which is facilitated by physical and geographical conditions, the nature of the relief and the hydrodynamic features of the area.

In geostructural terms, the territory is located within the north-western outskirts of the Ukrainian Crystalline Massif and is characterized by the presence of two structural floors: the lower one, which is represented by crystalline rocks of the foundation, and the upper one, which consists of horizontally lying sedimentary formations. The crystalline basement complex formed by ancient metamorphic and magmatic rocks are in their age of Archean and Proterozoic.

Soil cover of disturbed lands is diverse in genesis, mechanical composition, water-physical properties and fertility.

The relative plainness of the territory of the region and the low flow of groundwater led to the formation of large areas of wetland soils In the few elevated areas formed sod-podzolic soils, and the more elevated dominated loose sands. Swamp soils cover 50% of the area here, sod-podzolic - 29%, sod - 6%, meadow - 5%, sand hills about 10%.

The investigated area covered with wood (birch, hornbeam, alder, pine) with a typical shrub and moss vegetation (Table. 1).

To visualize the dynamics of change of forest cover, the archive data of the change of terrestrial cover of the quarter № 59 of Dubrovitske Forestry are presented below (Table 2, Fig. 2). Also are provided forecast data, in the absence of legislative regulation of the issue of illegal amber production.

Table 1

Specific composition of the forest fund in the context of forestry in the region

Forestry	Pine, Ha	Oak, Ha	Birch, Ha	Alder, ha	Spruce,. Ha	Hornbeam, Ha
Bereznivske	34082,2	2379,7	6263	4569	85	226
Volodimiretske	46947,5	2479,7	11822,9	12724,4	382	203
Dubrovitske	27516,8	2205,1	8839,4	3746,1	147.2	134
Zarichnenske	20210,7	1349,4	5023,6	5752,4	235,4	97

Klesivske	34069,5	1191,2	7263,7	2607,7	125,2	14,8
Kostopilske	19328,8	3116,3	4266,6	4754,9	307,8	104,8
Rokitnovske	35760,4	736,8	6382,7	2138	93,7	57,3
Sarnenske	27461,5	1246	6030,1	4017,3	144.7	64,7

Table 2

Dynamics of change of forest cover of quarter №59 of Dubrovitske forestry as a % of the total area

Type of territory / year	2009	2015	2017	2021
Amber extraction area	0,1	2,6	2,3	3,5
The surface is small and / or not covered with vegetation	3,1	13	12,9	14
Surface covered with undersized vegetation	0,3	2	2	4,5
Deciduous forest	65,5	61,1	58,6	58,2
Coniferous forest	31	21,2	24,1	19,7

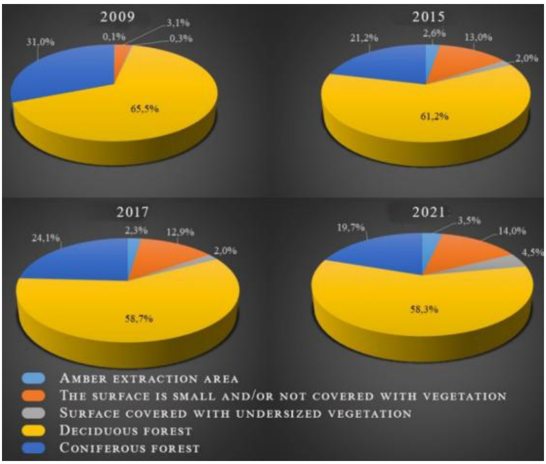

Fig. 2. Dynamics of change of forest cover of quarter №59 of Dubrovitsky forestry

2. The negative environmental effects of illegal amber production

It is clear that the illegal extraction of amber by common methods (trenching, underground hydraulic leaching, erosion in pits) has a negative impact on the ecological state of territories development.

An illegal production is uncontrolled, and not only in terms of raw materials extraction, but also in terms of extraction - violation of natural territories. Nature is a complex system where all the components are inextricably linked with one another. Therefore, disturbance of the geological environment inevitably causes negative changes in adjacent environments. The most negative impact is in the biotic component of nature [4-6].

When amber extraction method is trenching it completely destroys grass and shrub layer of the forest mechanically and damages root system of trees, and frequent trees often are cut and uprooted. Due to lack of soil and due to damage, the root system is not able to hold the trunk upright and the trees tilt or fall at all under their own weight (Fig. 3).

Fig. 3. Vegetation in the Rivne region after illegal extraction of amber

In this case, nearby plants may be damaged, the undergrowth dies. Over time, most trees die. Such territories are characterized by the almost complete absence of primary soil cover, but the huge number of pit holes reduces the area for seed development and, consequently, for the development of young forest.

This destroys the modern forest and does not create the conditions for its restoration.

A similar, but at the same time, another situation is observed in the extraction of amber by the hydropump method (Fig. 4). There are two types of negative consequences.

The first is the creation of funnel-like cavities in the soil, its overlapping with the covered material; other - breach hydrogeological regime of the area from the inflow of large amounts of water.

Mechanical impact on soil is its subsidence and burial under a layer of sandy-clay material. Excessive additional wetting with process water leads to a long rise in the groundwater level [7-9].

Fig. 4. "Moon relief" after the hydropump illegal extraction method of amber

As a result, the root system of the trees is incapable of keeping them in equilibrium in sparse sandy soil. Also raised groundwater level prevents the penetration of oxygen to the roots and they die from waterlogging. When the number of dead roots reaches the critical limit, the plant dies completely. The difference with the hydropumping method is that the soil and rock do not move into the dumps, but are evenly distributed throughout the territory by water flows, creating conditions for further self-healing of the forest.

On the basis of the analysis of the condition of the affected areas, it was found that the species similarity of the floristic composition of the majority of forest groups after amber extraction in the studied areas is very low.

It should be noted that deep canals were dug for pumping amber by pumps, which drain forests over large areas, which leads to changes in the hydrological regime of the area and weakens the ecological stability of forest ecosystems [10-12].

It was also found that in most cases the number of species in disturbed areas was lower than the original phytocenoses. This is indicative of the total destruction of phyto-diversity due to the illegal extraction of amber.

3. Known stages and directions of reclamation

Land reclamation is the implementation of various works that aim not only at the partial transformation of natural territorial complexes disturbed by industry, but also at creating in their place even more productive and rationally organized elements of cultural anthropogenic landscapes. That is, it involves the optimization of man-made landscapes, the improvement of environmental conditions.

The following terms are distinguished in a concise dictionary on land reclamation (1980): temporary land reclamation, permanent land reclamation and reclamation of landscapes.

Temporary reclamation is carried out on lands where their use is planned to change in the future. This reclamation is usually reduced to planting and fixing surface erosion, and compliance with hygiene standards.

Permanent reclamation is carried out on the land where the change does not provide the previous use of the land.

Reclamation of landscapes is a land reclamation, which is not limited to local measures to "tidy up" individual disturbed areas, but involves the complex transformation of disturbed land in the general system of measures to optimize man-made landscapes.

At the present stage of society development, many domestic and foreign scientists consider the reclamation of disturbed lands as a complex problem of restoring the productivity and reconstruction of landscapes disturbed by industry, created on the site of "industrial deserts" of new cultural landscapes.

The state standard "Nature Conservation. Land reclamation. Terms and definitions" treats reclamation as a complex of works aimed at restoring the productivity and national economic value of land, as well as improving the environmental conditions.

All the processes of reclamation of disturbed land are divided into two main stages: mining technical and biological. However, in practical terms considered more appropriate definition of three phases: preparatory, mining technical and biological.

The preparatory or design phase includes the examination and typing of disturbed and infringed lands; study the properties of cover soils and classify them as suitable for biological reclamation; determination of directions and methods of reclamation; drawing up feasibility studies (FS) and technical work on reclamation projects.

Mining technical, or engineering, a step, also called technical or mining technical reclamation, involves the completion of work on the preparation of lands released after mining operations for further targeted use in the economy. The following works are performed at this stage:

- selective removal, storage and preservation of cover soils suitable for biological reclamation, including fertile soil;
- selective formation of cover soils dumps;
- if necessary planning and coverage of the planned surface layer of fertile soil or potentially fertile soil;
- backfilling and planning of deformed surfaces (dips, karst funnels, etc.);
- installation of access roads;
- ameliorative and anti-erosion measures.

In the reclamation of disturbed forest lands, the technical stage includes only works related to the conservation and reuse of fertile soil (if available), surface planning, and (if necessary) chemical reclamation and ordering of drainage networks.

Biological reclamation is a set of measures to create favourable water-air and nutrient regimes for crops and forestry. The complex of measures of biological land reclamation is determined by the physicochemical properties of the underlying rocks and the applied fertile soil layer or potentially fertile rock. This complex covers the introduction of crop rotations, saturated with crops for fertilizer, the introduction of high standards of organic and mineral fertilizers, mulching etc.

The areas of reclamation determine the end use of the disturbed lands after the relevant mining, engineering, hydrotechnical and other measures. they are selected on the basis of a comprehensive account of the following factors:
- natural conditions of the deposit area (climate, soil types, geological structure, vegetation, wildlife, etc.);
- condition of disturbed lands up to the moment of reclamation (nature of technogenic relief, degree of natural overgrowth, etc.);
- mineralogical composition, water-physical and physical-chemical properties of rocks;
- agrochemical properties (nutrient content, acidity, presence of toxic substances, etc.) of rocks and their classification by suitability for biological reclamation;
- geotechnical and hydrological conditions;
- economic, socio-economic, environmental and sanitary conditions;

- service life of reclamation lands (possibility of repeated violations and their periodicity);
- technology and mechanization of mining and construction works.

The most common are the following areas of land reclamation:
- agricultural;
- forestry;
- water management;
- recreational;
- sanitary and hygienic;
- constructional.

4. Possible reclamation on disturbed lands

Reclamation works consist in the implementation of a set of organizational, technical and biotechnological measures aimed at restoring soil cover, improving the condition and productivity of disturbed forest lands, creating new plantations (biological reclamation).

Reclamation of disturbed forestry land has a direct impact on the following environmental components:
- soil cover;
- vegetation cover;
- soil (first from the surface) aquifer, including conditions of infiltration of precipitation and snowmelt;
- terrain, including surface planar runoff and snowmelt conditions.

Given that the existing parameters of all the above components of the environment are significantly impaired, they are under the influence of long-term degradation, or even close to those that have completely lost their natural potential (topsoil, groundwater, tree stand), and their reclamation restoration, it is safe to say that in general the planned list of works does not cause any negative environmental impacts.

Preparatory work should include the removal of the remains of disturbed stands, other perennial vegetation affected by illegal amber mining activities, tree stumps, etc.

Probably some of the wood (branches, shrubs, debris, etc.) can be burned at the site of work, which will result from atmospheric air pollution.

During the preparatory phase, surveys and typing of disturbed lands are carried out, the properties of cover soils are examined for suitability for biological reclamation; determine the directions and methods of reclamation; make feasibility studies and technical working projects on reclamation.

Preparatory work also includes the removal and storage of the upper fertile (humus) layer of soil together with forest floor in places where they are not disturbed (not swept away during the operation of the motor-pump). The preserved soil is further used in planting.

Restrictions on the use of forest land during preparatory work are conditioned by the need to comply with the procedure of felling in accordance with the provisions of the environmental legislation of Ukraine in the field of forestry.

The work directly from reclamation can be conditionally divided into three stages:

1. Geological exploration including pilot-industry extraction (PIE) with the extraction of discovered amber.
2. Mining technical reclamation.
3. Biological reclamation.

PIE and technical reclamation are carried out in a single set of works: the final stage of the PIE at each site is its technical reclamation, which includes planning, reclamation activities, waste disposal and more.

At certain stages of the work, a bulldozer is used to move the soil to the dump and to plan the surface. In some cases wheeled freight transport (dump truck) may be used.

Temporary dumps are arranged to accumulate cover and waste soil sand; for water reserve - sump.

All the heavy equipment that is planned to be used meets the current European requirements for ecology for off-road vehicles (emissions, noise, energy consumption, etc.).

During the technical phase, surface planning and filling of a layer of fertile soils are carried out. The purpose of phase - creating satisfactory conditions for the growth of trees and shrubs.

Disturbed lands are brought to a suitable state during technological works, or no later than one year after the end of the PIE.

The technical stage of reclamation after completion of the PIE will include the following set of works:

- selective formation of heaps;
- surface planning;

- chemical reclamation;
- covering the planned surface with a layer of fertile soil or potentially fertile soil;
- engineering equipment of the territory (roads, exits).

After completion of the PIE stage and technical reclamation, special agro-ecological and agrochemical studies should be conducted to determine soil contamination, fertility (humus content), chemical and organic properties, etc. Based on the results of these studies, it will be necessary to prepare recommendations for a set of agro-chemical and agro-cultural activities and to establish a dendroplan of plantations.

Prior to the beginning of biological reclamation, the area of 2-3 years must be maintained or sown with annual crops to enrich the soil with nutrients (legumes, alfalfa, etc.). The fertility recovery method is chosen after the soil condition has been studied.

During the biological phase, measures are being taken to create highly productive, sustainable forest biogeocenoses, which are an important and beneficial factor for environmental impact.

Biological reclamation covers a complex of agro-technical and phytomeliorative measures to increase fertility of disturbed lands.

Forest reclamation involves the cultivation of a particular set of forest crops on the reclaimed lands.

Biological reclamation includes a set of measures to create water-air and nutrient regimes for forest species. Biological reclamation involves:

- introduction of sidereal rotations;
- introduction of high standards of organic and mineral fertilizers;
- mulching;
- deep loosening.

Reclamation measures on the topsoil should be maximally effective in order to reduce the recoupment of reclamation costs.

One of the most common areas of land reclamation is forest reclamation. Conducting of forest reclamation on disturbed lands allows to create forest plantations of various types and purpose, which helps to improve the natural environment, improve the sanitary and hygienic conditions of human life, while increasing the area of forest land. The process of natural restoration of ecosystems on disturbed lands begins with almost complete absence of living components. Formed empty ecological niches are first populated by microorganisms, fungi and algae, and then they are inhabited by higher plants, which passes spontaneously and without human participation. It should be noted that this natural process of ecosystem formation is very long, so there is a need for artificial restoration of vegetation.

Staged forest reclamation is usually carried out in most countries, and direct reclamation is performed only on the richest soils. In the step-by-step forest reclamation, soil-improving plants are first planted and then replaced with more valuable forest tree species. Reforestation activities depend on many interrelated factors.

Factors that determine and influence the choice of forest restoration measures in disturbed areas:

- physical, geographical and climatic features of the area;
- physicochemical properties of the soil cover and underlying soils;
- area of the violated territory;
- forestry use of the territory until the moment of disturbance;
- promising targeted use of the restored territory.
- the amount of capital expenditures on timber and reclamation measures;
- availability of material and technical resources;
- social effect of phytomeliorative and reclamation works.

Restoration of forest vegetation on disturbed lands depends on the suitability of soils to create vegetation, which directly affects the technological process itself. Restoration of forest vegetation on disturbed lands depends to a large extent on the suitability of soils and potentially fertile species to cover vegetation. The soil mixture formed after the illegal extraction of amber will differ markedly from the previous composition of soils in its physicochemical composition. In this case, an important factor is conducting appropriate research. In this case, the main criteria for fertility should include: granulometric composition, water-physical features, indicators of salinity and acidity, humus and nutrition. It is also important to provide for the conditions of each ecotope a set of suitable species of tree species, since reproduction of the forest environment as close as possible to the original is the main purpose of the work.

As noted above, reclamation involves the preparation, technical and biological stages of the work.

The most important, biological, stage is the creation of artificial forest plantations of different purpose (protective, reclamation, landscaping, plantation, etc.) and is performed by forestry enterprises after mining technical reclamation. it includes measures to restore the fertility of disturbed lands for the cultivation of agricultural and forest crops. It is necessary to start the biological stage as soon as possible, in order to prevent the compacting of loose or level soils, the deterioration of their properties and the spread of weeds and manifestations of water and wind erosion.

The biological reclamation of disturbed lands depends on the composition and properties of the rocks and soils, the physical and geographical conditions of the environment, and the nature of the further use of the reclaimed lands. Soils are classified by suitability for biological reclamation into: suitable, unsuitable and unsuitable.

In biological reclamation widely used herbaceous plants. They quickly anchor the soil mixture and help stop wind and water erosion. In addition, herbaceous plants activate the soil-forming process on the dumps. The species composition of plants used in the creation of vegetation cover in disturbed lands is determined by the natural climatic factors and the water-physical properties of the soil mixture. Common properties for most soil mixtures of disturbed lands - unstructured, poor hydrological regime and nutrient poverty.

Biological reclamation as a set of measures to create a favourable water and air and soil nutrient regimes of forest plantations for different purpose, forestry companies performed after mine reclamation. This set of measures is determined by the physical and chemical properties of the underlying soils and involves the introduction of crop rotations, saturated with crops for sideral fertilizer, introduction of high standards of organic and mineral fertilizers, mulching, etc. It is necessary to start the biological stage as soon as possible in order to prevent compaction of uneven soil soils, deterioration of their properties, as well as the spread of weeds and adventitious species of herbaceous plants and manifestations of water and wind erosion.

It should be noted that the main mechanism of transformation of soil mixture into lithosomes (embryosomes) is the mineralization and transformation of organic matter by microbiological processes. The process of development of lithosomes (embryosomes) includes the following stages: humus accumulation or peat accumulation, leaching of carbonates and removal of soluble salts, structural reorganization of the solid phase, weathered primary materials and glaze. This process is slow.

It should also be noted that forest biogeocenoses are of great importance for solving the problem of restoration, stabilization and protection of the environment. Therefore the choice of direction reclamation should be consistent with the functional features of the landscape.

Experience of reclamation of disturbed lands of Rivne region involves application, after returning of fertile layer of soil to the surface, sowing of perennial lupine and rye.

Since during the illegal extraction of amber, the fertile soil layer was not preserved at all but only destroyed, it is not possible to return it to the level surface of the plots. That is, the environment for the creation of primary vegetation is generally unfavourable.

Therefore, it is considered advisable to apply organic fertilizers after levelling and sealing the surface of the sections, namely peat to increase fertility and to prevent compaction of uneven soil and to form a hard crust. Peat is spread evenly over the entire area of the plots. It should be evenly distributed over the level surface of the plots, in the volume of 30 - 60 t/ha.

The next step is to grow perennials and white lupines to increase fertility and protect pine seedlings from weeds, adventitious herbs and fires.

In the future, the creation of forest crops on disturbed lands greatly intensifies and accelerates the process of soil formation, where not least the breed composition of plantations plays. When selecting wood and shrub species for planting on disturbed lands, the most suitable are native species adapted to the conditions of this region.

After afforestation, the land is transferred to the user for continued use as intended.

Another option, which can be considered as an alternative, is to transfer forest land to another category of land.

An example is the creation of a forest lake with the restoration of part of the forest area. Such a reservoir can have both recreational and technological purpose - as a reservoir for fire or other purposes. The advantage of this option is its lower cost compared to the option of complete restoration of vegetation.

An alternative to the recalmation technology may be the exclusion of the PIE phase of the site. However, refusal to remove the amber remnants makes efforts to restore the natural environment of the affected areas useless, since it does not guarantee the return of the diggers, there are several difficulties of technical (mechanical) reclamation.

An illegal extraction of amber was carried out mainly by hand by chance tool, as well as a hydromechanized method to a depth of 6 ... 8 m, which led to the formation of large areas disturbed by holes, mounds and emissions of sand mass up to 1 m and continuous filling of the surface of the soil. The chaotic location of the holes and the extraction holes makes it impossible to carry out the maintenance and ordering of the forest. Mechanical interference with the soil horizons and soil hydrosurface also led to disruption of the natural system of sediment infiltration, the formation of numerous local zones of saturation with moisture, while simultaneously sealing their periphery, which changed the natural physical and mechanical properties of the soil horizons.

As a result of earthworks from the illegal extraction of amber, the soils have had a very significant impact: mixing of layers, watering the exits of groundwater and atmospheric precipitation, opening the lower horizons (humus-eluvial, eluvial, and even illuvial), precipitation and transformation of their physical and chemical properties under the influence of wind, temperatures and humidity. As a consequence, the light mineral substrate is significantly compacted and cemented, the fertile layer is lost and cannot be recovered naturally.

The chaotic nature of the disturbance of the soil layer makes it impossible to conduct a separate remediation of degraded areas. Permanent reclamation of the territories of all disturbed species is necessary with the preliminary removal of all vegetation cover and carrying out planning works at the stage of technical reclamation.

The above factors significantly hinder, first of all, the technical stage of reclamation. Mechanical backfilling holes and planning surface after removing residual stand will create zones of varying density, leading to the formation of numerous local depression. In order to equalize depression after a year, it will be necessary to re-plan the surface. According to the existing methodological recommendations, after the stage of technical reclamation it is necessary to maintain a certain period during which the soil masses stabilize; such period should last up to at least 3-5 years (after PIE - 2-3 years).

The effectiveness of possible alternative technical solutions, such as mechanical soil compaction by loading and / or periodic wetting, is doubtful not only in view of the high resource and energy costs, but also because they do not provide the necessary uniformity of the mineral soil substrate.

Conclusions

Thus, as a result of studies found that the issue of reclamation, due to the illegal extraction of amber, to prevent a possible environmental disaster of amber containing areas of Rivne, Volyn and Zhytomyr regions of Ukraine are sharp enough and needs urgent solutions, especially at the legislative level.

It is established that plant communities, which spontaneously and slowly develop in disturbed areas, differ from the original phytocenoses. In addition, the drainage ditches significantly violated the hydrological regime, there is a drying up of the territory, which causes the weakening of stands, loss of environmental sustainability and negative vegetation dynamics.

Considering this, it is necessary to carry out continuous reclamation of the sites, which were damaged as a result of illegal extraction of amber.

Studies have found that it is advisable to carry out the rehabilitation of disturbed lands in conjunction with pilot-industrial extraction for the complete removal of amber in order to protect the reclaimed land from unauthorized actions in the future.

Thus, the removal of forest residues and the complete development of land in an open manner is a necessary stage of reclamation. After completion of the pilot-industrial extraction and technical reclamation, additional ecological and economic evaluation of the affected areas should be carried out in order to further extract amber, afforestation and use for agricultural production.

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A STUDY ON THE INFLUENCE OF MINING FACE ADVANCE RATE ON THE RISK OF ROCKBURSTS AND DEFORMATION OF A ROCK MASS AND A LAND SURFACE

Krzysztof TOMICZEK[1]

Ph.D. Eng. Krzysztof Tomiczek, Department of Geomechanics and Underground Building, Faculty of Mining, Safety Engineering and Industrial Automation, Silesian University of Technology, Gliwice, Poland

Summary

The problem of the influence of face advance rate (FAR) on rockburst threats, or more precisely the question of what its rate's value should be, to make it as low as possible, is very controversial. There are various hypotheses about the influence of FAR on rockburst and tremors, they are sometimes contradictory. A similar problem occurs as of the influence of FAR on rock mass and land surface deformations. The chapter presents selected examples of opinions of Polish researchers and the results of numerical modelling for the influence of FAR on deformations of the rock mass and land surface.

Keywords: rock mass stability, face advance rate, influence of face advance rate, tremors, rockbursts, deformations, land surface subsidence

1. Introduction

As a part of the ongoing discussions being kept since the1950s on the influence of face advance rate (FAR) on the occurrence of tremors and rockbursts, the opinion of Professor Sałustowicz (1955) deserves attention. Sałustowicz presented the influence of FAR as a function of a certain critical rate. He wrote: "by increasing the face advance rate, we increase dynamic stress, and thus also the rate of bed deformation and tendency to rockbursts; at the same time, however, we reduce the deformations and the elastic stress. At a certain limiting rate, the static stress decreases to the value of the bed strength, and still below this value, thus the rock bursts disappear."

At the same time, subsidence, bending, cracks and/or landslides may occur on the land surface under the mining. The rock mass deformations arising in the immediate vicinity of the longwall and cover

[1]Krzysztof.Tomiczek@polsl.pl , k.tomiczek@yahoo.co.uk , https://orcid.org/0000-0001-9227-310X Researchgate: Krzysztof_Tomiczek

ever higher layers of the rock mass and propagate to the ground surface. There are two types of deformation: continuous deformation and discontinuous deformation. Continuous deformations, which occur most often, are of fundamental importance due to the wide range of occurrence. They are described by means of characteristic quantities called deformation coefficients. Discontinuous deformations are much less common, but are more damaging. They occur, i.e., in the form of thresholds, funnels, sinkholes and gaps.

The time of occurrence of mining influences depends on many factors: depth of mining, strength and rheological properties of rocks, method of roof control, volume of exploited rocks and face advance rate.

Despite many years of research, no clear answer has been found so far, neither on the influence of face advance rate (FAR) on the intensity of underground earth tremors and rockbursts, nor on the influence of FAR on the rock masses and land surfaces deformations.

2. Selected opinions on the influence of face advance rate on the intensity of underground earth tremors and rockbursts

2.1. Views of some researchers

In general, theoretical considerations and practical experience indicate that as the face advance rate (FAR) increases, the emission of seismic energy increases, and the induced underground earth tremors have higher energy. The relationship found is not linear but rather exponential. It causes that the influence of FAR on the rockburst threat is more pronounced at higher face advance rates (FAR). According to Professor Sałustowicz (1955 and later), bump mines should make slow rate to allow the rock mass for unstressing.

As the velocity of rock deformation increases, so does their strength. Research conducted by Griggs and Handin (1960) and later by other researchers showed that the immediate rock strength is higher than the so-called temporary remained strength. Over time, the strength of the rock decreases.

Wasilewski (1991) stated that the pressure manifestations in the vicinity of the longwall face increase with increasing face advance rate (FAR) and the value of pressure depends on the FAR. He recommends conducting technological tests in the mine in this scope. These considerations concern the rate of longwall face below the so-called critical rate. As long as the longwall face rate in beds of being endangered by rockbursts is up to about 80m per month (i.e. about 2.7 m/day), the critical rate mentioned above is not yet achieved and one must always expect a rockbursts. It is determined by the natural properties of the bed, secondary stresses caused by previous exploitations and stresses caused by the longwall face.

The coefficient of the influence of the face advance rate, thus conditioned, can be investigated only individually for each bed and for each mine.

An increase in the face advance rate causes an increase in the deformation rate, consequently also in rock strength. Consequently, rock damage occurs at higher stress levels.

With the face advance rate increasing rapidly, the strength of the rocks increases and the damage becomes more brittle (see Kłeczek, e.g. 2006). As the face advance rate increases, the amount of seismic energy released by the rocks increases.

The hypotheses assuming that faster rate is more advantage were recognized. For example, Professor Biliński (e.g. 1984) considered as the most dangerous the rate should be pointed from the range of 1.1 to 2.6 m/day.

The analysis of the experimental material carried out by Biliński for five different regions and mines showed that also the length of the longwall for different face advance rate (FAR) significantly determines the hazard of underground tremors.

The smallest tremors occur in longwalls less than 120 m and in longwalls over 300 m. This is probably related to the load-spanning capacity of the roof, thickness of roof layers, strong rock layers and the ratio of the length of the wall to these spans.

The smallest tremors occurred at face advance rates below of 1m/day and at rates of about 3m/day.

There may be a significant increase of hazard in the intermediate rate range. The relationship between the average FAR and the state of rockburst hazard in the rock mass disturbed by exploitation is shown in Fig. 2.1; the most dangerous FAR is the rate within $1.3<v<2.2$ m/day.

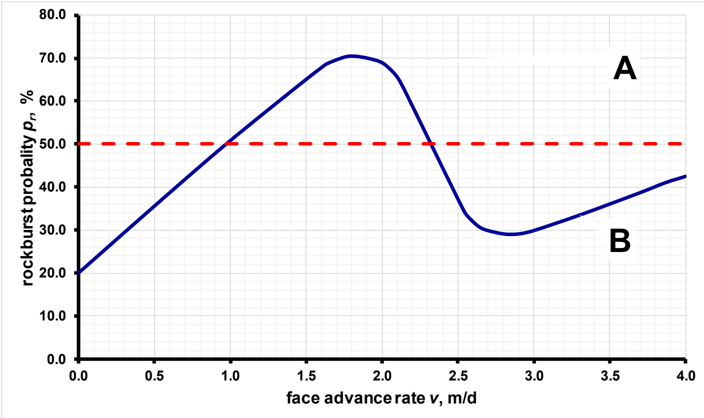


Fig. 2.1. The relationship between the average face advance rate (FAR) v and the state of tremor hazard in the rock mass affected by exploitation according to Biliński. A - I and II degree of rock mass stiffening, B - III and IV degree of rock mass stiffening

Biliński considers two rates of face advance (FAR), i.e. up to 1m per day or above 3m/day, due to rockburst hazards. Other studies show that for all geological and mining conditions, the face advance rate at which the lowest seismic energy is emitted can be determined. Higher energy is generated both above and below this rate. This rule is often impossible to define for a specific longwall. Most often the rule is not confirmed anymore for the neighbouring longwall, carried out in analogous conditions.

Filcek, Kłeczek and Zorychta (1984) analysed strength and deformation characteristics obtained in the rigid hydraulic compression machine. They concluded that there are critical load rates at which total energy has local extremes. These studies relate to the change in the state of stress in the roof due to the variable face advance rate (FAR). At the same time, the authors stipulate that despite the existence of a relationship between the value of total energy accumulated in the rock sample and the degree of rockburst hazard, it is inadvisable to transfer the obtained quantitative results to practice. There is a proportionality between FAR and the rate of loading the bed, but quantitatively the relationship between these quantities is still unknown.

Goszcz (2004) solved the problem of the face advance rate (FAR) theoretically based on the Budryk-Knothe theory and the relationship between the released seismic energy and the gradient of the subsidence rate. The results obtained by Goszcz indicate that the ratio of seismic (total) energy released at two different face advance rates is as much as the squares of the face advance rate. This relationship obtained by adequate boundary conditions and simplifications, describes the phenomenon in a qualitative way. Unfortunately, the verification of this solution is difficult, because there are few documented examples of mining in identical geological and mining conditions with different face advance rate. In most cases, changing the face advance rate is a slowdown, but the exponential nature of the relationship between seismic energy emissions and the face advance rate, rate reduction gives less effects.

According to Goszcz, the geomechanical interpretation of the problem of the face advance rate influencing on seismic energy depends on the rock damage mechanics. Part of the deformation in the loaded rock is elastic and the remaining part is resistant, pseudoplastic deformation. The share of energy depends on the rate of increase of deformation. The faster the deformation (the derivative of deformation in time) is, the greater share of elastic deformation. As a result, with a rapidly increasing load, the deformation of the rock is more elastic and ends with its brittle damage. Mining tremors are the result of brittle rock cracking.

Drzewiecki (2004) conducted research on the influence of face advance rate (FAR) on seismic hazard during the first longwall. He stated that this influence could be even greater when was a second longwall. The range of mining influences as a kinematic parameter of the range of main influences increases. "Disturbance of equilibrium in unstable zones, resulting tremors and rockbursts, may occur in such conditions even at a greater distance from the longwall front than at slow face advance rate, and due to the increase in rock strength, the energy of these tremors may be greater." According to Drzewiecki, a change in the face advance rate, resulting in a change in the range of the active volume of the rock mass and a decrease of rate, reduces the intensity of recorded seismic phenomena.

The Interministerial Institute of Geophysics AGH University of Science and Technology in Cracow researched the relationship between the energy emitted from rocks under different face advance rate. On the basis of over 17,000 mining tremors (with energy greater than 10^3J), recorded by the mines in Bytom's Basin, at different face advance rate, using a statistical calculation, it was determined that:

- there is a clear correlation between face advance rate and the number of low-energy mining tremors; with increasing rate - the number of tremors increases,
- due to the poor material for researching, no relationship was found between the face advance rate and the occurrence of high energy tremors. Generally, the research indicated the adverse influence of high face advance rate on the rockburst hazard.

Research in the area of the 51Z seam section of the "K" mine has shown that as the expansion of mining operations, there is a marked increase in seismic energy activity and tremors. It was found that under normal longwall progress, the zone of maximum energy release and seismic activity displaces deeper into abandoned workings.

In "W" mine in conditions of high rockburst hazard, sectors of longwall in the "clean field" were used primarily for the analysis.

Observations and measurements showed that the caving longwall occurs the phenomenon of side wall unload. This displaces the maximum compressive stress deep into the coal bed and tensile stress - in abandoned workings. This is due to the roof breaking. It should be concluded that the grouping of tremors on the coal field and in abandoned workings have a different character, which depends on the type of stress.

Opinions on the recommended face advance rate vary. It is not uncommon to find opinions about the existence of an optimal rate range $v_{\min} \leq v_0 \leq v_{\max}$, for which the rockburst hazard is lowest.

It is also believed that due to the lower hazard of rockbursts, caving longwall systems are safer than filling systems. This is confirmed by the statistics of rockburst occurrences in coal mines, while the statistics of the seismic activity of the rock mass in copper mines contradicts it. It also seems that due to the smaller deformations of the rock mass during filling operation there are objective conditions for limiting seismic emissions and rockburst hazards. At the same time, however, due to the lower technical level of the filling longwall equipment and the difficulties of full and continuous roof support, in these conditions even a small dynamics event can run to catastrophic effects in the form of rockbursts. Therefore, regardless of the objective causes of rockbursts, at the current stage of development of the mining technique, caving systems should be considered safer because of the rockbursts than filling systems.

Generally, in practice in conditions of high rock burst hazard, as a rule, the face advance rate and longwall heading progress is limited. In beds included in the 1st degree rockburst hazard it is not necessary to limit the face advance rate, and it is even possible to increase it.

However, current research and practice do not provide an unequivocal answer to this problem.

2.2. Opinions of experts-practitioners

Polish experts-practitioners (Konopko et al., 1995) presented their experience on the face advance rate (FAR) and the height of selecting thick beds stratified in layers (Figs 1.2 and 1.3, Tab. 2.1).

Table 2.1

Opinions of expert-practitioners on the face advance rate (after Konopko et al., 1995)

Problem	Type of longwalls	Hazards	Recommended rate
Face advance rate of caving longwalls	Initial	Lower	3.0m/day
	Following	Higher	-
	Final	Highest	2.3m/day
Face advance rate of filling longwalls	Initial	Lower	2.3m/day
	Following	Higher	2.4-2.6m/day
	Final	Highest	2.0m/day
Degree of rockburst hazard	Ist	IInd	IIIrd
Optimal face advance rate	No limitations or >3.6m/day	-	With limitations

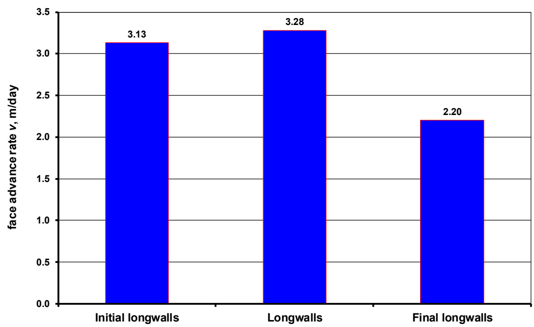


Fig. 2.2. Preferred progress for initial, following and final caving longwalls according to expert-practitioners (after Konopko et al., 1995)

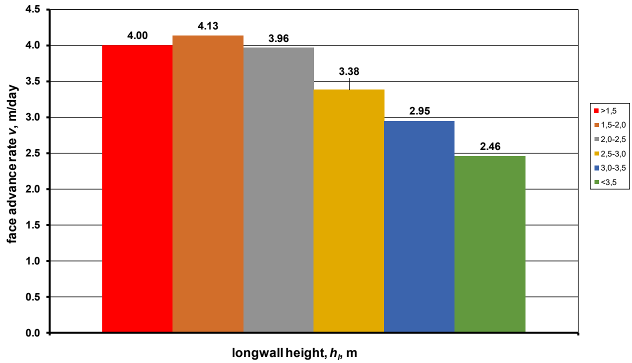


Fig. 2.3. The preferred face advance rate *v* of caving longwalls of various face heights *h* according to experts, assuming a stratification in layers of h_l=2.76-2.00m (after Konopko et al., 1995)

3. Mining-inducted deformations of land surface

The first attempts to measure land surface mining-inducted subsidence were made at the beginning of the 19th century. Further work developed in the field in the 20th century in Poland went in four directions:

- solutions based on empirical formulas; indicator of deformations are determined on the basis of formulas, nomograms, charts - pioneering solutions of J. Gonot, G. Dumont and P. H. Leontowski,

- geometric-integral theories based on axioms of empirical solutions (S. Knothe, Z. Kowalczyk, T. Kochmański, J. Zych, M. Chudek and L. Stefański; see e.g.: Borecki et al., 1980, Chudek, 1980, Knothe, 1984, Zych, 1987),

- research based on continuum models, in which the state of stress and displacement is determined by state equations dependent on the assumed medium model (A. Sałustowicz, J. Litwiniszyn, F. Dymek and H. Gil),

- the theory of the stochastic medium (J. Litwiniszyn, H. Smolarski and J. Mączyński).

3.1. Theoretical basis for calculating mining influence on the land surface and the short characteristics of geometric-integral theories

The rock mass movements resulting from mining are unique determined by the components of displacements (u,v,w) as a functions of the coordinates of the rock mass points (x,y,z) and time t

$$u=u(x,y,z,t)$$
$$v=v(x,y,z,t)$$
$$w=w(x,y,z,t)$$

The slopes of the mining subsidence T in the directions of the coordinate system axis (x, y) are functions of subsidences w and are

$$T_x = \frac{\delta w}{\delta x}$$

$$T_y = \frac{\delta w}{\delta y}$$

Curvatures of the subsiding trough K of continuous deformations

$$K_x \cong \frac{\delta^2 w}{\delta x^2}$$

$$K_y \cong \frac{\delta^2 w}{\delta y^2}$$

Vertical strains - derivative of vertical displacements

$$\varepsilon_z = \frac{\delta w}{\delta z}$$

Horizontal strains

$$\varepsilon_x = \frac{\delta u}{\delta x}$$

$$\varepsilon_y = \frac{\delta v}{\delta y}$$

The geometrical-integral theories are based on the principle that the exploitation of an element with the bed volume dV, that projection on the horizontal plane occupies the area dP results at point A above the roof of bed in the subsidence:

$$\mathrm{d}w_A=agf(x)\mathrm{d}P$$

where

a - exploitation coefficient depending on the method of roof control,

g - bed thickness,

x - horizontal distance of point A from the exploited element dP,

$f(x)$ - influence function; different theories using different forms of this function.

Professor Knothe, adopting the Gaussian distribution of influences based on the analysis of subsidence of land surfaces in the Silesian Coal Basin, derived the formula for the influence function $f(x)$

$$f(x) = \frac{w_{\max}}{r} e^{-\frac{\pi x^2}{r^2}}$$

The function corresponds to the slope of the subsidence through profile

$$T(x) = \frac{dw}{dx} = \frac{w_{\max}}{r} e^{\frac{-\pi x^2}{r^2}}$$

and curvature of the subsidence through profile

$$K(x) \approx \frac{d^2 w}{dx^2} = \frac{2\pi w_{\max}}{r^3} x e^{\frac{-\pi x^2}{r^2}}$$

Extreme curvature values occur at a distance of $0.4r$ from the edge of mining exploitation field and are

$$K_{\max} = \pm\sqrt{\frac{2\pi}{e}} \frac{w_{\max}}{r^2} = \pm 1.52 \frac{w_{\max}}{r^2}$$

The horizontal displacements of points of the subsidence trough profile are given by the formula

$$u(x) = \frac{w_{\max}}{\sqrt{2\pi}} e^{\frac{-\pi x^2}{r^2}}$$

wherein $u_{\max} = 0{,}4 w_{\max}$.

Horizontal strains

$$\varepsilon(x) = \frac{du}{dx} = -\frac{\sqrt{2\pi}}{r^2} w_{\max} x e^{\frac{-\pi x^2}{r^2}}$$

wherein $\varepsilon_{\max} = \pm \frac{w_{\max}}{r} e^{-1{,}5} = \pm 0{,}6 \frac{w_{\max}}{r}$.

3.2. The problem of the influence of face advance rate

Observations and field measurements show that displacements as a result of mining begin to occur with a certain delay, gradually increase and reach their maximum value after a certain time.

It was found that the highest subsidence rate usually occurs in the second year after mining. In some mining basins, maximum landsurfacedown rates occur in the first months after mining, while in others basins - after four years (Knothe, 1980).

When determining the expected displacements, the influence of the time was usually taken into account by introducing a *time coefficient* into the displacement formulas, determining the ratio of the instantaneous subsidence to the maximum final subsidence, depending on the time since mining ending (see e.g. Strzałkowski 2001, Ścigała, 2009.

The time coefficient is also associated with the influence of mining, i.e. the face advance rate (FAR) on the phenomena occurring in the rock mass and on the shape of the subsidence though and the values of land surface deformation coefficients (see e.g. Chudek et al., 2007).

In the 1950s and 1960s, after making many calculations, Knothe stated that the subsidence rate is proportional to the difference between the final subsidence w_k, that a point on the land surface may undergo as a result of exploitation a certain part of the bed up to the time moment t, in the area of mining influence on this point, and the value of the pointdown in the time t

$$\frac{dw}{dt} = c\left| w_k(t) - w(t) \right.$$

where

c - proportionality coefficient, for Silesian Coal Basin c=0.5/year for a predominance of thick and hard sandstones in overlying layers, c=7/year if there are brittle and/or plastic rock layers above the bed.

Measurements and calculations carried out by Professor Knothe have determined that the advantageous effect of increasing the face advance rate (FAR) to land surface protection increases with decreasing c coefficient. It is higher for mining areas where the time after which the maximum ground movements are longer, i.e. for the rock mass with thick and hard rock layers. The measurements also allowed the following conclusion: taking into account the depth of mining, the effect of increasing the face advance rate at smaller depths of mining is more advantageous, than when mining operations are at higher depths.

Based on Knothe's solutions, the problem of land surface deformation over time in Poland, researched: Skinderowicz, Trojanowski, Białek, Drzęźla, Rogowska, Sroka and others.

4. Numerical model

4.1. Geological and mining conditions

Numerical simulations were carried out on an example of mining of the 35B bed in the S coal mine. The simulation was with caving of the roof layers. Two face advance rate (FAR) were simulated: the first "low" and the second "high", i.e. ten times higher than the "low" rate. Calculations were made based on the Finite Differences Method FLAC2D v. 3.23 code.

It was a continuation of research on the behaviour of the rock mass in the vicinity of mining excavations started in 1992 in the Laboratory of Numerical Modelling of the Rock Mechanics Laboratory at the Department of Geomechanics, Underground Construction and Surface Protection (Silesian University of Technology, Kwaśniewski and Wang, 1994).

The rock mass is made of Quaternary and Carboniferous formations. Coal beds are variable with numerous thinning and outpetering. Slate-silty layers occur in the roof and floor. Sandy shales change into fine-grained sandstones. The dip of layers in the SW direction is 3-7°.

The 35B bed lies at a depth of -465m in the north to -580m in the south. The bed has an average thickness of 2.1m. It is a non-rockbursting bed. In the roof of this bed in the vicinity of the 100C longwall there are silt and sand shales, and in the floor - too. The mining area of the 35B bed with the 100 C longwall is enclosed by faults: from the north "K" with a throw of 60 m, and from the east "J" with a throw of 30 m. From the south it is limited by the line of the mining area of the M mine.

The high-efficiency longwall complex 100C equipment consists of:

- mechanized hydraulic support Glinik-13/26 licensed by MECO, equipped with electrohydraulic control,
- Anderson ELECTRA 1000 shearer from Anderson-Longwall, with a maximum mining rate of 9.97 m/min.,
- MECO longwall conveyor, with a nominal capacity of 1500 t/h,
- MECO chain longwall conveyor, the capacity 1500 t/h,
- MECO 6×420 heading crossing support with extended roof-bars.

Data characterizing the mining of the 35B bed by the longwall 100C:

- method of roof control: roof layers caving,
- longwall length: 300 m,
- longwall stopway: 1,248.8 m,
- date of starting mining 16.01.1993,
- date of finishing mining 31.12.1993,
- longwall height 2.1±0.2 m,
- face advance rate over a period of 1 month: from 62 to 196m/month,
- maximum daily rate 12.5 m/day,
- average daily rate: 3.57 m/day.

4.2. Characteristics of the FLAC2D code

The FLAC2D code is a finite difference program. It enables the building of numerical rock mass models and simulation of the behaviour of soil and rock both after reaching the point of plasticity (plastic flow) and the ultimate strength (brittle fracture). Due to its properties, FLAC enables solving rock engineering problems, analysis and design of excavations and their supports, underground and ground-based structures. FLAC2D is based on the Lagrangian calculation scheme.

4.3. Rock mass model

A 2D, numerical, structural and physical model of the rock mass was built, 2250m long and 650m thick. The model was divided by "W" and "J" faults into three parts: left - west, middle and right - east.

The based model was built in 1993 for the research work carried out at the Laboratory of Rock Mechanics of DoGUBaLSP as part of ministerial project No. 231/CS6-9/92 entitled "High-efficiency longwall complex and new coal mining technology in the S mine."

Fifty-two rock layers in the roof and twelve layers in the floor of the coal bed lying at the depth of 535.25m meters with a thickness of 2.25m were distinguished and modelled. The rock mass flat model was divided into 9750 rectangular elements $150(L)\times65(H)$ with an average length of 15m and a width of 10m, forming a finite difference grid with 9,966 nodes (Fig. 4.1).

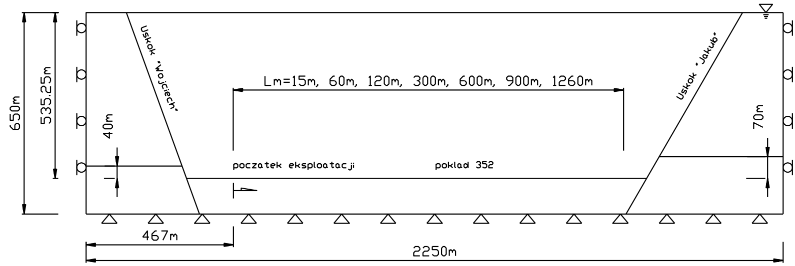


Fig. 4.1. A simplified scheme of the modelled rock mass plate with seven stages L_m of 35B bed mining

Two large faults - west "*W*" and east "*J*" dividing the tested rock mass into three parts, were modelled with the built-in FLAC2D model of the discontinuity between two parts of the finite difference mesh.

The longwall mechanized support was modelled using a 10-units structural element 3.5m width and 2.25m high.

In order to study the changes in stresses and displacements in the rock mass associated with the mining of the 35B bed with the 100C longwall, the bed mining was simulated in seven stages of L_m=15 m, 60 m, 120 m, 300 m, 600 m, 900 m, and 1260 m.

The 35B bed was mined with roof layers caving.

Depending on the distance from the longwall face, rocks in the caving zone were assigned different properties, for example values of volume density, bulk and shear modules (Figs 4.2 and 4.3).

Example:

Stage 1.

L_m=15 m $\quad$ L_c=0 m $\quad$ H_c=0 m

L_m - the length of the mined part of longwall

L_c - range of the caving zone

H_c - height of the caving zone

model null i=32, j=13 y=-535,25 m

Mechanized support:

support x=479,55 mod null i=32 j=13 y=-535,0

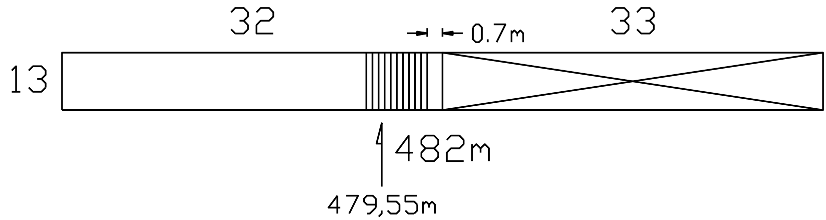


Fig.. 4.2. Simplified scheme of the model of the first stage of mining (L_m=15 m)

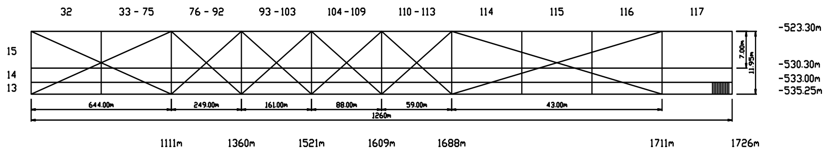


Fig. 4.3. Simplified scheme of the model the seventh stage of mining (L_m=1260 m)

Simulating the mining at "low" rate, calculations were completed after 13,000 steps, which corresponded to the 350-day mining of the 35B bed by a 100 C longwall and an average face advance rate (FAR) of about 3.6m/day. Simulation of mining at "high" rate (10 times higher) was completed after 1,300 steps.

The results of the first attempts to simulate exploitation at different face advance rate can be found in the research of Kwaśniewski and Wang (1994). The development and full analyses of the project are in the MD thesis (Tomiczek, 1995).

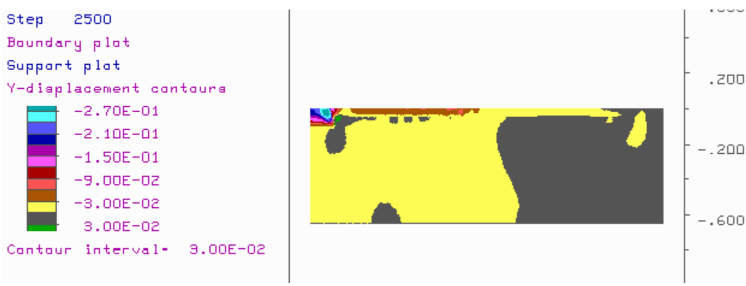
4.4. Behaviour of the rock mass at the end of selected stages of bed mining

Due to the volume limitations of the chapter, only selected simulation results are presented (Tab.1 and Figs 4.4-4.11). For the same reasons, the results of calculations were not discussed, leaving conclusions to the readers.

Table 4.1

Values of deformation coefficients in the part between "W" and "J" faults for various stages of L_m exploitation and face advances rates (FAR); L_f - distance from longwall face, L_e - range of mining influences in the right side (end point of mining)

Rate/ distance from longwall face L_f, m	w_{max} mm in the part between faults	ε_{xmax} mm/m in the part between faults Side		T_{max} mm/m in the part between faults Side		Range of mining influences in the right side (end point of mining) L_e, m
		Left	Right	Left	Right	
Stopway of longwall L_m=60m						
"low"	-74.90	-0.168	0.242	-0.376	0.590	-
L_f, m	100.50	724.5	490.5	705.0	471.0	1134.0
"high"	-62.30	-0.168	0.200	-0.245	0.521	-
L_f, m	393.00	724.5	490.5	276.0	471.0	527.0
Stopway of longwall L_m=600m						
"low"	-910.00	-1.690	0.985	-2.000	2.900	-
L_f, m	-322.50	-244.5	204.0	-614.5	28.0	516.0
"high"	-148.80	-0.245	0.284	-0.363	0.558	-
L_f, m	-439.50	-615.0	-49.5	-556.5	-49.5	535.5
Stopway of longwall L_m=1260m						
"low"	-1508.00	-2.058	0.558	-2.684	2.300	-
L_f, m	-747.50	-630.5	-396.0	-1234.5	-26.0	-
"high"	-486.10	-0.752	0.458	-1.020	1.479	-
L_f, m	-825.50	-903.5	-708.5	-1235.0	-630.5	-
Stopway of longwall L_m=1260m "high" rate in relation to the time of end of "low" face advance rate (FAR)						
"high"	-973.00	-1.030	0.200	-1.579	1.526	-
L_f, m	-513.50	-626.0	-1352.0	-1234.5	12.5	-



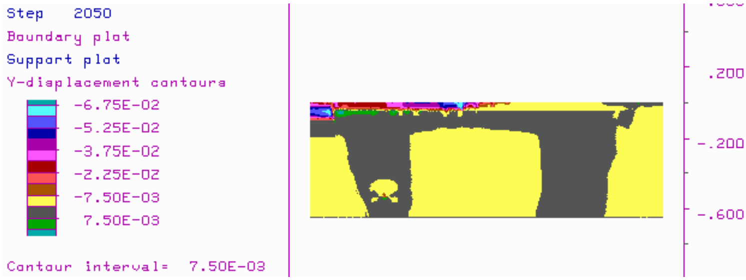


Fig. 4.4. Vertical displacement of the rock mass for "low" (top) and "high" (bottom) face advance rate, in the first (L_m= 15 m) stage of mining

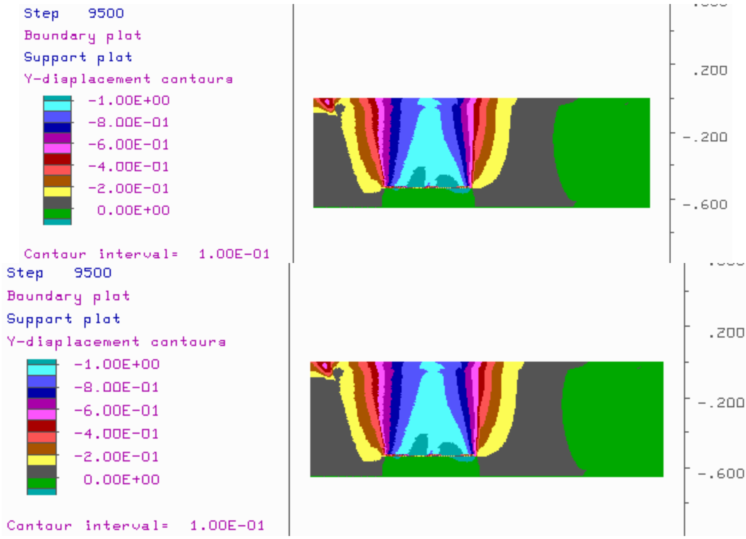
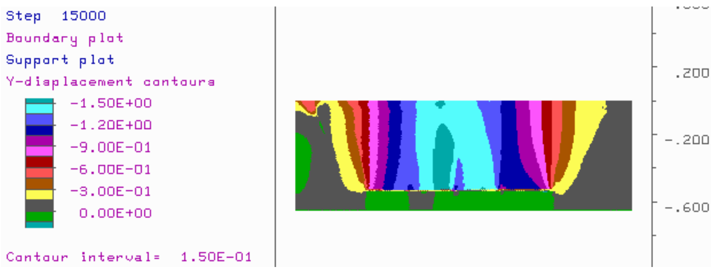


Fig. 4.5. Vertical displacement of the rock mass for "low" (top) and "high" (bottom) face advance rate, in the fifth (L_m=600 m) stage of mining



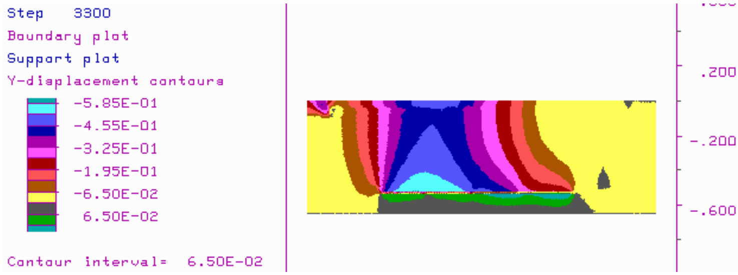


Fig. 4.6. Vertical displacement of the rock mass for “low” (top) and “high” (bottom) face advance rate, in the seventh (L_m=1260m) stage of mining

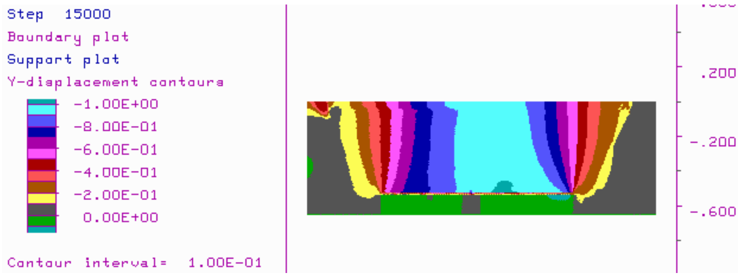


Fig. 4.7. Vertical displacement of the rock mass for “high” face advance rate, in the seventh (L_m=1260 m) stage of mining (after a time corresponding to the finish of mining with “low” rate).

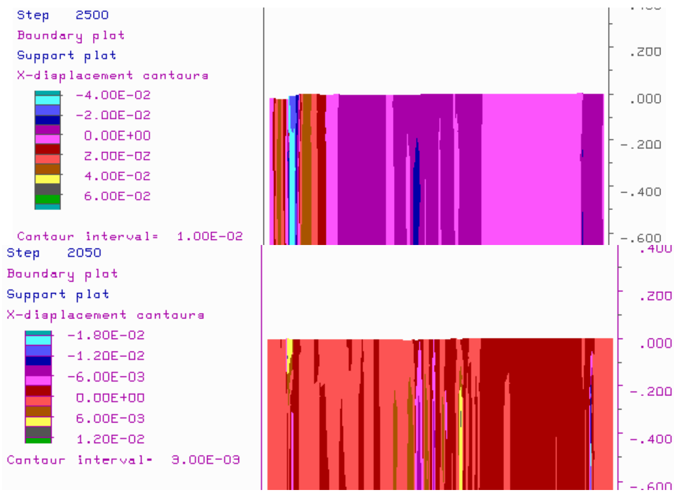


Fig. 4.8. Trough subsidence profile for “low” (top) and “high” (bottom) face advance rate, in the first (L_m=15 m) mining stage

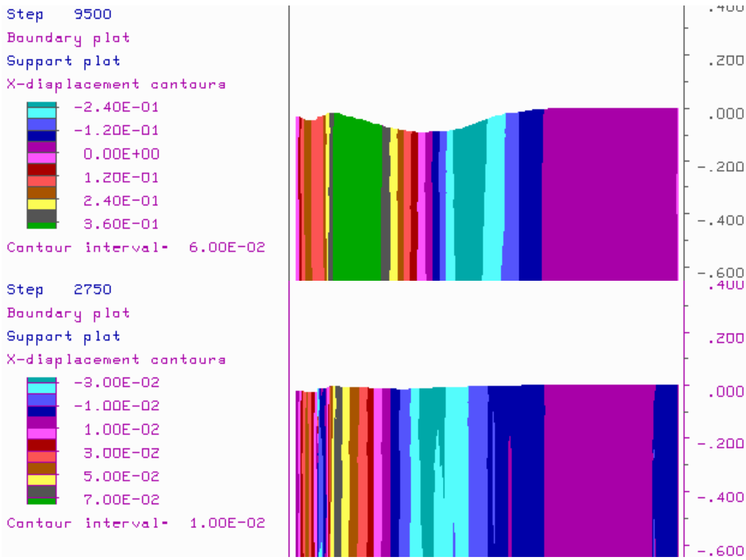


Fig. 4.9. Trough subsidence profile for "low" (top) and "high" (bottom) face advance rate, in the fifth (L_m=600m) mining stage

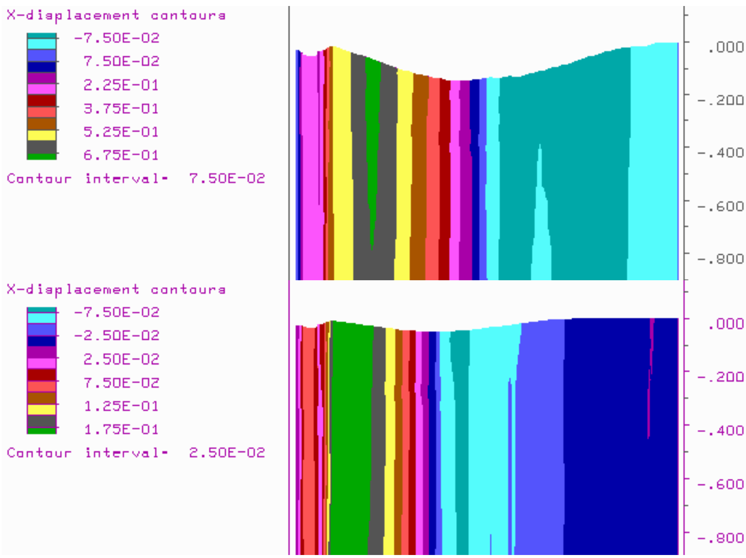


Fig. 4.10. Trough subsidence profile for "low" (top) and "high" (bottom) face advance rate, in the seventh (Lm=1260m) mining stage.

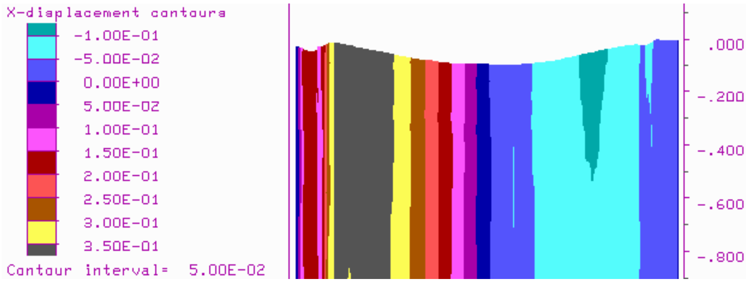


Fig. 4.11. Trough subsidence profile for “high” face advance rate, in the seventh (L_m=1260 m) mining stage (after a time corresponding to the finish of mining with “low” rate).

The results of numerical modelling and simulation of the rock mass simulation of coal bed longwall mining allowed to state that increasing the face advance rate leads to a decrease in the values of deformation coefficients and the range of continuous and discontinuous deformation.

In the initial stages of mining, e.g. in the first stage, when the bed was excavated a distance of only 15 m (L_m=15 m), these differences were small. For example, the maximum values of displacement vectors of the supported part of the longwall excavation roof were 1.80 and 2.23 mm for "high" and "low" rate respectively.

The differences, to the advantage of "high" rate, increased as the longwall stopway growths and became clearly noticeable from the fourth operating stage (L_m=300 m).

After finishing the bed mining (seventh stage, L_m=1260 m), the maximum displacement vectors in the vicinity of the longwall face, equal to 98.9cm in the case of mining at "low" rate, were more than four times larger than those equal to 23.5 cm, corresponding to "fast" face advance rate. The direction and distribution of displacement vectors in the immediate vicinity of the longwall face was different.

For "low" rate, the zone of increased rock mass subsidence is much more regular, symmetrical than that corresponding to "high" face advance rate. The maximum values of land surface subsidence equal to 150.8cm, are more than three times larger and cover larger land areas; for the "low" rate, the subsidence extended to the entire study area. In the depth of the rock mass, in the rock caving zone above a selected part of the bed, the maximum subsidence was 157.5

cm when mining at "low" rate. They were about 2.7 times larger than the subsidence resulted the "high" face advance rate. The values of the roof displacement vector in the supported part of the longwall excavation are more than seven times higher.

The damage zone corresponding to the "low" face advance rate occupies almost the entire rock mass between the "J" and "W" faults, while at "high" rate it is limited, except the plasticized soils near the land surface, only to the immediate vicinity of the selected bed.

Equally large differences occur in the maximum values of slopes and horizontal deformations of the land surface. Twice larger, reaching 2.68mm/m when operating at "low" rates are compared to those corresponding to the operation of "fast" face advance rate slope values. Horizontal deformations are also in the west part of basin three times higher.

5. Summary and final remarks

The problem of the influence of the *time coefficient* on the behaviour of the rock mass in the vicinity of longwall excavations has become very important because in-, but not only, coal mines, high-efficiency mining complexes are being bright in an increasingly larger scale.

These complexes allow mining exploitation coal beds at rates reaching several meters for a day. Due to the tremors and rock bursts, protection of the land surface, ground and underground structures, it is necessary to study influence of the face advance rate on seismic activity and rock mass deformation.

The chapter presents selected results of a literature study and own research on the problem, as a possibility of beginning further researches.

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RESOURCE-SAVING AND RELIABILITY OF TRANSPORTATION SYSTEMS

Tkhoruk E.I.
National University of Water and Environmental Engineering (NUWEE), Associate Professor, Candidate of Technical Science, Associate Professor, Department of Transport Technologies and Technical Service, Ukraine

Krystopchuk M.E.
National University of Water and Environmental Engineering (NUWEE), Associate Professor, Candidate of Technical Science, Associate Professor, Department of Transport Technologies and Technical Service, Ukraine

Soroka V.S.
National University of Water and Environmental Engineering (NUWEE), Associate Professor, Candidate of Agricultural Sciences, Associate Professor, Department of Transport Technologies and Technical Service, Ukraine

Holotiuk M.V.
National University of Water and Environmental Engineering (NUWEE), Associate Professor, Candidate of Technical Science, Associate Professor, Department of Transport Technologies and Technical Service, Ukraine

Abstract

The article is based on the analysis of the development of the theory of reliability and its indicators are formulated indicators of the reliability of transport systems and built a system of factors determining their reliability. It is established that the formation of the theory of reliability of transport systems today is not completed. Based on the specifics of the systems, new reliability indicators were proposed and substantiated such as emergency downtime and reliability of emergency downtime. On the basis of the performed studies, it was found that the specifics of the transport process require the introduction of emergency downtime close to the recovery time and the reliability of emergency downtime. It is necessary the introduction of two close parameters such as the average recovery time and the average emergency idle time. They are associated with the need to distinguish between the idle time of a technological system or a separate element when a failure occurs and the idle time is need-

ed. For the technological links of the transport system, the main purpose of which is to ensure efficient operation transport process, the probability of emergency downtime can serve as a criterion for assessing their reliability.

1. Introduction

The problem of reliability of transport systems is one of the main ensuring and improving the efficiency of their functioning. The reliability parameter of the transport system is one of the main indicators of the quality and efficiency of its operation, it manifests itself in time and reflects changes occurring in the system during the whole period of its operation. To date, the situation with the assessment and ensuring of the reliability of transport by road significantly impedes the achievement of the transport system indicators of transport strategy indicated in the Transport Strategy. The quality indicator in transport offered by scientists today is reliability taking into account mainly only the reliability of technical means, while reliability of performed services is practically not considered. There is also no standardization of reliability in road transport and in reports of motor transport enterprises there are no criteria characterizing reliability and efficiency of their work. The formation of the theory of reliability of transport systems is not completed to date. Therefore, the bulk of organizational and technological decisions to reduce the number of failures is based on the knowledge of the methodology for solving traffic safety problems, technical maintenance of vehicles, situational management of motor transport, the theory of reliability of technical systems, risk management, supply chain management and other scientific fields.

The quality of transport system operation is set by the consumer. Despite the large variety of indicators of quality of delivering goods and passengers by road, they are based on the same or similar parameters. It is appropriate to evaluate the quality of the transport system by its accessibility, functionality and reliability, as suggested by D. Baueroks and D. Kloss regarding to supply chains [18].

Based on this statement, reliability is one of the most important components of assessing the quality of the transport system. The established interconnection of efficiency, quality and reliability of transport system operation prevents these definitions, simplifies the choice of criteria of reliability estimation, formulation concepts of "reliability" and "failure", as well as the concept of "Reliability management of transport systems".

In researches, that examin the problem of quantitative assessment of the reliability of cargo delivery systems, usually is implemented an approach based on the use of terminology and results in the field of ensuring the reliability of technical facilities and conducting appropriate analogies. In technology, reliability includes, among other properties, failure-freeness [19]. Reliability is considered satisfactory if the failures do not exceed the specified level. A failure is called a breakdown, disability, or event after which the operating parameters of the unit or machine go beyond acceptable limits. The reliability of the functioning of transport systems should be evaluated by the number of breaches of the contract for delivery. If there are no such violations when performing the transport service, then the reliability is at the highest possible level.

Thus, the reliability of transport systems is a complex feature that includes the ability of the transport system to meet the agreed upon between the customer and the service provider requirements for the amount and condition of the cargo being carried, the safety of passengers and their luggage, adherence to the schedule of the transport process, as well as support and restoring the specified level of transport service.

By analogy with the definition given, it would be logical to assert that the management of the reliability of transport systems for the delivery of goods and passengers by road involves the implementation of measures ensuring the level of reliability agreed by both sides, its support and further enhancement to the economically feasible limit.

Fundamental to the theory of reliability is the concept of "failure". This definition is based on the methodological apparatus of analysis and quantification of the reliability of objects, processes and systems, regardless of their complexity, purpose and scope.

In the theory of reliability of technical systems, failure is an moment after which the operating parameters of the unit, machine or process go beyond the permissible limits. This definition is valid for both simple technical objects and complex technical systems and technological processes [20]. By counting the number of such events per unit of time, you can estimate the reliability of the object.

Reliability will be satisfactory provided if the number of failures does not exceed the specified level. In developing this approach, the failure of the transport system can be defined as a failure to fulfill an accepted order (request) for delivery, as well as a violation of the agreed in the contract requirements for the quantity and quality of the goods being carried, the safety of passengers and luggage, the schedule of the transport process, the restoration of a given level of transport servicing [21].

The problem of reliability in transport has significant features and is not limited to the reliability of technical means. It also determined the conditions and laws of all elements of the transport process [15]. The main problems of the theory of reliability are to establish patterns of occurrence of failures and their recovery; determining the quantitative characteristics, methods of calculation of reliability. The most studied in this it is considered theory of reliability technical systems. Principles of reliability of technical objects are adapted to the conditions of reliability of supply chains [5], [17]. Reliability transport systems and road transport process has methodological connection with the reliability of technical systems since to improve the quality of delivery of cargo and passengers as well as in technique applied reservation. There are structural [11] and functional reservation [10], [12].

In reliability theory, in particular, processes occurrence of failures, recovery of production systems elements are described by methods of probability theory [16].

Pronnikov A.S., Druzhynin G.V. consider the reliability of the system based on determining the function of readiness system that consists of n subsystems. In this case the function of readiness system describes the probability of finding the system in working condition at any time [13], [7].

The problem of reliability is complex both in terms of its characteristics and measurements, and its cause-effect displays. Thus the reliability of vehicles is based on the classic approach to reliability that investigates the reliability of all systems and mechanisms that in recent works called "mechanical reliability" [8], and the reliability of the technical maintenance and repair and reliability in the process of commercial exploitation also take into account organizational and economic factors. In reliability theory, considerable attention is given to the impact of damages from insufficient of reliability machines as an economic expression of technogenic risk [14]. Aulin V.V., Golub

D.V engaged in issues of legal regulation to provide reliability of functioning transport systems in Ukraine [1], also they have investigated methods for assessing and analyzing the reliability of automobile transport systems [2], [3], [4].

Classical methods of quantitative assessment of indicators of reliability of complex systems [9] applied to systems for which the notion of failure is clearly identified, that system is in one of two states: in a state working or in a state of recovery.

The proposed in the article recommendations are based on the definitions of probability reliability indicators of transport systems.

2. Materials and Methods

The main tasks of reliability theory are the establishment of patterns of occurrence of failures and their recovery, determination of quantitative characteristics, development of methods of evaluation and calculation of reliability. There are two approaches to solving these problems. The first approach is to study statistical the patterns of failure of the same type of technical means in certain operating conditions and is the basis for the establishment of distribution laws of the investigated parameters and obtaining of their numerical characteristics operational reliability.

Found in this way specifications which are used for calculating the reliability of technological systems consist of the types of technical means for which they are identified.

The second approach is aimed at studying the physical nature and mechanism of failures. It serves as a basis for the development of measures to improve the reliability of existing and planned technological systems. This approach is more efficient in terms of identifying physical essence reasons which caused the deviations options below acceptable limits, which means rejection. This path is used where, despite specificity of operation is still possible to obtain the necessary input data, such as the study of the reliability of transport systems.

To obtain analytical expressions of reliability of a transport system, we assume that it operates according to the following scheme: subsystems and elements, which were rejected, begin to recover; there are no restrictions on the number of restorations; failure of one of the elements or subsystems will result in the refusal of the transport system as a whole.

To evaluate the reliability of the elements of transport systems used probabilistic parameters of reliability.

The reliability of the functioning of transport systems and processes of road transport is a complex feature that includes the ability of the system to fulfill the requirements for the number and condition of the cargo carried, the safety of passengers and their luggage, compliance with the schedule of the transport process, as well as support and restoration of a given level of transport service.

One of the parameters of reliability is the probability of failure operation work. Failures in the transport system may be caused by some emergency situations in the functioning of its elements. Multi-transport system can occur in three forms: while eliminating the consequences of the situation in the transport process, when combined parties of their functions, the redevelopment of activity of the participants of the transport process. Any emergency situation caused by the refusal arising in the performance of transportation usually leads to changes in the technology of work its members. Reliability of transport system depends on several factors. (fig. 1)

Some emergency situations in road transport are common, such as a traffic accident or technical malfunction of the vehicle on line [11].

The algorithm of actions and approximate duration of treatment elimination of consequences of the typical situations is standard and is known in advance. Work by the transport process in the mode of elimination of the accident is a reserve (auxiliary) function and work as usual is the main function. Multifunctionality is reflected in the form of performances and methods of solving problems for managing reservation reliability of the transport process and its components.

In this case, based on the fact that at any time t state of the element or subsystem is described by the random vector

$$x(t) = \left[x_1(t), x_2(t)),...x_n(t)\right] \tag{1}$$

which can take two values always at the one-dimensional variable: $x(t)$=1, if the system or component is in working condition, and $x(t)$=0 in case of failure. The components of the vector $x(t)$ can be the values of various parameters of the system.

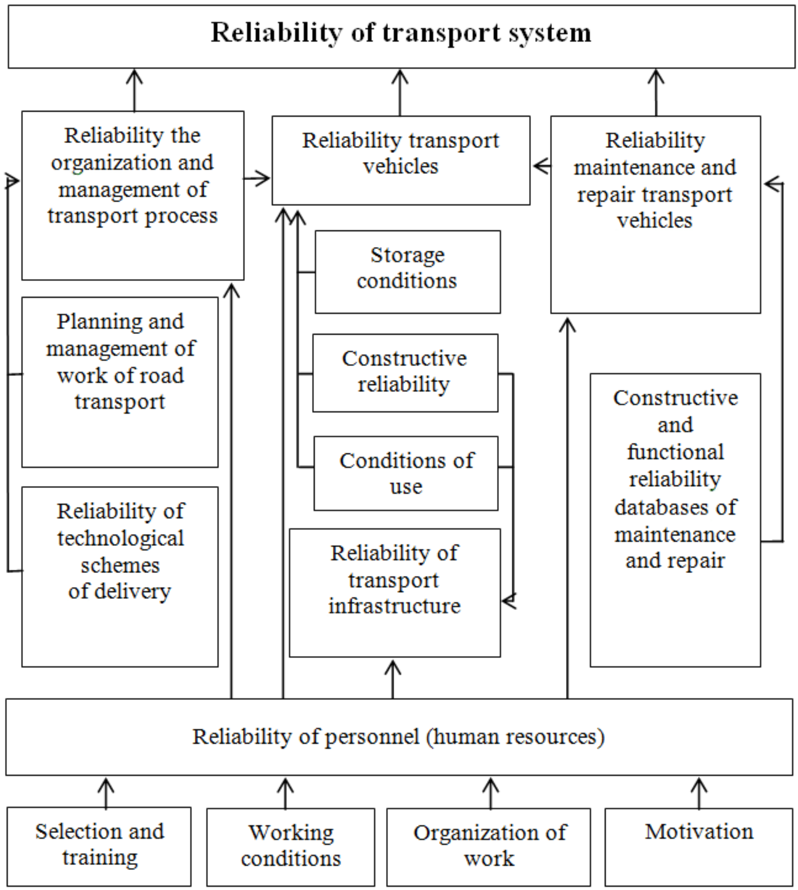

Fig. 1. Factors that determine the reliability of transport system

Random vector $x(t)$ is characterized by a probability distribution $F(x_1,..., x_2,x_n,t)$ i.e. the probability that the $x_1(t) \leq x_1,..., x_n(t) \leq x_n$.

Each of the limit values that characterize the state of the system meets the mathematical expectation of the objective function $G(t)$ at the interval of time $a \leq x \leq b$.

Reliability of elements that make up the technological system, characterized probabilistic indicators (table 1), the most important of which are the uptime and recovery time. These options are probabilistic in nature and describe the relevant laws of distribution.

Table 1

Indicators of reliability

Recoverable elements		
№	The name of indicator	Mathematical expression
1	Average uptime	$T_u = \int_0^\infty P(t)dt$
2	Average recovery time	$T_{ar} = \int_0^\infty \tau\, d\nu(\tau) = \int_0^\infty G(\tau)d\tau$
3	Reliabilities	$P(t) = Bep(T \geq t)$
4	Failure rates	$\lambda(t) = \dfrac{f(t)}{P(t)} = \dfrac{d\ln P(t)}{dt}$
5	Reliabilities for the time (0, τ)	$\nu(t) = Bep(T_B \langle \tau)$
6	The probability of no recovery time (0, τ)	$G(\tau) = 1 - \nu(t) = Bep(T_B \geq \tau)$
7	The density of probability of recovery the time τ	$\nu(t) = V\langle \tau)$
8	The intensity of recovery the time τ, τ_i – the duration of recovery of i-th failure, n- the number of failures	$\mu(\tau) = \dfrac{\nu(\tau)}{1 - V(\tau)}, \quad \mu = \dfrac{n}{\tau_i}$

Transport systems relating to renewable systems that are characterized by intervals periodically alternating correct operation and restore during the failure.

Because of this indicators of reliability is selected in such a way that they can be would be to assess the reliability of individual elements, and generally renewable systems consisting of different types of elements.

Indicator of reliability which taking into account the uptime T_u and downtime τ_d is readiness coefficient

$$K_R = \frac{T_u}{T_u + \tau_d} \tag{2}$$

and the coefficient of failure

$$K_f = 1 - K_R = \frac{\tau_d}{T_u + \tau_d} \tag{3}$$

In fig. 2 presented the dependence of the readiness coefficient and the coefficient of failure of uptime and downtime consumer.

a *b*

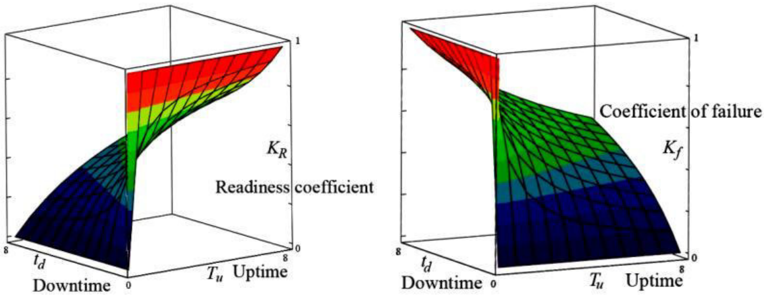

Fig. 2. Dependence of the readiness coefficient and the coefficient of failure of uptime and downtime: *a* - dependence of the readiness coefficient of uptime and downtime; *b*- dependence of the coefficient of failure of uptime and downtime

In view of the data table1 we get

$$K_R = \frac{\int_0^\infty t \cdot f(t)dt}{\int_0^\infty t \cdot f(t)dt + \int_0^\infty \tau \cdot dV(\tau)} \tag{4}$$

$$K_f = \frac{\int_0^\infty \tau \cdot dV(\tau)}{\int_0^\infty t \cdot f(t)dt + \int_0^\infty \tau \cdot dV(\tau)} \tag{5}$$

i.e. two parameters are characterized by distribution law uptime and recovery time and reflect the probability of finding components or systems in working order or in a state of failure, respectively.

The relative time of consumer's downtime by reason of failure of the system over a period of work.

$$\overline{P}(e) = \frac{t_D}{T_{co}} = \frac{t_D}{t_D + T_{UP}} \tag{6}$$

where: t_D is downtime of consumer for over a period of work;

T_{UP} is uptime for a certain period.

Indicator $\overline{P}(e)$ of (6) is the probability of stimulated (emergency) downtime and form with a probability of a working state of complete group of events

$$\overline{P}(e) + P(t) = 1 \tag{7}$$

These terms define more precisely the probabilistic nature of the indicators of reliability. They take into account the frequency of occurrence of failure and downtime and represented more comfortable by using.

Any considered technological scheme can be represented as a system composed of a number of elements. If the i-th element of the system has been in operation for the time T_E then this value consists of alternating time intervals, correct operation t_{co} and recovery time T_r. However, the recovery failure T_r does not fully reflect the time of consumer in forced waiting for service. This specificity characteristic for the transport process requires the introduction of another indicator that comes close to recovery time, namely, time of emergency downtime t_{ed}.

Significantly it affects the type of the distribution of the random variable on the accuracy of results in calculations for reliability.

For example, the exponential distribution law uptime element or some technological system indicates that the flow of failures can be adopted the simplest.

Be noted the basic properties of the simplest flow failures:

a - failure rate is constant (λ=const) and average uptime

$$T_u = \frac{1}{\lambda} \tag{8}$$

b - density of probability of time intervals between the neighboring failures

$$f(t) = \lambda \cdot exp(-\lambda t) \;\; f(t) = \lambda \cdot exp(-\lambda t) \tag{9}$$

c - probability of getting n events in the time interval $[t, t + \Delta t]$ gets according to Poisson

$$P_n = \frac{1}{n!}(\lambda t)^n exp(-\lambda t) \;\; P_n = \frac{1}{n!}(\lambda t)^n exp(-\lambda t) \tag{10}$$

where λt is the average number of failures in an interval of duration t;

d - the probability of absence of failures in the interval duration τ, which starts at a random time determined by the equation

$$P_0 = exp(-\lambda t) \tag{11}$$

3. Results

The simplest flow failures and recoveries in transport technological systems can be quite completely described by three numerical indicators of reliability, which are important to get to practical use. These characteristics are the average intensity of failures λ_{av}, average recovery time T_{ar} and the average emergency downtime t_{av}. The introduction of two close parameters: the average recovery time T_{ar} and the average emergency downtime t_{av} is associated with the necessity of the separation time finding in state of downtime technological system or a separate element with the appearance of failure and downtime of consumer.

Probability of emergency downtime defined by the formula

$$\overline{P}(e) = \frac{t_{ME}}{t_{ME} + T_{co}} \tag{12}$$

where t_{ME} is the mathematical expectation of time of forced (emergency) downtime, $t_{ME} \approx t_{aed}$;

T_{co} is the time of correct operation during the observation time;

$T_{co}+t_{ME}=T_o$. is the observation time.

Or

$$\overline{P}(e) = \frac{n \cdot t_d}{N \cdot T_o} = \lambda \cdot t_d = \frac{t_d}{T_u} \tag{12'}$$

where n is the number of failures of elements of the system during the observation time T_o.;

N is the number of elements of the system which are under the observation.

From (13) implies that the probability of emergency downtime represents size of relative of time of emergency downtime and therefore can be used to determine the economic damage caused by interruptions in the functioning of technological systems.

The probability of emergency downtime linked to probability of faultless work with the following ratio. With exponential distribution law uptime

$$P(t) = exp(-\lambda t) = exp\left(\frac{t}{T_u}\right) \tag{13}$$

whence

$$\lambda = -\frac{\ln P(t)}{t}, \quad \overline{P}(e) = \lambda t_d \tag{14}$$

If the probability of faultless work *P(t)* is given, then

$$\overline{P}(e) = \lambda_{av} t_{ed} \tag{15}$$

4. Discussion

Thus, specificity which is typical for transport process requires the introduction time of emergency downtime t_{ed}, which is close to the time of recovery. This indicator can be attributed to one of the main numerical reliability features that characterize a simplest flow failures and recovery in transport technological systems.

The introduction of two close parameters: the average recovery time T_{ar} and the average emergency downtime t_{aed} is associated with the necessity of the separation time finding in state of downtime technological system or a separate element with the appearance of failure and downtime of consumer.

For technological parts of the transport system, the main purpose of which consists in maintenance conditions for effective functioning of the transport process, the probability of emergency downtime can serve as a criterion for evaluating their reliability.

5. Conclusions

An important indicator to assess the reliability of renewable transport systems is the probability of an emergency downtime, which reflects both the frequency of failure and the idle time.

From the practical point of view, the above models allow us to determine the general probability of a forced idle or other reliability measure for virtually any scheme of the transport process, and on this basis to assess its reliability either in absolute value of the indicator, or with the help of economic expression of reliability in the form, for example, mathematical expectation of damage.

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SOME ASPECTS OF DEVELOPMENT AND APPLICATION OF THE BEARING-BOLT SUPPORTING TECHNOLOGY

Krukovskyi O.P.
Institute of Geotechnical Mechanics named by N. Poljakov of National Academy of Sciences of Ukraine, Corresponding member of NAS of Ukraine, Doctor of Technical Sciences, Deputy Director of the Institute, Ukraine

Krukovska V.V.
Institute of Geotechnical Mechanics named by N. Poljakov of National Academy of Sciences of Ukraine, Senior Researcher, Doctor of Technical Sciences, Senior Researcher in Department of Dynamic Effects of Rock Pressure Control, Ukraine

Bulich Yu.Yu.
Institute of Geotechnical Mechanics named by N. Poljakov of National Academy of Sciences of Ukraine, Researcher in Department of Rock Mechanics, Ukraine

Zemlianaia Yu.V.
Institute of Geotechnical Mechanics named by N. Poljakov of National Academy of Sciences of Ukraine, Chief technologist in Department of Rock Mechanics, Ukraine

Abstract. Geological and technical conditions in the most of Ukrainian mines are significantly more complicated than in the mines of other leading countries, which produced coal. A simple transition of the accumulated foreign experience of anchoring did not give any effective positive results. The goal of this investigation is to study more deeply mechanism of the roof-bolt operation and to specify the space-time laws of geomechanical processes occurred in the system "edge rock – roof bolts". The research tasks are: to design a mathematic elastoplastic model for calculating stress state of the rocks around the roadway supported with the bolts; to explore support formation around the bolt in the mine roof; to establish a concept of the rock-bolt support interaction, according to which supports unite into a single construction that can resist rock pressure; to develop the bearing-bolt supporting technology and expand range of its application.

These tasks have been completely resolved. The technology of the reinforced and powerful bearing-bolt supporting designed by our Institute of Geotechnical Mechanics was successfully realized in practice in 52 Ukrainian mines for driving more than 700 preparatory roadways and permanent workings. This approach helped to achieve stable state of the roadways under the complicated mining and geological conditions and obtain good economic effect thanks to the cut expenses spent to the roadway supporting and repair.

Key words: anchoring, complicated mining and geological conditions, geomechanical processes, mine workings, numerical simulation, supporting technology.

1 Introduction

Maintaining of the mine roadways in stable operational state during the whole period of their exploitation is the most critical problems of the mining industry. Safety of miners, provision of working places with air and materials, productivity of the heading and mining machines, load on the longwalls, and, in whole, on the mine operation – all of these issues depend on the safety roadway supporting. With increased depth of the coal seam development, rock stresses reach their maximum values; besides, rheological effects occur, which all together lead to great loads on support causing its deformation and rock displacement towards the roadway [1].

The commonly used arch supports do not provide resistance sufficient for essential reduce of deformation processes in the rocks around the roadways. During the last 25 years, bearing capacity of the yielding arch support has been improved by 2-2.5 times mainly due to higher steel intensity. Despite of this fact, a significant number of mine roadways equipped with the arch supports are in emergency state and require repair (fig. 1) because even the arch supports with high steel intensity are characterized by low bearing capacity, which does not exceed 1.5 MPa.

Expenditures to the support setting in the roadways are not limited by the factor of the steel intensity. They are added by the costs of the support transportation and installation. Significant weight of the arch support elements and low level of mechanization of process of their installation contribute to the increased injury risk during these works. Besides, due to the arch deformation, cross section of the roadways decreases: consequently, the situation is complicated by the dust-and-gas mode, which worsens working conditions in the roadways [2]. In general, it can be stated that enormous financial inputs spent for securing the roadways by the arch supports do not ensure their keeping in a reliable operational condition and that traditional yielding arch supports are not able to fully compensate for the increasing displacements of the rocks into the roadways.

Fig. 1. Roadways with the frame supports

Instead, rock deformation can be controlled with the help of the roof bolts, which help to improve hardness and bearing capacity of the rocks. Roof bolting is widely used by many countries in their coal industries and other underground construction projects [3]. Until the 1970s, bolts with mechanical (locking) fasteners were mainly used. Development of polymer fasteners with fast setting and high strength allowed fastening the roof bolts along the entire length. Use of such roof bolts improved state of the roadways including those with weak and layered roofs. Under these conditions, mechanical roof bolts usually ruined, especially over time. In comparison with mechanical fasteners, roof bolts with polymer fasteners along the pole length (fig. 2) feature greater resistance to the lateral rock displacement. Besides, they are also less susceptible to corrosion, that significantly increases their effectiveness.

From the late ninetieth, Ukrainian miners began to use bolts with polymer fasteners. Scientific and technical aspects of these jobs were supervised by researchers of the Institute of Geotechnical Mechanics of National Academy of Science of Ukraine [4]. At that time, technology of the roof bolting was based on the then world practice.

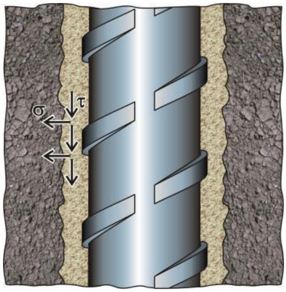


Fig. 2. Bolt with polymer fixing in the bore hole

Different normative documents specified conditions for using the roof bolts and technical requirements for the bolts and their elements, equipment and work technique. Basically, the roof support was used in the roadways with rectangular cross-section (fig. 3), and anchors were installed perpendicularly to the bedding, fig. 4.

Fig. 3. The roadways with rectangular cross-section

However, in the process of accumulation of practical experience on usage of the roof bolting in the Ukrainian mines, where new equipment was implemented and, therefore, loads on the longwall increased and geological conditions were worsen, it became clear the following. Bearing capacity of the most of the known schemes and designs of the roof bolting did not give a possibility to build roadways with the working life more than 4 years. As well, the existing techniques of the roof bolting made impossible to reuse the gate roads though it was very important in view of improving the coal produce costs.

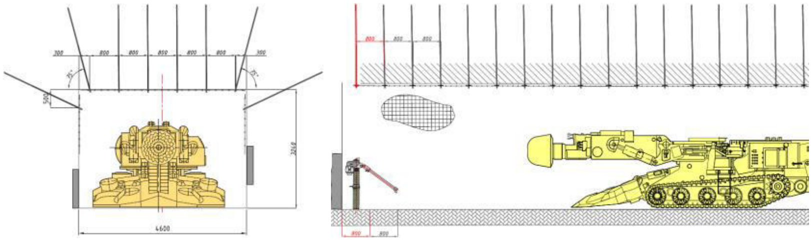


Fig. 4. Typical scheme of the roof bolting for roadways with rectangular cross-section

The reason was that geological and technical conditions in the most of Ukrainian mines were significantly more complicated than in the mines of other leading countries, which produced coal. Therefore, a simple transition of the accumulated foreign experience did not give any effective positive results. The situation demanded to study more deeply mechanism of the roof-bolt operation and to specify the space-time laws of geomechanical processes occurred in the system "edge rock – roof bolt".

2 Methods

In order to study mechanism of the roof bolt operation, we designed a mathematic elastoplastic model for calculating stress state of the rocks around the roadway supported with the bolts. The process of the rocks deformation is described by equations

$$c_g \frac{\partial u_i}{\partial t} = \sigma_{ij,j} + X_i(t),$$

where c_g - the damping coefficient, kg/(m^3·s); u_i - the displacements, m; t - time, s; $\sigma_{ij,j}$ - the derivatives of the stress tensor components along x, y, Pa/m; $X_i(t)$ - the projections of the external forces acting on the volume unit of a solid body, N/m^3.

The problem is solved in an elastic-plastic formulation. For the mathematical description of the process of rocks changeover into a disturbed state, the Mohr-Coulomb failure theory is applied. The initial and boundary conditions

$$\sigma_{yy}\big|_{t=0} = \gamma H; \quad \sigma_{xx}\big|_{t=0} = \lambda\gamma H; \quad u_x\big|_{t=0} = 0; \quad u_y\big|_{t=0} = 0;$$

$$u_x\big|_{\Omega_1} = 0; \quad u_y\big|_{\Omega_2} = 0,$$

where γ - the averaged weight of the overlying mine rocks, N/m^3; H - the mining depth, m; λ - the side thrust coefficient; Ω_1 - the vertical boundaries of the outer contour; Ω_2 - the horizontal boundaries of the outer contour.

This model took into account parameters of the roof bolts and roadway, strength and deformation properties of the rocks, as well as time and site when and where the roof bolt start working. In order to solve this problem, we applied a finite element method [5-7].

The roof bolt was simulated by the rod finite elements, washer and grab simulated by the triangle (prismatic) contact elements, and polymer fastener simulated by special contact elements. Each of the elements of the roof bolting featured certain physical and mechanical properties.

Level and character of the rock stress state change nearby with the roadway was estimated by the following parameters:

- reduced difference between the maximum and minimum components of the key stresses (Q=(σ_1-σ_3)/γH), this parameter characterizes difference between the stress-filed components and possible occurrence of the rock breaking;
- reduced minimal component of the principal stresses (P=σ_3/γH), this parameter specifies a possible type of the breaking.

3 Results and discussion

3.1 Simulation of the roof-bolt operation

In view of studying interaction between the roof bolts and rocks, we made a series of calculations.

The first step was simulation of the stress state in rocks around an unsupported roadway (fig. 5*a*). Result of the simulation is the basis for the further analysis.

In undisturbed massif, outside zone of mining operation influence, rocks are in the state of triaxial compression.

Vertical component of the stress field is determined by the weight of the overlying rock mass, and two horizontal components are determined by the lateral arching of the rocks.

a *b* *c*

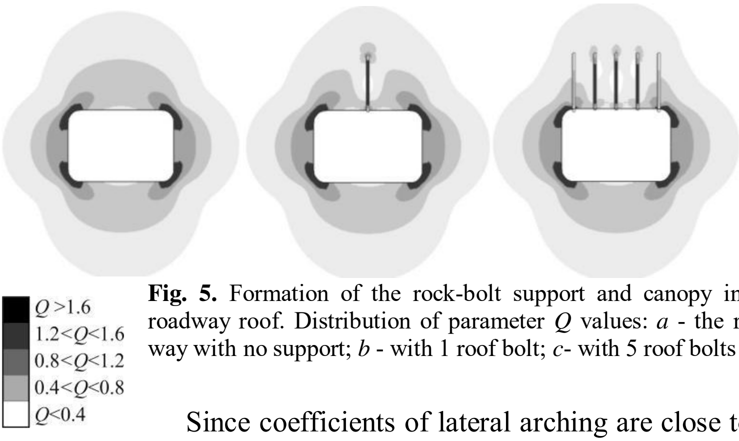


Fig. 5. Formation of the rock-bolt support and canopy in the roadway roof. Distribution of parameter Q values: *a* - the roadway with no support; *b* - with 1 roof bolt; *c*- with 5 roof bolts

Since coefficients of lateral arching are close to 1, state of the rocks in their natural occurrence is close to uniform compression. Difference between principal stress components in undisturbed massif is usually close to zero. That is why, regardless of the depth of the rock bedding, process of the rock breaking does not occur. This state of rocks is also observed in the massif with the roadway, but at some distance from it. At driving the roadway within zone of its influence, stress field in the rocks changes. Minimum component converges to zero, and maximum component increases and becomes greater than the maximum component in the undisturbed massif. In the edge zone, the unsupported rocks are deformed and their breaking is developed, fig. 5*a*.

When one bolt is installed, a certain volume of the roof rocks connected with the bolt is kept from displacing. With the course of time, rocks in zone around the bolt do not unloaded and are still in the compressed state. At a certain distance from the bolt, deformation processes are developed in the rock mass by the same pattern as in the roadway with no bolts. It is possible to specify this zone around the bolt as a support due to its function, fig. 5*b*.

A system consisting of 5 bolts (fig. 5*c*) prevents the edge rocks of the roof from shifting into the roadway and keeps the edge massif in a compressed state. As you can see, size of area with triaxially compressed rocks in the roadway roof is increased significantly (in comparison with the roadway with one bolt installed). Now it covers most part of the reinforced space.

3.2 New design of the roof bolting

Roof bolting is a space system of the roof bolts fastened in the bore holes. The key target of the installed bolt system is to arrange such layout of the bolts, which can ensure maximally effective interaction between supports created by the bolts. How to improve efficiency of interaction between the rock-bolt supports? To increase density of the bolt installation? But when parallel bolts are installed very close to each other it can trigger crack formation between them. Under certain geological conditions, when the bolts are installed vertically, a rock mass can be divided into blocks. And we know cases when such blocks have fell down into the roadways, fig. 6.

Fig. 6. Close setting of parallel roof bolts

The most promising method for improving interaction between the bolts is such their layout in the space, which would keep the rocks of the edge massif in a triaxially compressed state. As tests of the samples show, even a slightly increased lateral backup increases ultimate strength by 1.5-2.5 times.

We have analyzed results of our long-term observations and laboratory, computational and mine experiments conducted in more than 30 coal mines of Ukraine and established on their basis a concept of the rock-bolt support interaction, according to which supports unite into a single construction that can resist rock pressure (fig. 7). We also have solved a problem of how to improve efficiency of interaction between the rock-bolt supports [8, 9].

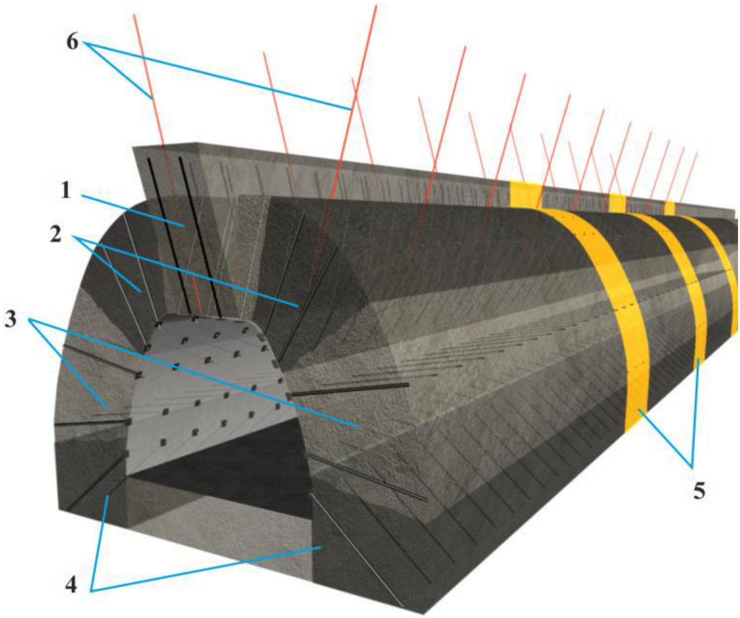


Fig. 7. Formation of the rock-bolt blocks in the structure of the rock-bolt canopy: 1 – is a load-bearing part of the canopy; 2 – is a backing-up part of the canopy; 3 – are supports of the canopy; 4 – is a base of the structure; 5 – is an element of the rock-bolt structure which we call a "bridge"; 6 – cable bolts

Functions of elements in the roof-bolting construction [10].

The load-bearing part of the canopy (fig. 7) counteracts deformations of the roof rocks and their displacement into the roadway. This element is the main and mandatory one. It is located in the central part of the roof along the entire roadway and is formed by the bolts in amount from 2 to 5 depending on the size of the roadway cross section and required capacity of the construction. This structural element can be reinforced by the deep rope bolts.

The backup part (fig. 7) takes the load from the load-bearing element of the canopy and transmit it to the support. This part is also needed for controlling life period of the construction depending on the roadway operation. This backup part consists of two elements located on both sides of the load-bearing part in the construction, not necessarily symmetrically about the axis of the roadway. Differences in the pattern of the backup part formation on each side of the roadway can be caused by:

- inclination of the coal seam and rock layers;
- cleavage and direction of natural fracturing;
- technological requirements for the winning operations (roadway to be closed or stored for reuse);
- coal-cutting with the goaf.

The backup part is a mandatory element of the roof-bolting construction and should consist of 1 to 4 steel-polymer bolts.

Supports of the canopy (fig. 7) transmit load from the canopy to the base. Load-bearing part of the construction is located in central part of the roadway walls, in the coal seam, in the rock above or below it. If the roadway walls are represented by the hard rocks, then in areas with partial supporting, this element is not mandatory. The load-bearing part is represented by two elements that are not always symmetrical about the axis of the roadway.

Quantity of bolts in the structural supports depends on the:

- roadway height;
- coal seam thickness;
- hardness of the coal seam and enclosing rocks;
- roadway function;
- roadway life.

Usually, number of bolts per one support does not exceed 4.

The base of the construction (fig. 7) facilitates increasing of stability of the roadway wall and decreasing of the floor heaving. It is located in the bottom of the roadway walls provided that hard rocks in the roadway walls are not obligatory element for the roof-bolting construction. The base consists of 1 or 2 steel-polymer bolts (for the cases when coal seam is located in the bottom of the roadway section, plastic or wooden bolts can be used).The floor bolts are inclined downward at an angle of 10-25° and can cross the floor line.

The element of the rock-bolt construction, which we call a "bridge" (fig. 7). On the scheme, it is marked by yellow. This element is a sector of the roadway with the higher stiffness thanks to the additionally set bolts. The "bridges" prevent the edge rock mass from breaking due to the released strain energy, which is accumulated with each elongation of the roadway. Staggering of such sectors makes possible to control the roadway stability, restore the needed safety factor and reduce expenses for the roadway exploitation.

The element “protective bridge” consists of several consecutive rows of the roof bolts with more powerful structure than the basic support.

Length of each of the protective jumpers and distance between them are determined by mining and geological conditions, function of the roadway and its service life. Usually, length of the bridge is 4-5 rows, and distance between them is 20 m. With the scheme with full roof bolting canopy, sectors are supported by the bolts in the following areas:

- at the beginning and at the end of roadway with roof bolting (transitional sections);
- with changed geometry of the roadway cross section;
- with geological disturbances;
- in conjugate areas with other roadways;
- with other roadway underworking or overworking.

Angle of the bolt inclination in the elements is determined by the type of the construction: simple, reinforced, powerful. For the simple construction, bolts are installed in the plane of the cross section perpendicular to the axis of the roadway.

For the reinforced one some bolts are installed with inclination towards the face at an angle of 70-75°. For the powerful construction, one part of the bolts is installed with inclination towards the face at an angle of 70-75° and another part is installed with inclination backwards, towards the roadway mouth, at an angle of 70-75°. Such bolt inclination significantly increase interaction between the bolts in comparison with traditional schemes of unidirectional reinforcement of the edge rock massif.

In order to test the roof-bolting constructions in the mine conditions, we developed “The Program and Methodology for Industrial Testing of the Roof-Bolt Constructions in the Main Workings at the Pavlogradugol Mine”.

As a result of the tests, operability of the roof bolt and its elements was validated, and indicators for efficiency and safety of its use were determined (fig. 8).

Fig. 8. Testing of the roof-bolt constructions in coal mains

3.3 The technology of the bearing-bolt supporting

Thus, the technology of the roof bolting was further developed [11], fig. 9, and range of its application was essentially expanded thanks to the following principles of the support setting:

- firstly, implementation of the system of bolts, which are set inclined along the roadway axis, and which improve interaction between the rows of the bolts and give a possibility to form the rock-bolt constructions;
- secondly, structural division of the of rock-bolt canopy into blocks (the bolts are installed in groups). It makes possible to maintain stability of the roadway as a whole system thanks to the safety displacement of the blocks relatively to each other;
- thirdly, arrangement, from time to time, of the areas with enhanced structural stiffness along the whole length of the roadway.

The technology of the bearing-bolt supporting is used for retaining the enclosing rocks in undisturbed and monolithic state, close to the state of a uniform compressed undisturbed rock massif. To this end,

- time period between extraction of coal-rock mass from the face and setting of the roof bolts and
- distance between the face and the first row of roof bolts should be reduced to a minimum.

In this case, edge rocks in the unsupported part of the roadway roof and walls have no time for getting unloaded from the rock pressure and their solidity will be fully preserved.

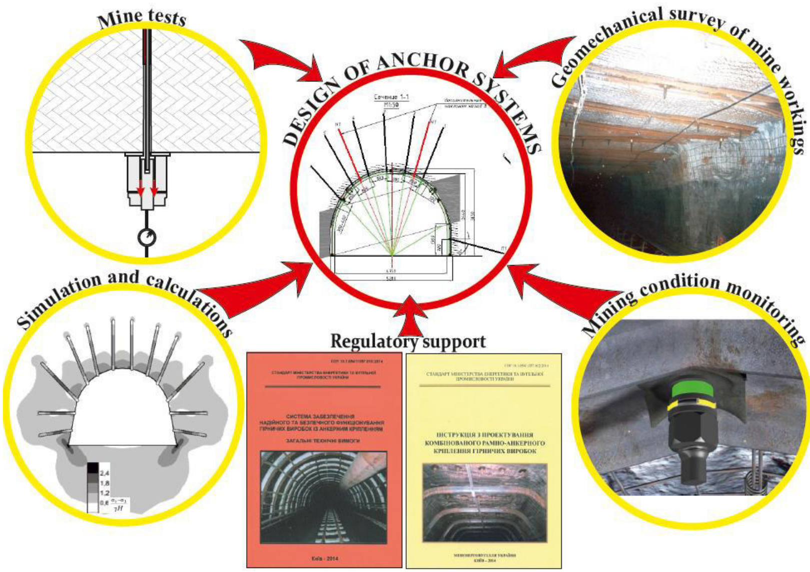


Fig. 9. Elements of the technology of the bearing-bolt supporting

Technological cycle of installation of the roof bolt construction consists of preparatory works and support setting for the roadway roof and walls. In order to reduce duration of this cycle, the following aspects were optimized:

- sequence of technological operations performed by the miners;
- the required amount of materials and equipment;
- layout of materials, equipment and tools at the workplace in order to minimize the length of route covered by the miner during the shift;
- devices for quick and accurate execution of technological operations (a template for observing the row spacing and inclination angles when drilling holes; a template for marking on the ground and saving the lower point of the drilling rig installation).

The support and scheme of the bolt setting are calculated basing on the requirement to exclude rock falling around the roadway. Bearing capability of the rock-bolt supports is determined depending on geological conditions of the mining operations, characteristics of the bolt strength and the bolt fastening in the bore hole. An optimal quantity of the bolts and scheme of the bolt setting are calculated in terms of the support resistance required for blocking displacement of the roadway contour.

One of the key requirements of the technology is to set the roof bolts into the enclosing rock mass, which is not unloaded from the rock pressure. In this case, the support, just after having been set, immediately starts working on resisting the forces of the rock pressure. The edge rock mass maximally saves its natural monolithic state. If not, the bolts will just "sew together" the stratificated roof rocks.

3.4 Duration of the roof bolt setting taken into account while determining technological parameters

As well, we developed a new method for calculating parameters of the stress-strain state of the rocks and construction of the rock-bolt bridge for technological cycles of setting the rows of the roof bolts. This method takes into account the rock unloading during the cycle and a moment of time when the row of the roof bolts start to operate [12, 13]. In order to study influence of the speed of the roadway driving and distance between the site of the bolt setting and working face on the roadway stability, we calculated several technological schemes (fig. 10, 11).

a *b* *c*

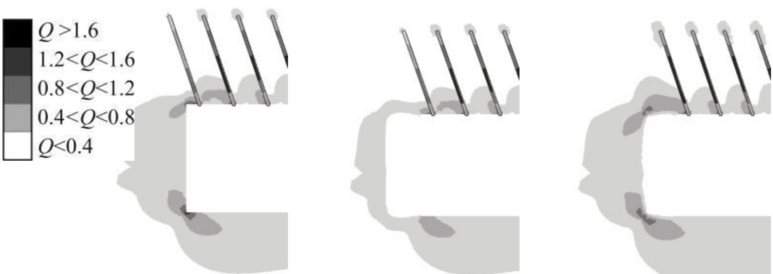


Fig. 10. The stress field (*Q*) changing during technological cycle of the roof bolt setting, distance between the site of the bolt setting and working face – 0.3 m: *a* - before the face driving; *b* - on the next iteration after the face driving; *c* - in 30 iterations

Speed of the roadway driving was changed in the range of from 5 to 40 m per day. Distance between the site of the bolt setting and working face was changed within the range of 0.3-5 m.

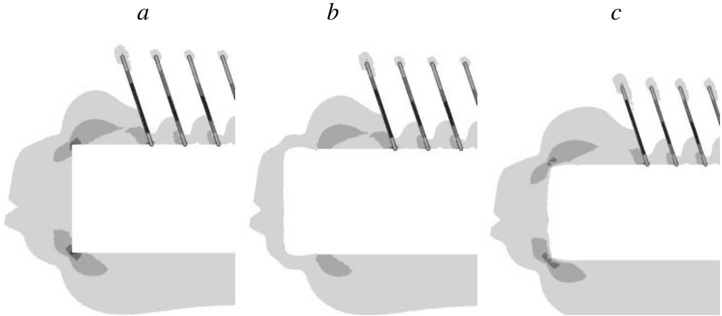


Fig. 11. The stress field (Q) changing during technological cycle of the roof bolt setting, distance between the site of the bolt setting and working face – 1.9 m: *a* - before the face driving; *b* - on the next iteration after the face driving; *c* - in 30 iterations

Figure 12 shows diagrams of the roof displacement when the bolts were set at distance 0.3, 1, 2 and 5 m from the face at speed of the roadway driving 10 meters per day. In the first variant, the roof bolt is set at distance of 0.3 m from the face. As the face is further driven the bolts are loaded and prevent the rock from displacing into the inside of the roadway. When the bolts are set at a longer distance from the face with the lag, the edge rock mass can easily displace during certain period of time and become unloaded.

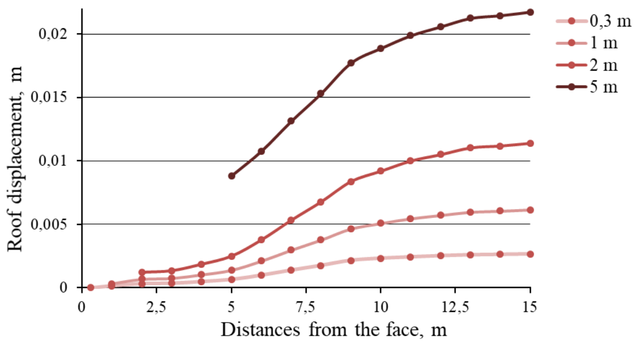


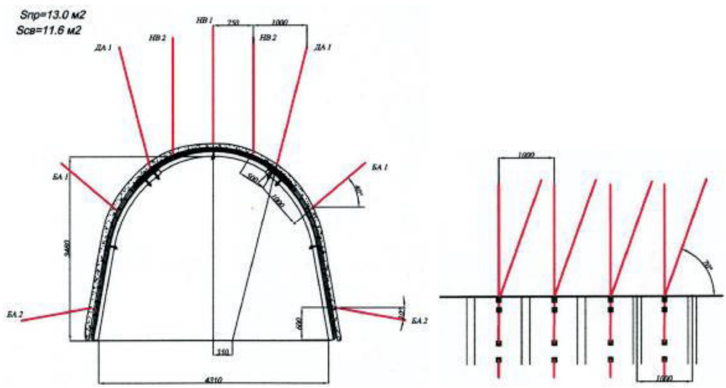
Fig. 12. Roof displacement with the bolts set at different distances from the face

Thereby, the bolts will be set into the disturbed rocks or, in case of the soft rocks with low tensile strength, into the broken rocks.

3.5 Reuse of the preparatory roadways

Special difficulties present maintenance of the roadways for their further reuse while mining an adjacent longwall [14, 15]. Application of bearing-bolt supports showed good results for these complicated conditions as well [16]. For example, the reinforced construction of the roof bolting with the powerful protective bridges was used in the mother entry 585 of the Yubileynaya Mine (fig. 13).

a



b

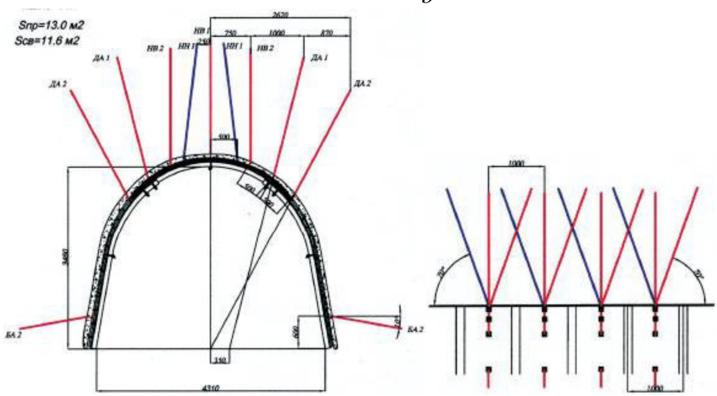


Fig. 13. Combined frame-bolt support with reinforced roof bolting in the roadway 585 of the Yubileynaya Mine: *a* - Simple sector; *b* - sector “bridge”

The roadway had an arch cross section. It was kept in operation after the first and the second longwalls had been mined out and later. Results of monitoring showed satisfactory stability of the roadway at all of the stages, fig. 14. The roof was not broken, it just sagged as a single block with turning, first, towards one side and then – towards another. Periodical reinforcement of the bridges helped to restore stability of the roadway and not to accumulate negative affect of the rock pressure with each elongation of the roadway.

a *b*

Fig. 14. Condition of the roadway 585 of the Yubileynaya Mine: *a* - 10 m to face end; *b* - 60 m after the face

3.6 Influence of the roof bolting on the rock permeability

One more advantage of the bearing-bolt supporting must be mentioned, which is also best for driving roadways through the gas-bearing coal seams and rocks and undermining of the water-flooded rocks. Results of solving the problems of elastoplastic deformations and fluid filtration were confirmed by data from the mines and have showed the following [17, 18].

In the roadway where roof is supported by the technology of the bearing-bolt supporting, an impermeable rock-bolt canopy is formed. This canopy prevents escaping of methane from the gas sources in the roof into atmosphere of the roadway. When density of the bolt setting is increased up to 1.1 bolt per meter, average speed of methane filtration and discharge decreases by 75 percent (fig. 15).

Application of roof bolting for the blocks in the roadway driven through the gas-bearing coal seam keeps the seam and enclosing rocks in stable state, and blocks processes of crack formation and coal pressing-out. Each of the bolts, set into the wall of the roadway, decreases filtration permeability by 20-30 percent. Such effect essentially decreases volume of methane escaping from the coal seam during mining and exploitation of the roadway. The bolts which are set in the bottom of the roadway walls prevent the walls from breaking and decrease zone with inelastic deformation in the roadway floor and, consequently, diminish floor heaving and filtration permeability.

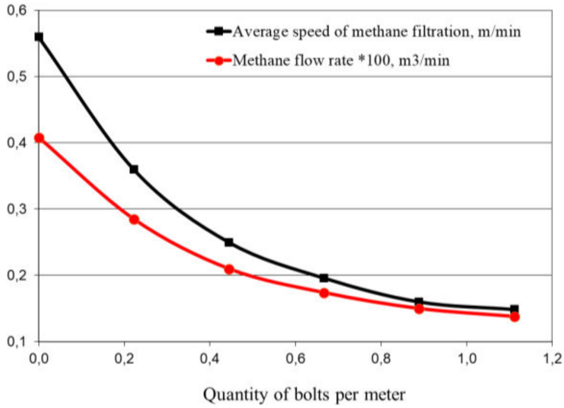


Fig. 15. Change of speed of methane filtration and consumption depending on density of the bolt setting in the roadway roof

Therefore, the roof bolting can be considered as a technological method for decreasing methane content in the roadways and water inflow during undermining of the water-flooded rocks [17-20].

4 Conclusions

Basing on the research findings, we have developed and approved a normative document, which regulates parameters for the technology of the bearing-bolt supporting. We also have developed an album with typical constructions, which regulates choice of the schemes for the roof bolt setting in typical mining, geological and technological conditions of the roadway drivage.

The technology of the reinforced and powerful bearing-bolt supporting designed by our Institute of Geotechnical Mechanics was successfully realized in practice in 52 Ukrainian mines for driving more than 700 preparatory roadways and permanent workings. This approach helped to achieve stable state of the roadways under the complicated mining and geological conditions and obtain good economic effect thanks to the cut expenses spent to the roadway supporting and repair.

However, the most weighty contribution of the roof bolting technology into cutting costs of coal production is, to our mind, higher rates and essentially better safety of the winning operations.

Thanks to the stable, almost unstressed state of the face end, necessity in setting additional supports falls down, and time period needed for the end operations during the face mining becomes essentially shorter. Besides, our technology makes it possible to use effectively any powerful mining machines and significantly speed up rate of the face mining.

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PROCESSING TECHNOLOGY FOR REFRACTORY GOLD-BEARING ORES

G.E. Askarova
teacher, Kazakh National University after Al -Farabi Kazakhstan

M.R. Shautenov,
Professor, Professor, Satbayev University, Kazakhstan

K.A. Nogaeva,
Professor, Kyrgyz state University, of geology, mining and development of natural resources named after academician Usengazy Asanaliev Kyrgyzstan

Abstract

Gold in nature is most often found in the form of native metal, intermetallic compounds and minerals containing gold, silver, copper, iron, mercury, bismuth, platinum, palladium, iridium, rhodium, and minerals - gold gellurides. In addition, gold is found in the form of compounds with organic acids.

Native gold is never chemically pure and contains up to 50% impurities. The presence of impurities in gold dramatically affects its quality and properties. For example, arsenic, lead, platinum, cadmium, bismuth, tellurium give gold brittleness, which leads to overgrinding and sludging of gold in the processes of ore preparation. In addition, native gold grains can be coated with films ("shirts") of iron oxides and rock minerals, which significantly complicates the extraction of gold during amalgamation and cyanidation.

Native gold has a high density - from 15.5 to 19.7 t/m^3. Gold minerals also have high density. Depending on the size, native gold is classified into coarse (more than 2 mm), fine (0.05-2 mm), pulverized (5–50 microns) and finely divided (less than 5 microns). The behavior of native gold grains and gold-bearing minerals in the enrichment processes depends on their composition and dissemination [1].

Gold closely associated with sulphides is well extracted using processes designed to extract sulphides (flotation, depositing, concentration on tables, etc.). Isolation of gold from sulfide products is usually carried out after the destruction of the sulfides themselves by calcination or biochemical method [1].

Schemes and processing regimes of ores substantially depend on the mineral composition of ores, their degradability, the presence or absence of impurities that complicate the extraction of gold, and also on the particle size of gold. These and some other properties of ores are mainly determined by their origin, according to which gold ore deposits are divided into two main groups - primary and placer. About 75% of gold is mined from primary deposits, and 25% from alluvial deposits. [1]

Introduction

A very low degree of development is characterized by the gold resource base of the republic. The volume of ore gold mining in Kazakhstan decreased by 2 times compared to 1989, and the degree of development of reserves fell to 0.6%. At the same time, the decline was due to a decrease in production from the actual gold deposits, in which 66% of all the country's proven gold reserves are located. Active development of this particular part of the gold resource base, and first of all the largest deposits of Vasilkovsky and Bakyrchiksky, can provide a significant increase in gold production in Kazakhstan.

The main problem of large-scale industrial development of these facilities is the difficulty of relatively relatively low-grade ores of gold (from 4 to 9 g/t), requiring the use of expensive efficient and environmentally friendly processing technologies.

The increase in consumption and the cost of precious metals forces gold mining companies to expand their raw materials base and study and develop processes aimed at increasing the extraction of gold from ores and concentrates.

The expansion of the production of precious metals largely depends on the involvement of “refractory” gold-bearing raw materials, including pyrite, arsenopyrite, pyrite-arsenopyrite ores and concentrates, in the sphere of industrial processing. In addition, mixtures of sulfides are often found: sphalerite, galena, chalcopyrite, etc. In all these concentrates, gold is present in various forms: native on the surface of minerals or in their lattice and in the form of intrinsic minerals (tellurides, aurostibnite). Differences in the ratio of these forms of gold, as well as the main minerals, determine a different approach to the processing of concentrates. The practical importance of raw materials of this type has been underestimated previously due to the complexity of the technological extraction of gold.

The persistence of sulfide ores and concentrates is due to micro- and submicroscopic fineness of gold, its thin impregnation with close association with pyrite and arsenopyrite, which does not allow to obtain high recovery rates when processed by standard methods:

- technological features of gold-containing raw materials determine the choice of technology for its processing;

- the main problem of processing refractory gold-bearing ores is the presence of harmful substances (antimony, arsenic, carbonaceous and clay components); free and associated associations of dispersed gold with metal sulfides and oxides, which complicate and impede the extraction of gold from ore;

- practically all known technologies in the head of the technological scheme use gravity methods of enrichment in order to separate free large, medium and small gold;
- methods and apparatuses known in foreign and domestic practice for gravitational ore dressing and removal of clay-sludge fine-grained particles are ineffective;

In recent years, the scope of work on the technological evaluation of gold-bearing ores, as well as on improving the technology for the extraction of precious metals in existing factories, has significantly increased. New methods of processing ores have appeared, requirements for the complexity of using mineral raw materials, as well as for the preservation of the environment and safety precautions, have increased.

Technological studies of gold-bearing ores mainly consist of analyzes and experiments necessary to determine the material composition of ores and the technology for extracting all industrially valuable components from ores. The ultimate goal of the study is to develop a technology for the maximum extraction from ores of all industrially valuable components with the greatest economic effect, subject to the requirements of safety and environmental protection.

In addition to ores, the objects of research are often products of their processing - amalgation and flotation tails, gravity and flotation concentrates, cinder, etc. [2]

The most typical studies are:

research of ores of new deposits at the stage of preliminary exploration;

research of ores of new deposits at the stage of their detailed exploration;

research of ores (or processing products) of exploited deposits.

The research of ores of exploited deposits is extremely diverse in content and volume. In the conditions of existing mines, technological studies are carried out in connection with a change in the nature of ores during mining from lower horizons or new sections of the deposit. Research is being carried out at enterprises and institutes in order to improve the ore processing at existing factories (reducing the loss of gold and other recoverable elements, improving the quality of concentrates, increasing the complexity of using raw materials, etc.). To do this, they are testing new enrichment and hydrometallurgical methods, more advanced processing schemes and modes, new reagents and apparatuses. Research is carried out both on a laboratory and semi-industrial and industrial scale. Despite the wide variety of technological research, almost all of them are carried out in the following sequence:

familiarity with the relevant literature and reporting data;

sampling at a field or factory;

preparation of samples for research;

study of the material composition of ores;

technological experiments in the laboratory;

verification and refinement of the results obtained in the laboratory at semi-industrial continuously operating plants or factories;

technical and economic assessment of the results of work and drawing up a report.

Gold ore in terms of material composition are very diverse. In some ores, more than 90% by weight is quartz; in others, along with quartz, the prevailing minerals are barite (up to 50-60%), carbonates (up to 20-30%), iron oxides (up to 25%), tourmaline (up to 50%)) The sulfide content (mainly pyrite, arsenopyrite and pyrotin) ranges from 0-80%. In various quantities, many other minerals are also present in ores, as well as host rocks (schists, granites, diorites, etc.). Ores vary in physical condition. Most of them, after mining, are represented by strong lumpy material, some have the appearance of a loose clay mass with separate pieces. Ores are even more distinguished by the properties of gold and its association with minerals.

When performing technological research, those signs of the material composition that determine the ore processing technology to the greatest extent are of primary interest. These signs are:

the presence in ores along with gold of other useful components having industrial grade;

the content of oxidized minerals in ores as compared to sulfide ones, i.e. the degree of oxidation of ores.

the presence in the ores of components that substantially complicate the processing technology;

the nature of gold in ores, primarily the size of gold particles.

The presence of other industrially valuable components in ores other than gold is one of the decisive factors in the choice of processing technology. The value of this factor is constantly increasing. This is facilitated, on the one hand, by an increase in the number of useful components with an increase in the depth of extraction of gold-bearing ores and, on the other hand, by the requirement for the integrated use of processed raw materials.

According to the degree of oxidation, the ores are divided into primary (sulfide), partially oxidized (mixed) and oxidized.

Currently, primary ores are of the greatest industrial importance; the sulfide content in them ranges from tenths to 80-90%. In some ores, oxidized minerals are also present, but in such a small amount that they practically do not affect the processing technology.

A characteristic feature of oxidized ores is the presence of iron oxides in them. A number of others contain oxidized minerals of other metals, as well as slimy (clay) components. Sulfides in ores are practically absent or in small quantities and do not affect the technology.

Methods

The properties of impurity gold differ from the properties of pure gold: impurities reduce the density of gold particles, change the structure, gold becomes less noble in chemical terms. Impurities of iron give gold magnetic properties. Differences in gold composition are noticeably manifested in flotation. Impurities reduce the flotation ability of gold, and the more, the easier they are oxidized. Often, gold particles have surface coatings consisting of oxides of iron and manganese, acanthite, covellite, galena, kaolinite and some other minerals. Coatings, besides natural origin, also appear as a result of mechanical grinding of the gold surface with solid particles during grinding. As a rule, gold with coatings floats worse than gold with a clean surface [2].

The forms of gold release are the most diverse: disseminated, vein-interspersed, vein, spongy, dendritic, scaly, lamellar-nodosum, in oxidized films, porous, magnetic, in intergrowths and others. Flaky and scaly particles float better.

Given the behavior in technological operations, gold particles size is divided into large (larger than 70 microns), small (smaller than 70, but larger than 1 mm) and finely dispersed (smaller than 1 mm).

It is advisable to isolate very large (larger than 0.5-0.6 mm), and in finely dispersed - colloidal or submicroscopic (finer than 0.1mm) [2].

The work was carried out using a complex of experimental and analytical research methods, including mathematical modeling methods using computers, mathematical statistics, physical modeling, experimental studies on various scale models and technological studies under production conditions. Physical, physicochemical methods were used in experimental studies Chemical and flotation methods: flotation 'laboratory and industrial tests on ores of various substance.

Experimental research

Ore gold has about 20 minerals. Of greatest industrial interest is native gold, represented by various metallic particles in size, composition, shape and structure in association with many minerals, most often with quartz, pyrite, arsenopyrite, barite.

Given the behavior in technological operations, gold particles are divided into large particles: large (larger than 70 μm), small (smaller than 70, but larger than 1 μm). Sometimes in large gold it is advisable to allocate very large, larger than 0.5-0.6 mm.

When grinding ores, large gold is freed from bonding with minerals (free gold), is easily captured by gravity enrichment, but does not float well and requires prolonged cyanization.

Coarse-grained gold-bearing minerals, as well as large and small native gold, are well recovered by gravitational processes, but are poorly floated and slowly cyanide.

Dusty and partially fine gold is poorly recovered by gravitational processes, but it floats well and cyanides well if it is not associated with tellurides.

Fine gold is poorly extracted not only by gravitational processes, but also by flotation, if it is not associated with carrier minerals. Such gold is quite satisfactorily extracted only as a result of hydromegallurgical processing.

Fine gold in the crushed ore is partially in a free state, partially in intergrowth with minerals (in aggregates). Free fine gold floats well, dissolves quickly during cyanidation, but is hardly enriched by gravitational methods.

Fine gold in intergrowths also successfully dissolves during cyanidation, and its flotation is determined by the flotation ability of the mineral associated with it.

During flotation with sulfhydryl collectors, gold floats well in intergrowths with sulfides, and in intergrowths with non-sulfide minerals - only at a certain ratio between the exposed surface of gold and the surface of the mineral associated with it.

Study of gold

The characteristics or properties of ore gold are established by technological tests of ore, mineralogical and phase analyzes, and special studies.

There is no standard technique suitable for all types of ores. Each researcher chooses it depending on the nature of the ore, the purpose of the study and the available equipment.

The most reliable results are obtained using a complex of methods. [3]

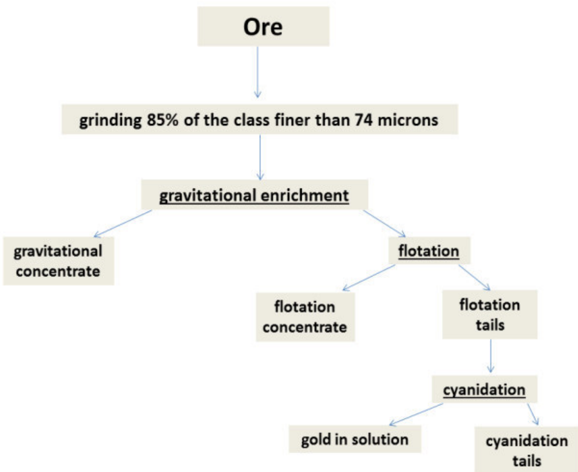

Fig. 1. Gold ore processing scheme

The definition of gold in the particle size classes of ore gives only some idea of the properties of gold. Ores with a grain size of 2 mm or smaller are divided into classes by dispersion and elutriation. It is advisable to classify ore of such a size at which gold will be extracted by the main method for this ore. Typically, this size is in the range of 90-95% -0.315 mm to 90-95% -0.071 mm. Ore weight is divided into classes, for example, -0.01; -0.02÷+0.01; -0.04÷+0.02; -0.071÷+0.04;

-0.1÷+0.071; -0.2÷+0 , 1 and +0.2 mm. Each class is weighed and analyzed for gold. If the material of any class is not enough for analysis, then take a second sample of ore and extract the desired class from it. According to the data obtained, it is possible to judge the uniformity (or unevenness) of the gold content in the classes and its division into classes. In large classes there may be free particles of large gold, as well as fine and finely divided gold associated with minerals. In this regard, it is impossible to draw a conclusion about the presence of large gold in ore based on only the gold content of large classes.

The idea of the properties of gold in particle size classes can be substantially supplemented by the establishment of correlation dependencies between the content of gold and other elements in the classes, as well as cyanidation of ore classes. In classes, in addition to gold, it is advisable to determine the content of silver, non-ferrous and rare metals, sulfide sulfur and tellurium. The use of cyanidation as a phase analysis operation to determine the properties of gold is described below.

The object of research is persistent refractory, clay ore weathering crust. The bulk of the free gold particles in this ore have a fineness of less than 20 microns.

The aim of the research is the selection and justification of the combined gravity-flotation method of enrichment of refractory gold-bearing raw materials and the hydro and pyrometallurgical method of processing enrichment products based on the study of the technological properties of the feedstock and enrichment products.

For research, two samples of a mixture of ores rich (5.65 g/t) and poor ores (1.2 g/t) were prepared.

Gravity dressing is designed to extract mainly large gold and, to a lesser extent, fine and gold-containing sulfides from gold-bearing ores. The most common industrial methods are depositing and beneficiation at locks; less commonly used enrichment in drum (pipe) concentrators, in heavy suspensions, on shaking (concentration) tables and in hydrocyclones. All of these methods, except enrichment in drum concentrators, can be tested in laboratory conditions.

Jigging is experienced in the study of almost all gold and complex gold-bearing ores.

Preparation of the machine for work begins with the installation of the specified parameters - the frequency and amplitude of oscillations of the diaphragm, the size of the mesh openings, the diameter of the fraction, the height of the layer of the fraction, the flow rate of under-water, productivity. When choosing parameters, it is necessary to take into account the following approximate dependencies between these parameters and deposition indices:

the concentrate yield increases with an increase in the amplitude of diaphragm oscillations, a decrease in the consumption of under-sieve water, and a decrease in machine load;

Gold recovery with increasing yield of concentrate increases but, as a rule, is not proportional. The indicated dependences are valid only under certain conditions, which can only be established experimentally.

Enrichment on a shaking table is used to clean deposit deposits, to separate free gold and gold-bearing heavy minerals. And also studies were conducted by Kazakhstan hydro-hub [3-5].

Gold flotation (7.5-8.5) must be created by adding soda to the mill (fig.1).

Gold and gold-containing sulfides should be floated with sulfhydryl collectors: xanthates (butyl, amyl, ethyl) and aeroflot. Xanthate consumption 100-150 g/t. Aeroflot is useful for controlling flotation at a flow rate of 40-50 g/t. Xanthates are introduced into the pulp in the form of 0.5-1% aqueous solutions.

If during the flotation of sludge ores a lot of gangue will be transferred to the concentrate, then the preliminary treatment of the pulp with liquid glass should be tested. The consumption of liquid glass is 0.5-1kg/ t, the processing time is 1-2 min.

It is advisable to test the developed flotation regime on water intended for use in an industrial environment. If the chemical composition of such water is known, then flotation can be carried out with distilled water after adding the appropriate substances to it.

To determine the influence of numerous factors on the flotation results according to the described methodology and to determine the optimal process mode, a significant number of flotation experiments are required. The situation is complicated by the fact that many factors are interrelated, i.e. with a change in one factor, the other also changes. Therefore, even after conducting a large number of experiments, there is no complete certainty that the developed flotation regime is indeed optimal. In this regard, it is advisable to use mathematical methods of planning experiments and processing the results obtained in studies, which will allow making objective conclusions with a minimum number of experiments.

The flotation concentration regime for ore was selected according to the method of the planned experiment (steep climb) with variables: grinding size, consumption of butyl xanthate and foaming agent T-66 and T-80, consumption of copper sulfate and soda. A fractional replica of the four and five-factor experiment was implemented. According to the experimental results, it was found that it is not possible to obtain flotation tailings of the initial ore with a gold content of less than 1.0 g/t with a fineness of ore grinding up to 80-85% of the class less than 0.074 mm. The output of flotation concentrate depends on the flow rate of the blowing agent and amounted to 8-12%. In an enlarged experiment, the following results were obtained when returning the intermediate product of cleaning the primary flotation concentrate and concentrate of the 11th flotation:

When consumed in the 1st flotation

T-80-50 g/t

Xanthate -175 g/t.

In the 11th flotation, T-80-50 g/t, xanthate -175 g/t and grinding size 80% in the class less than 0.074 mm, the yield of flotation concentrate was 11.75 with a gold content of 77.8 g/t and gold recovery in concentrate 90.7%. The gold content in the flotation tailings was 1.0 g/t.

Analysis of gold losses in the flotation tailings by size classes showed that more than 60% of all gold losses occur in classes larger than 40 microns. With the release of these classes of 43.81 g/t, the gold content in them was 1.45 g/t, and the loss with them of gold was 61.55%.

For ore, studies on flotation concentration were carried out on gravity tails with a gold content of 2.4 g/t cording to the method of the planned experiment (steep ascent), a fractional replica of a four-factor experiment with variables was implemented:

Grinding size in a class smaller than 0.074 mm (X_1);

Consumption of foaming agent T-80 (X_2);

Consumption of butyl xanthate (X_3);

Consumption of copper sulfate (X_4) (table 4-5).

Table 1

Matrix planning of experiments on the flotation

Indicators	Variable factor			
	size of grinding %, smaller than 0.074 CL.	(extraction) g/t	butyl xanthogenate g/t	copper sulphate g/t
Basic level	75	60	125	75
The range of variation in	10	20	25	75
Top level	85	80	150	150
Lower level	65	40	100	0
№ samples experiments				
159	-	-	-	-
160	+	-	-	-
161 167	-	+	-	-
162	+	+	+	-
163 168	-	-	+	+
164 169	+	-	+	+
165	-	+	-	+
166 170	+	+	+	+

Table 2

Conditions for conducting experiments

№ samples experiments	Size of grinding %, smaller than 0.074 CL.	Reagent consumption, g/t		
		T-80	butyl xanthogenate	copper sulphate
159	65	40	100	0
160	85	40	100	0
161	65	80	100	0
162	85	80	150	0
163	65	40	150	150
164	85	40	150	150
165	65	80	100	150
166	85	80	150	150
167	65	80	100	0
168	85	40	150	150
169	65	40	150	150
170	85	80	150	150

Table 3

Results of experiments performed on the experiment planning matrix

№ experiment numbers	Products	Output, %	Gold content, g/t	Weight of gold, g/t	Division, %
1	Concentrate	5.81	25.1	1.458	60.72
	Tails	94.19	1.0	0.942	39.24
	Source	100	2.4	2.4	100

Continuation of table.3

2	Concentrate	5.88	24.8	1.459	60.78
	Tails	94.12	1.0	0.941	39.22
	Source	100	2.4	2.4	100
3	Concentrate	6.81	22.91	1.56	65.0
	Tails	93.19	0.9	0.84	35.0.
	Source	100	2.4	2.4	100
4	Concentrate	8.5	16.4	1.3935	58.06
	Tails	91.5	1.1	1.0065	41.94
	Source	100	2.4	2.4	100
5	Concentrate	9.63	13.7	1.316	54.83
	Tails	90.37	1.2	1.084	45.17
	Source	100	2.4	2.4	100
6	Concentrate	8.79	20.0	1.762	73.42
	Tails	91.21	0.7	0.638	26.58
	Source	100	2.4	2.4	100
7	Concentrate	9.94	16.0	1.59	66.25
	Tails	90.06	0.9	0.81	33.75
	Source	100	2.4	2.4	100
8	Concentrate	9.27	17	1.58	65.8
	Tails	90.73	0.9	0.82	34.2
	Source	100	2.4	2.4	100
9	Concentrate	6.55	19.5	1.279	53.29
	Tails	93.45	1.2	1.121	46.71
	Source	100	2.4	2.4	100
10	Concentrate	8.4	11.1	0.934	38.92
	Tails	91.6	1.6	1.466	61.08
	Source	100	2.4	2.4	100
11	Concentrate	8.76	17.0	1.488	61.98
	Tails	91.24	1.0	0.912	38.02
	Source	100	2.4	2.4	100
12	Concentrate	9.59	11.8	1.59	66.1
	Tails	90.41	0.9	0.81	33.9
	Source	100	2.4	2.4	100

Results and discussion

The reproducibility of experiments is estimated by variance

$$\sigma_x^2 = \frac{\sum_{i=1}^{n}(X_i - X^-)}{N-1} \quad (1)$$

Where X← is the average value of the optimization parameter; X_i is the value of the optimization parameter of an individual repetition; N is the number of experiments,

Table 4

Variance of experiments (has been compiled to calculate the variance)

№ samples	E_1	E_2	E_1+E_2/2	σ_x^2
159	60.76	-	60.76	-
160	60.78	-	60.78	-
161 167	65.0	53.29	59.15	68.44
162	58.06	-	58.056	-
163 168	54.83	38.92	46.88	126.405
164 169	73.42	61.98	67.7	65.37
165	66.25	-	66.25	-
166 170	65.8	66.1	65.95	0.045
			Σ=481.53	Σ=260.26

Combined processing technology of the studied ore. In order to more fully evaluate the use of the cyanide process of the ore under study, in addition to the research plan, enlarged laboratory experiments were conducted to cyanide the initial ore, gravity tails and flotation tails.[4-9]

The experiments included cyanidation in the presence of sorbents: AM-2B resin during cyanidation of ore or gravity tails and flotation tails and activated carbon during cyanidation of gravity and flotation concentrates and their mixtures.

Table 5

The effect of the duration of cyanidation on the extraction of gold from their tailings flotation of ore 3

Conditions and Results	№ of experiments					
	39	40	41	42	43	44
Cyanization duration, hour	2	4	6	8	12	18
Pulp density. % tv	40	40	40	40	40	40
The concentration of cyanide,%						
Initial	0.045	0.045	0.045	0.045	0.045	0.045
the ultimate	0.035	0.035	0.035	0.035	0.035	0.035
Cyanide consumption. kg/t	0.15	0.15	0.15	0.15	0.15	0.15
Lime consumption, kg/t	1.0	1.0	1.0	1.0	1.0	1.0
Sorbent, g		1.0	1.0	1.0	1.0	1.0
Gold Content, g/t						
in the original product	0.8	0.8	0.8	0.8	0.8	0.8
in sorption tails	0.7	0.6	0.5	0.45	0.45	0.45
ld recovery,%	12.5	25.0	37.5	43.75	43.75	43.75

Conclusion

Based on the results of the studies, the following options for ore processing schemes are recommended:

Isolation of large free gold and sulfides in the grinding cycle-sorption cyanidation of the tails of gravity. Technological mode:

- the size of ore grinding 80-85% of the class is smaller than 0.074 mm;

- Duration of cyanide 4-6 hours;

- Duration of sorption cyanide -15 hours;

The consolidated (balance) experiment according to the scheme in Figure 1 was performed under the following conditions: – the size of the grinding of gravity tails in the 1st flotation-65 % CL. smaller than 0.074 mm;

- fineness of the tailings of the 2^{nd} flotation 77.9% CL. finer than 0.050 mm;

- the mode of the 1st flotation: - duration of flotation - 5 min;

- consumption of T-80-60 g/t;

- consumption of xanthogenate - 75 g/t;

- the mode of the 2^{nd} flotation: - duration of flotation - 10 min;

- consumption T-80-20 g/t;

- consumption of xanthogenate – 25 g/t;

- 3rd flotation mode: - duration of flotation - 15 min;

- consumption T-80 - 20 g/t;

- consumption of xanthogenate – 50 g/t.

The research was carried out within the framework of the Project GF MES RK «Research and instrumentation of the processing technology of refractory arsenic-sulfide gold-bearing ore of the Kazakhstan deposit» №0210PK00912

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MARKETING INNOVATION IN THE MANUFACTURING OF STONE-GRINDING PRODUCTS

Kovshun N.E.
National University of Water and Environmental Engineering (NUWEE), Doctor of Economic Sciences, Professor, Department of Enterprise Economy, Ukraine

Savina N.B.
National University of Water And Environmental Engineering (NUWEE), Doctor of Economics, Professor, Department of Labor Resources And Entrepreneurship, Ukraine

Malanchuk L.O.
National University of Water And Environmental Engineering (NUWEE), Candidate of Sciences (Economics), Assistant Professor, Department of Public Administration, Documentation and Information, Ukraine

Radko A. O.
National University of Water And Environmental Engineering (NUWEE), Candidate of Sciences (Economics), Ukraine

Abstract

The introduction of innovative technologies in the extraction of basalt helps to increase the efficiency of the mining enterprise. Volyn Basalt has its own features in terms of blockage and fracture. Studies of basaltic outcrops in the conditions of the Rafalovsky and Berestovetsky quarries showed their high fissure, clearly expressed separately, limited by vertical and horizontal fissures. Such characteristics affect the technology of stone extraction and processing. The physical, mechanical and chemical properties of the basalts of Private Joint Stock Company (PJSC) "Rafalovsky Quarry" allow it to be used for the organization of production of types of products. It is advisable to organize the production of stone, which will allow you to make a cloth of super-thin basalt fiber. The economic substantiation of the effectiveness of such an innovation was carried out on the example of the enterprise of PJSC "Rafalovsky Quarry" on the basis of traditional methods of project analysis. In order to ensure the successful functioning of the enterprise in the production of new products, it is necessary to create a marketing service, as an important component of the structure of the enterprise.

Introduction

The use of innovative products and technologies has a significant impact on improving the production and economic activity of the enterprise, increasing its competitiveness and increasing profits and the rational organization of the innovation process is the key to maintaining a stable market position and further functioning and development. Development, production and sale of innovative products in the production of stone-crushed stone products require the introduction of specific marketing and management measures.

One of the innovative areas for Rivne mining companies is the production of basalt wool. It is by far the highest quality and most demanded thermal insulation product. Advantages of mineral wool insulation made on the basis of basalt fiber, developers explain their high thermal and sound insulation performance, as well as flammability [1].

The production of such products in an individual enterprise is economically feasible, but requires improvement of organizational activity. An important aspect is to take into account the features of the basalt rocks extracted by the enterprise.

Volyn basalts attract researchers with their unique mineralogical and chemical composition.

Their isotopic age according to the potassium-argon method is 510-598 million years.

They are represented by two varieties. Aphonite basalts are black and dark gray aphonite rocks. Basically, it is plugonite basalt. Its mineral composition: plagioclase - 36%, pyroxene - 33%, glass - 19%, palagonite - 6%, mineral - 6%. These basalts are exposed in the quarries of the villages of Berestovtsy, Yanova Dolina, Ivanchi, Polyssia and others. Almond-shaped basalts are a greenish-gray fine-grained rock with a large number of tonsils up to 15 mm in size. Mineral composition: plagioclase, ore mineral (magnetite, ilmenite), apatite, volcanic glass. The main eruptions are in the Styr river basin. The density of the rock and the chemical composition of the basic basalts are presented in table. 1 (weight percentage).

Table 1

Chemical composition of basic basalts by quarries

The compositions of chemical elements,%	Detection Places	
	Hodosy, Gutvin, Yanova Dolina, Midsk	Berestovtsi, Yanova Dolina (quarry №2)
SiO_2	45,04	49,5
TiO_2	2,54	2,85
Al_2O_3	14,3	12,79
Fe_2O_3	6,03	3,36
FeO	6,46	10,63
MnO	0,4	0,21
MgO	8,47	6,19
CaO	6,58	9,38
Na_2O	2,42	2,78
K_2O	0,48	2,05
P_2O_5	0,17	0,57
SO_3	0,3	0,2
CuO	0,03-1,2	0,22
H_2O	0,72	0,8
Інші	1,88	2,41

To destroy the ore body and the subsequent extraction of minerals, it is necessary to expend a significant amount of energy. The development of new technologies for processing and preparing the rock mass involves the use of the properties of rocks inherent in nature to reduce energy consumption for their destruction, to increase the disclosure of ores and the recoverability of minerals. Therefore, it is relevant to study the natural separateness of various rocks in the basalt massif, which can significantly change the approaches and principles of technological impact on them for the subsequent destruction and extraction of [2] minerals. According to many researchers, the destruction of the rock mass separately occurs most pronounced in centrifugal crushers and mills [3]. For example, centrifugal crushers are used to produce crushed stone of a cubic shape at relatively low energy costs. The destruction of the ore body rock containing metal nuggets occurs without grinding metal inclusions. The destruction in this case, obviously, occurs along the so-called reticular planes, characterized by the number of nodes (atoms, ions) of the planar lattice per unit of its plane (in accordance with the Bravais hypothesis). Obviously, this rule applies not only to crystals, but also to rocks in general.

Based on the positions of blocking and fracturing, the study of basalt exposures under the conditions of the Rafalovsky and Berestovets quarries showed their high fracture (bottom height 15–20 m), pronounced individuality limited by vertical and horizontal cracks. At present, basalt is mined for crushed stone. The technological scheme of production includes overburden operations (the thickness of the sand-chalk layer of the overburden is 2-5 m), drilling of wells in accordance with the passport of drilling and blasting operations, blasting the face, followed by excavator loading in automobile vehicles for delivery to the crushing and screening site. Commercial products in the form of crushed stone in three classes of fineness (-10; 10-20; 20-40 mm) are obtained using gear, cone, centrifugal crushers and vibrating screens.

The economic substantiation of the basalt cotton wool production project in this study was conducted in conventional units in order to show its effectiveness and to reflect the traditional for Ukraine ratio of different types of costs on the example of one of the mining enterprises of Ukraine, located in Rivne region - PJSC "Rafalivsky Quarry".

Trends in modern construction are a clear testament to the growing focus on thermal insulation and energy conservation. In many countries, in addition to effective insulating materials made of polystyrene, polyurethane foam, glass staple fiber, insulation materials based on mineral wool made of [4] basalt stone have been used for more than 30 years. However, it should be noted that the main problem currently facing many developers is the acute shortage of this material, which is increasing over time.

Basalt fiber insulation materials are delivered to the market by a number of domestic manufacturers (there are 7 of them, total design capacity - up to 1 million cubic meters of conventional cotton wool per year). Thus, the Irpine mill «Progress» was the first in Ukraine (since 1969) to start producing ultra-thin fibers on the basis of basalt rocks. Similar products in Ukraine are produced by Belitsky Plant «Thermal Sound Insulation», Kyiv Industrial Complex «Budindustriya», Chernivtsi Plant of Thermal Insulation Materials «Rotis», Zhytomyr Mineral wool Factory.

The strong position of mineral wool from basalt is based on its excellent properties [5]:

• good insulation ability over a wide temperature range, even at high temperatures (the value of the thermal conductivity coefficient is in the range 0.032-0.038 W / mK);

• fire safety - basalt wool does not burn (it effectively prevents the spread of flames and can be used as fire insulation and fire protection);

• good sound insulation (basalt wool is widely used in floor, wall, interior partitions to reduce noise, the best sound insulation is achieved by applying an additional air gap between the insulation and the outer treatment layer);

• good mechanical and chemical resistance (basalt cotton wool products do not shrink and are not susceptible to temperature deformation; no cracks are formed at the junctions between the slabs and joints between the plates, which could cause heat leakage and become the center of moisture condensation) , (has high resistance to chemicals, neither solvents, alkaline, nor acidic media have any effect on it);

• compressive strength;

• a wide range of products;

• does not absorb water (basalt cotton wool is non-hygroscopic, moisture content under normal operating conditions is less than 0.5% by volume);

• safety and environmental friendliness (basalt wool is safe during installation and operation).

Therefore, the physical, mechanical and chemical properties of the basalts of PJSC "Rafalovsky Quarry" allow it to be used for the organization of new industries and types of products. First of all, it is an organization of stonecutting, which will make it possible to produce BSTV (basalt superfine).

The technological module for manufacturing basalt-fiber thermal insulation materials is shown in Fig. 1.

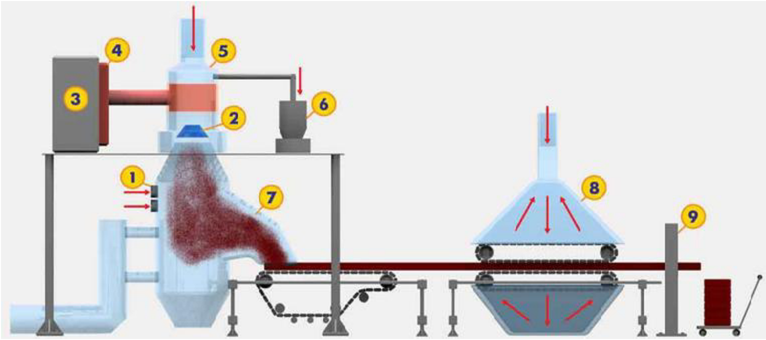

Fig. 1. Technological module for the production of basalt-fiber thermal insulation materials: 1- sprays; 2- blowing head; 3- generator; 4- block of loading circuit; 5- induction furnace; 6-dispenser; 7- fiber-deposition chamber; 8- drying chamber; 9- cutter

The raw material for the production of thermal insulation materials is a basalt crumb with a fraction size of 3-15 mm.

Through the feeder-dispenser crumb enters the melting furnace at t = 1400-1500 ° C is melting.

The temperature in the oven is controlled by the amount of gas supplied and the air.

The air is fed through a metal recuperator, where it is heated by the use of flue gas.

The obtained melt through the feeder filters under the action of its own weight, is extracted in the form of continuous filament, the diameter of which is in the range of 100-250 μm [6].

Production time of the UB STB 42 installation is 1-1.5 months, and installation, commissioning - 0.5-1.0 months.

The composition of the installation and the technical and economic indicators of UW STV 42 are shown in table. 2 and Table. 3 respectively.

Table 2

Composition of the UB STV 42

№	Denomination	Number
1	Oven	4
2	Gas supply system	4
3	Recuperator	4

Continuation of table.2

4	Loader dispenser	4
5	TEC transformer	4
6	Feeder feeders with suspension mechanism	4
7	The hood mechanism	2
8	Inflatable camera	2
9	System of gas-air supply of inflatable cameras	2
10	Diffuser	1
11	Fiber Deposition Camera	1
12	Combustion products cleaning chamber	1
13	Ventilation of removal of combustion products	1
14	Electrical equipment	1
15	Ground	1
16	Water cooling system	1

If necessary, additional sectional ovens can be ordered and installed to increase production capacity. When 2 additional furnaces are installed, the production capacity increases 2.5 times. The BSTV production plant is serviced by one operator per shift.

The mode of operation of the furnace is round-the-clock, the standard of use - 350 days a year. In each shift, except for the operators, there should be: the debugger of the technological equipment, the operator on control and measuring devices and equipment (KVPiA) and the locksmith mechanic.

Accordingly, the need for staff is 15 people (Table 4).

Table 3

Technical and Economic Indicators of UB STV 42 Installation

№	Indicator	Units of measurement	Value
1	Productivity: - hour - production capacity at a coefficient of use of CFRC 0.9	 кg/h t	 30,4 250
2	Gas consumption max per 1 kg BSTV	m^3/кg	3,2
3	Electricity consumption per 1 kg BSTV	kilowatt /кg	2,0
4	Density	кg/m^3	30
5	Installation dimensions: - length - latitude - height	 m	 8,2 5,4 4,4

Table 4

Staffing requirements

№	Position	Number, people	Annual Fund of Remuneration, $
1	Operator	3	54000
2	Debugger	3	43200
3	Mechanic locksmith	3	37800
4	The composer	6	97200
	Total	15	232200

The cost of the equipment is 205 thousand USD, the training of 3 operators and the 1st debugger is 9.6 thousand USD, and the cost of installation, commissioning works is 21.6 thousand USD. It is envisaged to purchase the installation, staff training and preparatory work at their own expense (Table 5). It is envisaged to finance the purchase of additional equipment to provide the shop with compressed air at the expense of a bank loan, using the services of PJSC "State Export-Import Bank of Ukraine".

When forecasting the volume of BSTV output, design capacity is taken into account when the CFRF utilization coefficient is 0.9.

It is planned to receive annually from the production of BSTV 3750 thousand USD. 6).

Table 5

Assignment of investment funds

№	Assignment of investment funds	Sum, thousand USD	Source of funding
1	Purchase of installation	205	Own funds
2	Purchase of equipment to provide the shop with compressed air	105	Credit facilities
3	Training of 3 operators and 1 debugger	9,6	Own funds
4	Installation, commissioning	21,6	Own funds
Total:		341,2	

Data on the total volume of production of BSTV were used to draw up a plan of income and expenses for the implementation of UB STB 42 PJSC "Rafalovsky Quarry" for the relevant years of its operation. Based on the technical and economic characteristics of the UB STV 42 installation, we will calculate the cost of natural gas and electricity. It is known that natural gas consumption is 3.2 m^3/kg and electricity is 2 kW/kg.

Natural gas consumption per year is:

$$250{,}000 \text{ kg } 3.2 \text{ m}^3 = 800 \text{ thousand m}^3$$

The price of gas per 1000 m^3 for industrial enterprises as of May 1, 2010 is 2637,78 USD (without VAT).

The annual cost of natural gas is:

$$800\ 2637.78 = 2110224\ \$.$$

Electricity consumption per year is:

$$250{,}000 \text{ kg } 2 \text{ kW} = 500 \text{ thousand kW}$$

$$500\ 000\ 0.65 = 325\ 000 \text{ UAH}$$

Based on the above data, we calculate performance indicators for the implementation of the installation [7] (Table 7).

The calculated data shows that the project is acceptable. Because the yield index is larger than one unit and the payback period is 7.8 months.

Determination of break-even point of sales and financial strength are given in Table 8.

Table 6

Plan of income and expenses of the implementation of the UHF STB 42 installation of PJSC "Rafaliv Quarry" in thousand USD

№	Indicator	Year				Total
		base	first	second	third	
		0	1	2	3	
1	Revenue from the sale of products	-	3750	3750	3750	11250
2	Costs of everything, including	-	2854,5	2854,5	2854,5	8563,49
3	material costs	-	2435,22	2435,22	2435,22	7305,67
4	wages of workers	-	232,2	232,2	232,2	696,6
5	payroll	-	87,1	87,1	87,1	261,3
6	Selling expenses	-	100	100	100	300
7	Profit from sales of products	-	895,5	895,5	895,5	2686,5
8	Total investment, including: -own funds -credit resources	341,2 236,2 105	0	0	0	341,2 236,2 105
9	Annual credit payment	-	35	35	35	105
10	Interest on the loan	-	22,05	14,7	7,35	44,1
11	Credit balance at year-end	-	70	35	0	105
12	Loan Amounts Paid	-	57,05	49,7	42,35	149,1
13	Profit before tax	-	838,451	845,801	853,15	2537,40
14	Income tax (25%)	-	209,61	211,45	213,29	634,35

Continuation of table.6

15	Net profit	-	628,84	634,35	639,86	1903,05
16	Amount of depreciation	230	15,33	15,33	15,33	45,99
17	The total income of the enterprise	-	644,17	649,68	655,20	1949,05

Table 7

Calculation of economic efficiency indicators for the implementation of the UWB STB 42

№	Indicator	Year				Total
		base	first	second	third	
		0	1	2	3	
1	Capital investments, thousand USD	341,2	-	-	-	341,2
2	The total income of the enterprise, thousand USD	-	644,17	649,68	655,20	1949,05
3	The discount rate (20%)	1	0,83	0,69	0,58	-
4	Discounted income thousand USD	0	524,03	440,52	370,29	1334,84
5	Yield Index	1334.84 341,2				3,9
6	Payback period, months	341,2 524,03				7,8

Table 8

Calculation of break-even volume of production and stock of financial strength

№	Indicator	Units measurement	Forecast year
1	Production volume	m^3	8333,3
2	Price	dollars	450
3	Sales revenue (excluding VAT)	thousands of dollars	3750
4	Total variable costs	thousands of dollars	2435,22
5	Variable cost per 1 m^3	dollars	292,23
6	Fixed costs	thousands of dollars	434,6
7	Break-even production volume	thousands of dollars	1239,57
8	Stock of Financial Sustainability (SFS)	thousands of dollars	2510,43
9	Stock of financial stability	%	67

According to the calculation method, the break-even point will be calculated as

$$T_b^{natur} = \frac{UPV_t}{P_0 - UZV_0}$$

$$T_b^v = T_b^{natur} \cdot P_0.$$

$$T_b^{natur} = \frac{434600}{450 - \frac{2435220}{8333{,}3}} = 2754{,}6 \quad m^3$$

$$T_b^v = 2754{,}6 \cdot 450 = 1239570 \quad dollars.$$

Therefore, the estimated break-even point of the rendered services of PJSC «Rafalovsky Quarry» is. This means that the break-even points of PJSC «Rafalovsky Quarry» can reach the first four months of the plant's operation.

We present a graphically defined break-even point and a margin of financial strength (Fig. 2).

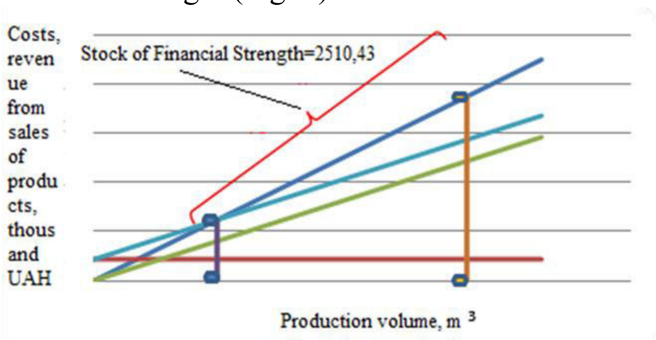

Fig. 2. Determination of critical sales volume of basalt superfine and financial strength

As the above chart shows, in order to cover all costs, the company must sell products worth 1239.57 thousand USD. (TB). According to estimates, sales revenue will exceed TB by 67% ($ 2510.43 thousand), which will mean a fairly stable financial position in the future.

In the current fierce competition and increasing influence of changes occurring in the external environment, the company should focus not only on the internal state of affairs, but also take into account and identify in their activities possible changes in the environment [8].

Reliable, complete information about the state of the external environment of the enterprise is considered, evaluated and analyzed in such a functional element as marketing. Marketing as a process involves the analysis of market opportunities; selection of target markets; development of marketing complex and implementation of marketing activities.

Studying the possible behavior of the closest competitors is indispensable. If the company does not pay attention to its competitors, it enters the competition blindly. Analyzing the organizational structure of PJSC «Rafalovsky Quarry», it should be noted that the enterprise does not have a marketing department and people who would be engaged in marketing research and development of the marketing complex of price, commodity, marketing and communication policies. Therefore, the lack of a marketing service makes the enterprise unstable and vulnerable to changes in the external environment.

Considering the fact that in the production of new types of products the enterprise should focus on the promotion of goods to the market, namely: to determine the best way to achieve a strategic goal for each division of the enterprise, to formulate an effective commodity, price, marketing and communication policy. Therefore, the creation of a marketing service at PJSC «Rafalovsky Quarry» is an extremely urgent task and an important component of the enterprise structure to ensure the successful operation of the enterprise in the future.

Marketing managers don't always aim to drive sales. Their goal may be to support existing sales while reducing costs for advertising and promoting products, or even reducing demand. In other words, the marketing service must support demand at the level set in the senior management's strategic plans. Marketing helps the company evaluate the potential of each business unit of the company, set goals for each of them and then successfully achieve them.

Therefore, it is advisable to investigate the dependence of sales revenue on sales costs as components of marketing costs, where x is sales costs and y is sales revenue. To investigate separately the impact of this category on sales revenue, you need to examine the relationship between them based on metrics. Namely: the correlation coefficient, the parameters of the regression equation.

Using Excel, we derive the relationship of the relationship between the studied indicators, as well as the correlation coefficient and parameters of the regression equation.

The regression equation looks like:

$$y=20.66x - 1029$$

Therefore, the relationship between sales costs and sales revenue is close and straightforward, which means that by increasing costs by $ 1, sales revenue will increase by $ 20.66.

Therefore, it is advisable for the company to spend on marketing. Calculate the coefficient of elasticity

$$E = a_1 \frac{\bar{x}}{\bar{y}} = 20{,}66 \frac{535{,}66}{10037{,}62} = 1{,}1$$

According to the calculated coefficient of elasticity, the revenue will increase by 10% while the increase in sales costs by 1%.

The cost of organizing the marketing department at PJSC «Rafalovsky Quarry» will be $ 140,000 per year.

Advertising costs will be $ 12800, which includes the creation of your own website on the Internet, advertising in periodicals and more.

It is planned to introduce a position of marketer in the field of innovation. The main task of which is the search, research, analysis and implementation of new products at the enterprise, as well as benchmarking. Benchmarking is a comparison of your metrics and processes with those of other companies, most notably market leaders. The focus of benchmarking is the question: Why are others doing better than we are? The main purpose and purpose of benchmarking is to identify differences with a comparable analogue (benchmark), identify the causes of these differences, and identify opportunities to improve the objects of benchmarking. Objects of benchmarking can be: methods, processes, technologies, quality parameters of production, indicators of financial and economic activity of enterprises (structural units). When researching production processes, methods or technologies of production and marketing, the focus is on finding reserves to reduce production costs and increase product competitiveness through the introduction of [9] innovation.

It should be noted that the study of the innovative component of the macro environment allows the company not to miss the "technological leap" and maintain its competitiveness at the appropriate level. Therefore, the efficiency of the enterprise depends on the innovation factors.

Thus, the analysis of innovations, the study of "external innovations" allows to notice in a timely manner the opportunities which science and technology opens for production of products, improvement and modification of technology of production and marketing of products [8]. This category includes expenditures on science and technology (state, investors), patent-license protection of technologies, innovative processes in the field of enterprise functioning. PJSC «Rafalovsky Quarry» is difficult to keep track of new directions of technology development, since their development is beyond its scope of activity, so there is a real threat of delayed implementation and, consequently, loss of market share. There is the task of constantly monitoring changes in the design of new technologies, which will be fixed precisely by the marketer in the innovation sphere. One important approach to addressing this is technology transfer. It is that laboratory developments are brought to the market where they can be purchased.

In the end, the use of modern technology and technology will enable the company to reduce its costs per unit of output, produce better quality and upgraded products, directly increase production, which will allow the company to generate more revenue.

Due to the organization of the marketing service, it is planned to increase the production and sales of stone-crushed stone products in the next year, and thus to generate 10% more revenue than in the previous year, according to the calculated coefficient of elasticity, as well as to obtain a net profit of 788.4 thousand USD in the next reporting period (Table 9).

Table 9

Comparative characteristics of indicators before and after organization of marketing department at PJSC «Rafalovsky Quarry»

№	Indicators, thousand USD	To	After
		organization of marketing service	
1	Volume of sales	12312	13543,2
2	Production costs	12352	12492
3	Balance sheet profit	(40)	1051,2
4	Income tax	-	262,8
5	Net profit	-	788,4

The company must have people capable of conducting market analysis, planning marketing activities, their implementation and control. The practical use of theoretical material and the creation and implementation of a marketing service at the enterprise will have a positive effect, increase the level of profit and will help to expand its market share.

Therefore, it is recommended to improve the organizational structure and attract qualified marketing specialists (including benchmarking) to form a market research unit.

Conclusions

The implementation of the UB STV 42 installation will provide production of BSTV for the amount of 3750 thousand USD. Labor productivity will grow from 112.55 thousand USD/person up to 122,61 thousand USD/person (108.9%). Net adjusted income for 3 years will be 1334,84 thousand USD.

Introduction of marketing service will increase the sales volume of products by 10% in the next reporting period, which will amount to 13543,2 thousand USD. The total expenses of the enterprise will increase by 140 thousand USD, correspondingly the net profit will be 788,4 thousand USD. The profitability of sales will increase in the next period and will make 5,8%.

Therefore, thanks to the introduction of innovations and their marketing support, the company will expand its product range and become more flexible to changes in the environment.

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PROSPECTION GEOPHYSIQUE POUR LA RECHERCHE DES EAUX SOUTERRAINES A SILIDARA (CONCESSION DE CBG)

Kourouma Mory[1], Keita Daouda[1], Viktorov Gleb[2]
ISMGB, Guinée
Toumnyne, Guinée

Résumé: la Compagnie des Bauxites de Guinée, dans le cadre d'approvisionnement de ses sites miniers à SILIDARA à Sangaredi (Concession CBG) en eau potable, a réalisé des forages d'hydraulique. La société de prospection TOUMNYNE S.A.R.L. a été chargée d'exécuter ce projet. Cette société a effectué, à cet effet, la prospection géophysique des eaux souterraines avant l'implantation des points d'eau sur ces sites. La méthode géophysique utilisée par TOUMNYNE S.A.R.L. est la méthode magnétique avec filtrage des résultats. La prospection magnétique a été faite suivant sept (07) profils à l'aide du magnétomètre à protons de marque MMP-203MS (production de la Russie).

Le traitement des résultats ont permis de mettre en évidence trois zones de fissures accompagnés de broyage (03 zones de broyage) disposées de façon linéaire. La valeur maximale des anomalies magnétiques a été fixée à 32 *nT*.

Mots clés : sites miniers, SILIDARA, prospection magnétique, anomalies magnétiques, fissures, zones de broyage.

Abstract: the Bauxite Company of Guinea, in the procurement framework of its mining sites in SILIDARA Sangaredi (CBG Concession) in drinking water, has made hydraulic drilling. The exploration company TOUMNYNE S.A.R.L. was responsible for executing the project. This company had, for this purpose, the geophysical groundwater before implantation of water points on these sites. The geophysical method used by TOUMNYNE s.a.r.l is the magnetic method with filtering results. The magnetic survey was made following seven (07) profiles using the branded proton magnetometer MMP- 203MS (Russian production).

The treatment results helped to highlight three areas of cracks accompanied by grinding (milling zones 03) linearly arranged. The maximum value of the magnetic anomalies was set at 32nT.

Key words: mining sites, SILIDARA, magnetic prospecting, magnetic anomalies, fault, broyage areas.

I. Introduction

Les travaux de prospection magnétique ont été effectués dans le cadre de solution du problème de ravitaillement en eau potable de sites miniers au NORD de la région de Boké.

Avant la prospection magnétique, on disposait des données suivantes :

La coupe géologique générale de la zone d'étude est caractérisée par une couverture sédimentaire récente d'épaisseur prédominante de 5 - 10 mètres, composée de latérites et d'argiles.

Par l'intermédiaire de 3 à 5 mètres de roche mère altérée, elle repose soit sur la roche cristalline saine représentée dans cette région essentiellement par la dolérite du Mésozoïque, soit par des aleurolites/argilites dévoniennes.

Les dolérites comme les aleurolites sont affectées de plusieurs failles, fractures, zones broyées et affaiblies.

Concernant les conditions hydrogéologiques, on peut dire avec une grande probabilité, que le seul réservoir capable d'encaisser des quantités d'eau considérables est celui des fissures, des fractures avec zones broyées etc. dans les aleurolites et dolérites.

La recherche des sources des eaux sera donc orientée vers des fractures majeures dans les roches mères à prédominance doléritique.

La tache essentielle qu'on ait posée devant la prospection magnétique est la recherche de failles et de zones de broyage comme des structures perspectives pour implanter des forages d'eaux.

II Matertiels et methodes

2.1. Equipement

Pour réaliser les travaux de prospection magnétique sur le terrain on a organisé une équipe de travail contenant un opérateur, un aide opérateur et un secrétaire. Cette équipe a utilisé le magnétomètre moderne à proton de marque MMP-203MS (production de la Russie) qui donne la possibilité de mesurer le module du vecteur total du champ magnétique terrestre dans le diapason entre 20 000 et 100 000nT. La valeur minimale mesurée égale à 1 nT.

2.2. Maille d'observation

On a choisi, d'après les objectifs des travaux, la maille d'observation de 10×10 m : 10 m est la distance entre les profils et 10 m est la distance entre les points d'observation. La maille 10×10 m à été implantée par la station totale TRIMBLE M3.

2.3. Contrôle des mesures

Pour effectuer le contrôle de mesure du champ magnétique on a répété des observations aux certaines nombre des points sur les profils d'observation. Les mesures de contrôle nous ont permis de calculer l'erreur moyenne d'une lecture isolée d'après la formule suivante

$$\varepsilon = \sqrt{\frac{\sum_{i=1}^{N}(X_i' - X_i)^2}{2N}}$$

X_i' - lecture ordinaire;

X_i - lecture de contrôle ;

N - nombre des points réitérés.

III. Resultats et discutions

3.1. Etude de variations du champ magnétique terrestre

Pour étudier le caractère des variations du champ magnétique terrestre on a choisi une station de mesure des variations diurnes du champ magnétique sur laquelle on a mesuré le champ magnétique toutes les 5 minutes à partir de 10h et jusqu'à fin de travail. Le caractère de variations du champ magnétique terrestre est présenté sur la fig. 1. On peut voir que d'une part, le champ magnétique terrestre monte du matin vers le midi, et puis redescend dans l'après midi ; et d'autre part, les variations diurnes du champ magnétique portent le caractère monotone linéaire. L'amplitude maximale des variations diurnes ne dépasse pas 35 nT. Pour éliminer l'influence des variations diurnes du champ magnétique il a été enregistré au niveau de chaque piquet le temps de mesure ce qui a permis d'introduire les corrections nécessaires.

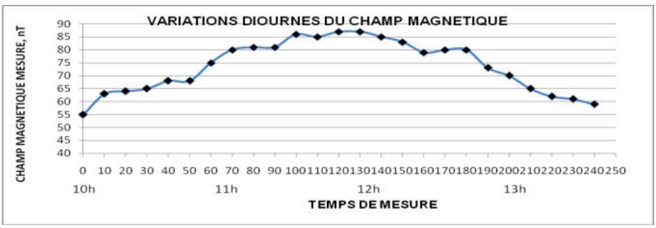

Fig. 1. Variations diurnes du champ magnétique des sites

3.2. Résultats de mesures du champ magnétique

La valeur moyenne générale du champ magnétique mesuré sur les sites est égale à 32101 *nT*, ce qui est très proche du champ magnétique normal de la Terre dans cette région (33654 *nT*), calculé d'après la formule suivante

$$T = \frac{\mu_0}{4\pi} \frac{M}{R^3} \sqrt{1 + 3\sin^2 \varphi_m}$$

R – rayon de la Terre (R=6371 km) ;

M - moment magnétique de la Terre ($8{,}3 \cdot 10^{22}$ A m^2) ;

μ_0- constante magnétique ($4\,\pi \cdot 10^{-7}$ m^2 kg/sec^2 A^2) ;

φ_m - latitude du site de travail (dans notre cas φ_m= 10,5°)

Sur la fig. 2 sont représentés les graphiques du champ magnétiques corrigés sur les variations diurnes du champ magnétique terrestre et le champ magnétique normal.

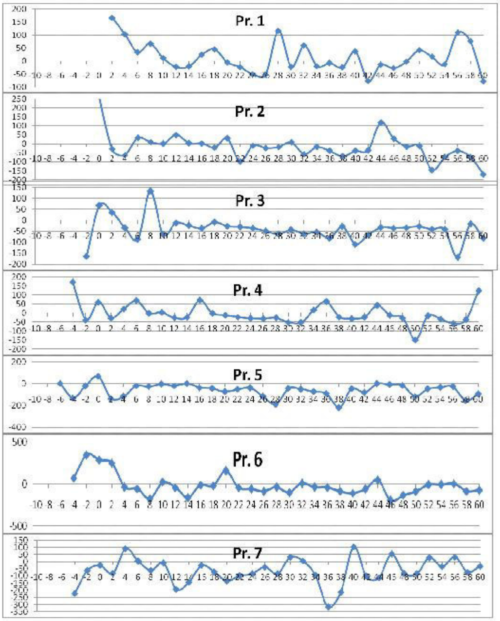

Fig. 2. Traitement primaire de la prospection magnétique effectuée (Correction sur les variations diurnes et le champ magnétique moyenne). L'axe horizontal – les piquets et l'axe vertical – champ corrigé en *nT*

3.3. Traitement géologique des résultats de la prospection magnétique

Pour trouver le caractère des anomalies magnétiques provoquées par des zones de broyage dans les dolérites nous avons construis le modèle géomagnétique (Fig. 3) et calculé l'anomalie magnétique d'après la formule ci-dessous:

$$T = 2J{\cdot}2b{\cdot}(h \cos i{+}x \sin i)/(h^2{+}x^2)$$

ici T = module du vecteur totale du champ magnétique normale de la Terre (0,25 Oersted);

J = vecteur d'aimantation (300 10-6×0,25 Oersted);

$2b$ = épaisseur horizontale de la zone de broyage (20 m);

H = épaisseur de la couverture (5 m);

I = angle d'inclinaison du vecteur T (11deg)

X = axe des abscisses;

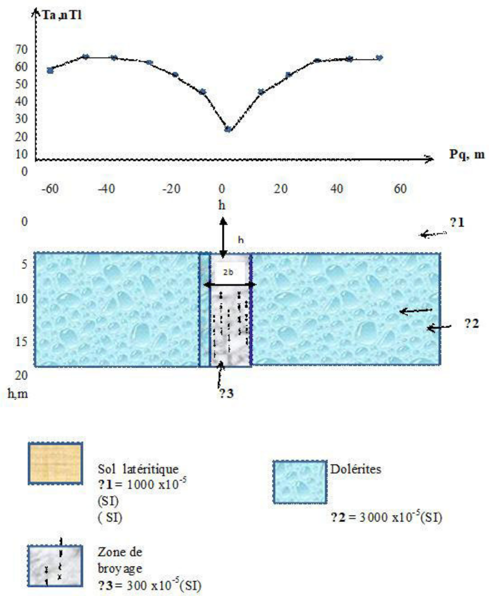

Fig.3. Anomalie magnétique provoquée par les zones de broyage

A partir de cette condition, on a trouvé que la valeur maximale d'anomalie à l'épicentre de la zone de broyage est égale à $6{\times}10^{-4}$ Oersted où 60 nT.

Donc, la méthode de la prospection magnétique de haute précision, réalisée par le magnétomètre à proton, donne la possibilité à détecter les zones de broyage dans les coupes géologiques, qui sont favorables au point de vue alimentation en eau potable des villages.

Carte des profils de la prospection magnétiques

Filtre 20×20 entre les profils et 30 m le longue des profils.

Le caractère des variations chaotiques du champ magnétique mesuré sur les profils (voir fig. 2), ne donne pas de possibilité effectuer correctement leur l'interprétation géologique puisque les anomalies d'origine géologique sont masquées par les variations apériodiques du champ magnétique d'origine industrielle.

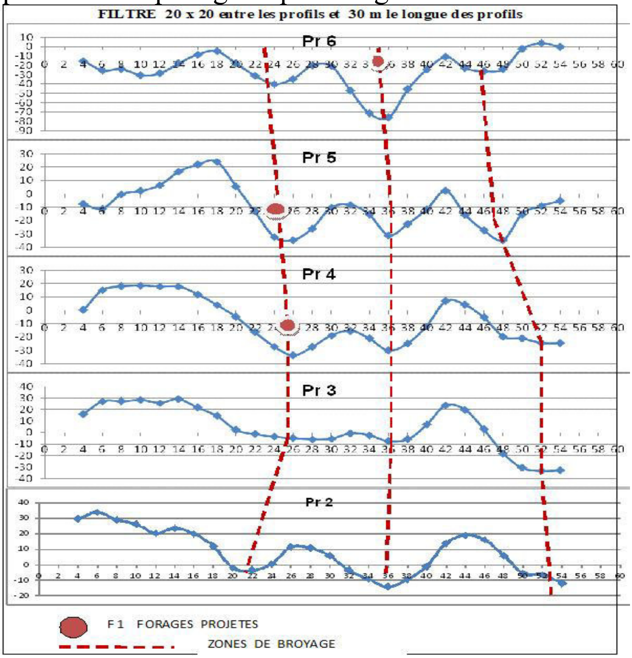

Fig.4. Traitement géologique de résultats de la prospection magnétique sur les sites

Pour diminuer l'influence des variations technogènes et démasquer les anomalies d'origine géologique, nous avons utilisé la filtration du champ magnétique mesuré par le filtre linéaire de 30 m de longue des profils et la fenêtre 20×20 m entre les profils. Résultats de filtration sont représentés sur la fig.4.

Cette méthode de filtration a donné la possibilité découvrir les anomalies géologiques sur les profils d'une part et effectuer leur corrélation entre les profils d'autre part.

Le caractère linéaire des anomalies donne la possibilité de les identifier comme des anomales provoqués par des zones de broyage et zones de fissures.

Sur la carte des profils (fig. 4) nous avons présenté les sites de trois forages hydrauliques implantés dans les zones de broyage.

IV. Conclusion

La méthode magnétique, avec l'emploi d'un magnétomètre moderne à protons MMP-203MS, a été utilisée par TOUMNYNE sur les sites miniers de GBG de façon professionnelle.

Cette méthode a permis de détecter des fissures et des zones broyées sur les sites miniers qui constituent des structures favorables à l'accumulation des eaux souterraines et par conséquent des points favorables pour l'implantation des sondages d'adduction d'eau potable.

Il y a lieu de signaler dans cette zone minière, les variations apériodiques du champ magnétique d'origine industrielle qui provoquent variations chaotiques des valeurs sur les profils. C'est ainsi que la filtration des données recueillies sur le terrain a été utilisée pour éliminer ce phénomène en obtenant que des variations d'origine géologique pour le calcul des anomalies magnétiques.

Pour les travaux futurs dans cette région, nous recommandons l'utilisation de deux (02) stations de prospection magnétique, une fixe et l'autre mobile, pour éliminer les variations d'origine industrielle en évitant la filtration.

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ABOUT A QUESTION OF A DECREASE OF A ROCK PRESSURE AT AN ORE DRAWING FROM THE BROUGHT DOWN BLOCKS

Shepel O.L

PhD (Engineering), associated professor of the Department of Underground Mining of Mineral Deposits, Kryvyi Rih National University, 11 Matusevych St., 50027 Kryvyi Rih, Ukraine

Rymarchuk B.I.

DSc (Engineering), Senior Scientific Associate (Engineering), professor of the Department of Underground Mining of Mineral Deposits, Kryvyi Rih National University, 11 Matusevych St., 50027 Kryvyi Rih, Ukraine

Purpose. The analysis of researches efficiency of an ore drawing, as basic process of underground mining operations at mining methods of a sublevel caving of ore and adjacent strata which provide improvement of parameters of an ore drawing is the purpose of the given operation.

Methodology. The detailed analysis and researches of domestic experience of release of the brought down ore from panels and the blocks, which actual for present time; analytical researches of process of an ore drawing on all plane of the block at horizontal contact between ore and leaning soils and absence of a strong rock pressure on development workings; ore drawing experimental researches in laboratory conditions on models as much as possible approached to real conditions; program methods for processing of results of experiments.

Object of research. Law of change of pressure of broken-down rocks on the block bottom is established at mining of thick steeply dipping deposits of the big prodeleting.

Research conclusion. The effective way and a preparation order, a location and section of developments are offered. On the basis of the executed laboratory researches of an ore drawing in the separate block located the long side an across the strike of a deposit, the size and character of distribution of pressure of broken-down rocks on the bottom basically depends on the sizes of the block along the strike. With increase in the size of the block along the strike increases not only absolute size of pressure, but also non-uniformity of its distribution.

Introduction. Thanks to practical data of mining of ore deposits, it is known, that an ore is drawing – one of the important operations of mining methods floor and a sublevel caving on which depend, both qualitative, and quantitative results of extraction. Therefore, improvement of technologies of working of deposits which provide improvement of parameters of an ore drawing is an actual scientific and technical problem.

Analysis of recent studies and publications. Deep analysis of a problem shows, that earlier on the bottom of blocks at release of broken ore scientists were not engaged in questions of a decrease of a rock pressure. But it is established, that the lowest losses and dilution will turn out at horizontal contact of a surface of the brought down ore to lowered baring's, the minimum distance between exits of cone raises on an undercut level and a uniform ore drawing from all hatches. However practically to create such conditions for release it is possible seldom, owing to difficulty of their simultaneous performance and consequently the question usually should be solved compromise by [1,2].

Release on all area of the block at horizontal contact between ore and settling soils and absence of a strong rock pressure on development workings below an undercut level usually happens, let us assume, under following conditions: 1) at the small area of deposits developed by one block; 2) at an extraction of blocks cut in a cranch or ore and soils (in such conditions half of blocks in chessboard order their extractions is developed, and also a part of blocks and at their consecutive extraction; 3) in some cases at an extraction of blocks only one lateral face, adjoining a goaf, in particular it can be admissible at moderate power of a deposit with the vertical or very abrupt pitch angle, lying down in more or less steady adjoining rocks and developed blocks the in width on all power; 4) at an extraction of blocks and the several sides adjoining a goaf, but at small their horizontal sizes.

Usually on development workings simultaneous release is made for pressure decrease on a part of the area of the block at inclined contact between ore and soils.

The size of a corner of an inclination of contact changes in limits from 30 to 70 degrees, but on the average prevails 45-60 degrees [3].

With increase in this corner of an inclination pressure upon development workings goes down, but simultaneously with it losses and ore dilution increase. The ore drawing is made in regular intervals from all working cone raises and whenever possible in small amounts that the contact surface between ore and soil remained more or less equal.

The maximum area of release is defined by a horizontal projection of a surface of contact between ore and soils at its distribution to an undercut level.

The analysis of experiences shows, that the increase in a corner of an inclination of a surface of contact (especially more than 50 degrees) negatively influences release indicators: the volume of the pure ore extracted prior to the beginning of dilution, decreases, and the volume of added barren rocks increases [4-8].

From methods of mining with a roof fall of system floor and a sublevel caving give the greatest losses and ore dilution. The least losses and dilution happen at a cutting of blocks in a cranch thanks to release of the most part of ore without contact with soils and their small size, and at its termination owing to horizontal contact at it of ore to soils.

The lowest qualitative and quantitative results of extraction are given by an extraction of blocks several lateral faces adjoining a goaf, on contact with which regular losses and dilution, from the beginning and till the end of release having the much bigger value, rather than taking place on contact to soils covering from above turn out.

Application of ways of preparation of blocks and mining methods basically depends from natural (power, a pitch angle, an ore hardness and adjacent strata, technical (the applied equipment) and technological (parameters of blocks) factors [9].

Cost of carrying out preparatory and access developments makes a considerable part of the cost price on mining method (from 33,0 % to 56,2 %). In spite of the fact that from preparatory and access roads 5-10 % of pure ore are extracted only, labour input of development workings occupies one of the main places, in general more than labour expenses on sublevel caving mining method.

The accepted way and preparation order, location and section of developments should correspond to following requirements: 1) to answer modern lines of development of technology of conducting mining operations; 2) to provide timely preparation of levels and blocks for preservation of a constant reserve of the reserves of ore prepared and ready to an extraction; 3) to promote a rock pressure decrease on developments within panel; 4) to lead to reduction of expenses on fastening and its repair; 5) to provide the maximum concentration of mining operations and intensive working of reserves of the block; 6) to raise productivity of stope; 7) to reduce labour input on drivages; 8) to provide on system as a whole the minimum cost price 1 t the extracted ore; 9) to plan possibility of application of the high-efficiency chisel, loading and transport equipment; 10) to reduce ore losses in earth entrails; 11) to meet the requirements of sanitary-and-hygienic conditions, safety of conducting preparatory and coal-face works.

One of conformity variants to these requirements are without the whole scheme of preparation of the blocks, which widespread.

The most important advantages of this way of preparation of blocks: the location under floor orts under an ore strata, and air orts in the adjacent block provides their stability at working of reserves of ore from a lying side to the trailing; expenses for a retimbering under level roads are reduced; good ventilation of stopes for all period of working of a sublevel remains; possibility of a selective mining of ore; sufficient heat-sink capacity of the ore chutes falling to each stope at working of reserves of the block; possibility for working of thick deposits in the conditions of a high rock pressure is created at an intensive actual mining of panels and secondary use of preparatory orts for massif drilling out by deep wells; necessity under floor over a pillar ort thanks to what losses are reduced is eliminated and quality of the extracted ore raises [10].

Lacks: complexity of conducting coal-face works on different horizons at increase in number of sublevels; necessity of strict observance of sequence of working of panels for blocks; increase in length of preparatory and access developments owing to carrying out additional under floor orts and increase at 5 m of length of drifts of scraping [11].

So, optimum parameters of the block at sublevel caving system are such which provide the greatest possible extraction of pure ore prior to the beginning of dilution, the minimum expenses for developments and the minimum production cost 1 t ores. Optimum parameters of the block: a height of sublevel - 40 m, length of the block - 60 m [12, 13].

Statement of the problem. Thus, it is necessary to develop effective technology of working of deposits which is characterized by the big indicators of an ore mining. In turn the big indicators of an ore mining probably to receive with application of thick high-efficiency loading technics.

Statement of the material and the results. Pressure at an ore drawing from the brought down blocks located on depth of some hundreds of meters and more, considering from a surface, establishes the specific problems before analysis of dynamics of this process. At an ore drawing from the brought down blocks under leaning soils in the conditions of the raised rock pressure the expiration of a loose material from a vertical cylindrical vessel (release from bunkers) proceeds much more difficult, than. In the latter case problems of dynamics of release are reduced to definition of pressure of a stream of a loose material at its movement on the bottom and walls of a continuous part. A pressure source here is the stream of a moving loose material. In the conditions of underground ore mines constructive elements of the block and the brought down ore even before its release are under the static pressure which size in tens, time exceeds the pressure rendered by streams of brought down ore at its movement to final apertures.

Release of the brought down ore from the block or the panel causes redistribution of this pressure and can be in communication by this important action for a decrease of its size in a zone of coal-face works.

The main constructive element of caving systems subject to pressure, the bottom of the block and its wall in a zone of abutment vertical pressure is.

Horizontal pressure under influence, the so-called broken expansion, caused by formation of a crest and of interest at the expiration of loose bodies from silage towers, at release of the brought down ore recedes on a background. First, because calculation of walls of the block on durability is not necessary, and, secondly, because its size is insignificant in comparison with a vertical abutment pressure. As an ore drawing conduct, as a rule, from a series of final apertures of the block with formation of the general zone of breakage within all fulfilled area, sealing of ore under the influence of horizontal expansion at release from any aperture remains at inclusion in operation of the next apertures adjoining to it.

Besides this phenomenon has no serious value for practice as lateral dilution at an ore drawing from the apertures adjoining on a goaf, is observed always in spite of the fact that sealing of dead rocks under the influence of horizontal tensions should take place.

It is all it is necessary to consider if to let out ore from one final aperture in delivery developments. But the situation can change, if release to make zones. Therefore in this case horizontal expansion will influence sealing of barren rocks and a decrease of lateral dilution that it will be possible to establish in the long term in laboratory conditions.

It is accepted to understand set of such practical actions for artificial regulation of size of character of increase and distribution of pressure to associates as pressure control under level roads which allow to provide stability of soils in one developments and on the contrary, uniform development of their caving in others. At an underground extraction of ore deposits this complex of actions consists in a choice of the rational sizes for the given concrete conditions and forms of developments, pillars and application of certain sequence of conducting coal-face works within the block and all deposit [14-16].

Transition of mining separate, located in a massif, blocks to wide application of systems of a floor caving, thus mass application of system has faced the whole complex of the phenomena which had no place at an extraction of the isolated blocks. The strengthened display of pressure concerns their number in slusher level developments, their complete destruction on the considerable areas, and on occasion – destruction of an ore in place of the bottom and developments in it before the termination of a block undercutting.

On fig. 1 the model of release of the brought down ore from final apertures near which strain gauges for definition of value of loading on each strain gauge settled down is presented. The model is presented executed to scale M 1:100 with a forward glass wall. The distance between final apertures - 5 m, diameter of final apertures – 2 m. As a loose material is used martite ore from granules by metric structure +1÷-5 mm. Height of the brought down layer in the panel – 40 m. Depending on a location of strain gauges on the panel bottom accordingly change values of loading at an ore drawing.

From received a drawing (fig. 1) it is visible considerable nonuniformity in pressure distribution. The profile maximum is dislodged to a lying side and corresponds to pressure of a column of broken-down rocks.

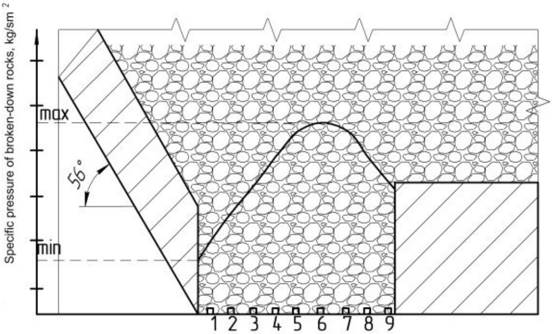

Fig. 1. Absolute values of loading on each strain gauge: 1 - 9 – accordingly a location of strain gauges on the block bottom

Table 1

Results of laboratory researches

Strain gauge number	1	2	3	4	5	6	7	8	9
Absolute values of loading on each strain gauge, кg/cm^2	20,8	26,0	31,0	41,5	47,5	53,3	49	48	39,2

Pressure of broken-down rocks upon the block bottom at mining of thick steeply dipping deposits of the big prodeleting depends not only on depth of the mining operations, determining the elevation a column of broken-down rocks over the bottom, but also from the horizontal sizes of the block, from intensity of an ore drawing and sequence of operations in a mine field, defining the sizes and a location of the areas which are simultaneously being at a stage release.

Accepting certain parities between these parameters it is possible to regulate size of pressure of broken-down rocks, over a wide range keeping it at level considerably smaller weight of all columns of broken-down rocks.

The analysis of results of laboratory researches allows drawing following basic conclusions:

1. In the separate block located the long side an across the strike of a deposit, the size and character of distribution of pressure of broken-down rocks on the bottom basically depends on the sizes of the block

along the strike. With increase in the size of the block along the strike increases not only absolute size of pressure, but also non-uniformity of its distribution. Thus the pressure maximum in the block located in a massif is mainly dated for points of a lying side, and in the block located between broken-down rocks – in the center.

2. Along with the horizontal sizes of the block considerable influence on size and character of distribution of pressure caused in the weight of broken-down rocks, renders intensity and an order of an ore drawing from the block. The increase in intensity of an ore drawing promotes pressure decrease on the bottom. Experiences, however, show that depending on width of the block and a massif condition on boundaries of the block release influence is shown variously. Thus the release order has extremely important role. At non-uniform release by the most widespread in practice, instead of a decrease increase of pressure upon those sites of the bottom in which limits an ore drawing rather is observed is less intensive.

In connection with non-uniform distribution of reserves of the brought down ore over ore chutes for preservation of horizontal contact between ore and leaning barren rocks more intensive release is made usually from ore chutes at a lying side. Thereof along with sharp pressure decrease in this part of the bottom pressure increase in the center of the block and at a trailing side is observed.

It occurs, because at an ort to the scheme of preparation of the bottom of inset horizon the storage drift usually settles down in soils of a lying side, therefore release occurs basically from first and second pair of final cone raises. From here more often, the air roadway where the ore drawing occurs not intensively (the big length of scraping) from the cone raises located around air roadway fails, and pressure upon bottom developments in this area first of all therefore increases.

The release role is especially great in the event that as vertical boundaries of the block broken-down rocks serve.

At a non-uniform and not intensive ore drawing from the block adjoining from two sides with broken-down rocks, pressure is considerable above, than in the block being in a massif, even in the event that from the block in a massif the ore drawing is not made.

3. Direct dependence of pressure on the bottom of the block which is in a stage of release, from the size is along the strike fair not only for the separate block, but also for a number of simultaneously developed

blocks divided by intermediate pillars of sufficient width, presented by an ore in place or immobile broken grounds (fig. 2).

Under the specified conditions the size and character of distribution of pressure in each block is defined by its horizontal sizes and features in the release organization the same as and in the isolated block.

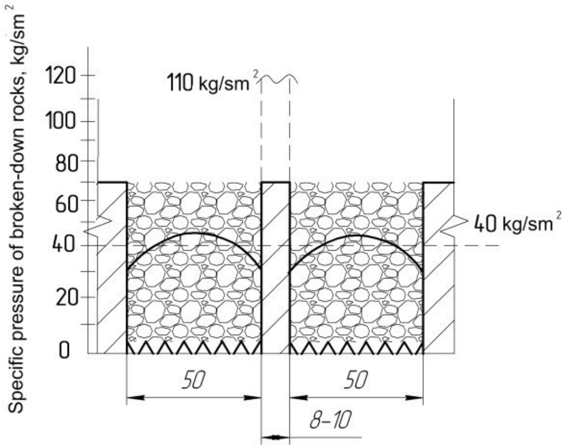

Fig. 2. Simultaneously developed blocks divided by intermediate pillars of sufficient width

Sharp change of pressure upon the top plane of intermediate pillars is observed.

The absolute size of average pressure upon an intermediate pillar is considerable and in 1,3-2,0 times of more pressure of all column of broken-down rocks, increasing with reduction of width of a pillar (fig. 3).

Loadings on a pillar are distributed non-uniformly, increasing from the middle to lateral planes.

Specific pressure of broken-down rocks at width of a pillar: 8 m - 110 kg/cm^2; 10 m - 100 kg/cm^2; 20 m - 65 kg/cm^2; 30 m - 52 kg/cm^2; 40 m - 52 kg/cm^2.

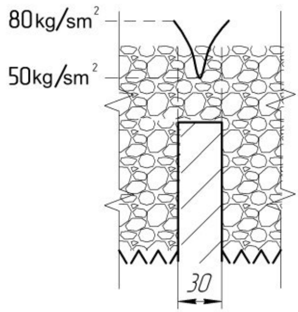

Fig. 3. Absolute size of average pressure upon an intermediate pillar

4. A role of pillars at a simultaneous ore drawing from a number of adjacent blocks can carry out also between the unitized sites presented by dabed brought down ore, release from which is not made. At non-uniform release at the expense of unloading of pressure upon bottom sites from which intensive release at present is made, the next volumes of broken-down rocks are in addition loaded. Thereof along with sharp pressure decrease on the sites of the bottom located at a lying side where intensive release is made, pressure in the block center considerably exceeds pressure of all columns of broken-down rocks [17,18].

At an ore drawing there is a redistribution of pressure upon the block bottom. Over a final aperture from which made release, the loosening zone in communication, with what pressure within a zone is formed decreases. The zone radius in which there is an appreciable decrease of a rock pressure, depends on quantity of the let out ore (size of a dose of release) and can be defined under the formula

$$R_з = \sqrt[3]{Q} + 1{,}07d, \tag{1}$$

where Q - quantity of the ore let out from an aperture, kg; d - diameter of a final aperture, m.

Outside of a breakage zone the brought down ore condenses, and pressure upon the bottom increases. The gain size depends on initial pressure (before release from a final aperture) and from point distance in which gauging's of pressure to a final aperture from which release is made are made.

By experiences it is established, that after an ore drawing from an aperture 100–120 м^3 on distance of 8–10 m from its axis pressure increases by 15–20%, and on distance of 12–16 m – on 30–40% from initial. On more remote sites pressure increase gradually fades. Such laws of distribution of pressure are characteristic at release from one aperture.

At an in regular intervals-consecutive mode of an ore drawing from the brought down panels and blocks these laws continuously repeat in process of inclusion in operation of other apertures. Pressure change (a decrease or increase) occurs in steps in the expiration of ore from a final aperture, and its size changes from a minimum to a maximum in any point of the panel (block). The size of the minimum pressure does not depend on a location of an aperture within the panel (block) and (its) its sizes, and is defined only by quantity of the released ore [19-21].

At release from an aperture of ore of 20-30 cm^3 the size of the minimum pressure is in limits 35-40 g/cm^2. The further ore drawing causes rock pressure slight increase in connection with increase in height of an ellipse of loosening. The size of the maximum pressure depends on the sizes of the panel (block), intensity and an ore drawing order.

The increase in length of the panel (block) an across the strike at invariable to width along the strike, intensity and an ore drawing order leads to growth of size of the maximum pressure upon the bottom of the panel (block). In panels in the size 24×60 cm is pressure was on 25-27% more than in panels in the size 24×30 see especially sharply the size of the maximum pressure upon the panel bottom increases at increase in width of the panel along the strike.

Influence of an order of release on rock pressure size:

1. The most expedient from the point of view of rock pressure reduction is simultaneous release by echelons from an ore in place to the center of the area of the brought down panel and from the center to a goaf. At stepping release and an ore drawing echelons in a direction from the center to a goaf it made 0,82-0,85$H\gamma$ - weight of a column of the material filled in in model [22].

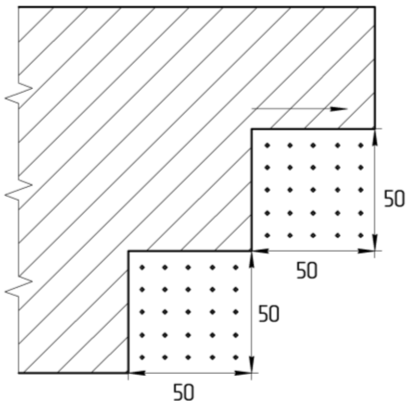

Fig. 4. Stepping ore drawing in a direction from the center to a goaf

2. At stepping release in a direction from a goaf and simultaneous release by echelons from the center to flanks it approximately on 10% more low.

The most expedient order of release in relation to first two on 26% more low also makes only 0,57Hγ.

Influence of a dose of release and intensity on rock pressure size:

1. At intensity increase, and, hence, and frequencies of an ore drawing from separate apertures pressure upon bottom developments decreases.

2. The pressure size, both minimum, and maximum with increase in a dose of release increases.

From the point of view of rock pressure size on the bottom simultaneous release from all final apertures of the panel when ore delivery is made by shuttle scrapers or conveyors is optimum. To the release beginning thanks to inclusion in operation of all final apertures of the panel there is a gradual breakage of ore on all area and pressure decrease on the bottom. Intensity of g/cm^2; t/m^2; in a minute or a day for the second example (*I* - 0,02 g/cm^2; *II* - 0,04 g/cm^2; *III* - 0,072/cm^2 [23,24].

The deviation of size of the maximum pressure at in regular intervals-consecutive release to pressure at simultaneous release from all final apertures of the panel fluctuates from 1,48 to 1,63.

Laboratory researches by definition of indicators of extraction of ore and rock pressure sizes (fig.5) on developments of the bottom of inset horizon at recommended technology of working of thick ore deposits of a Kryvyi Rih.

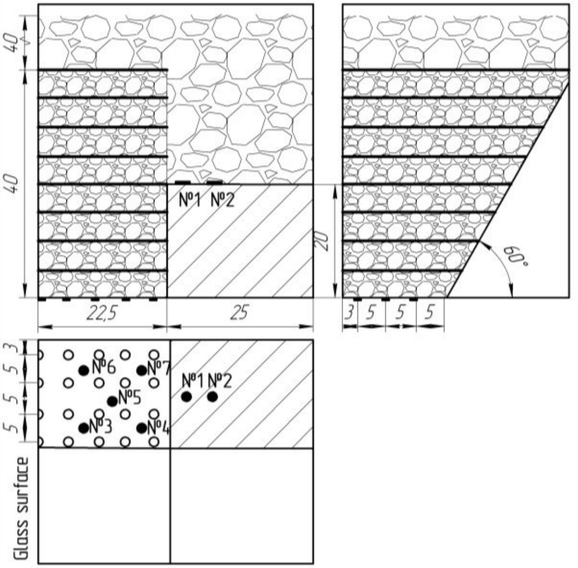

Fig. 5. Volume model for laboratory researches

The volume model is executed to scale M 1:100 with a forward glass wall (fig. 1). The distance between final apertures – 5 m, diameter of final apertures - 2 m. As a loose material is used martite ore from granules by metric structure +1÷-5 mm. Leaning soils are presented to models by a crumb of granite from granules metric structure +1÷-5 mm. Height of the brought down layer in the panel - 40m. Length of the panel which caves in along the strike - 25 m, and in an across the strike - 20 m.

For definition of pressure upon the bottom of inset horizon depending on a way and an ore drawing order in the panel bottom 5 strain gauges (fig. 1) take places. For definition of pressure upon a temporary pillar, a length along strike – 25 m and height – 20 m take places two strain gauges on the central line of a pillar on distance – 5m from territory of a pillar and 10 m between strain gauges on the central line.

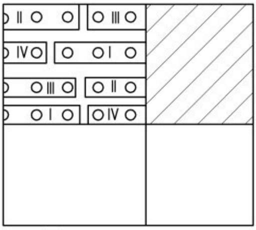

Fig.6

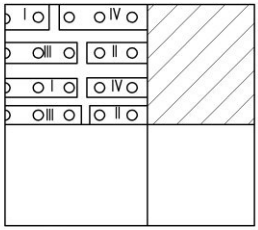

Fig.7

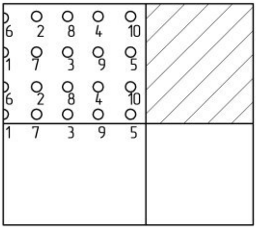

Fig.8

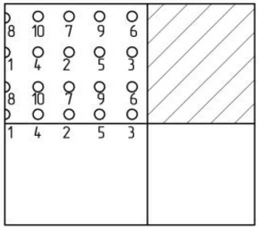

Fig.9

Fig. 6, 7, 8, 9. Order of release of the brought down ore in the panel

On fig. 6, 7 the order of release of the brought down ore in the panel is shown at a recommended way of an ore drawing by in regular intervals-consecutive zones on 2–3 final developments. On fig. 8 and fig. 9 is shown an order of release of the brought down ore at an in regular intervals-consecutive mode of release by echelons in a direction from a lying side to a trailing side (fig. 9) and is stepping in a direction from a lying side to a trailing side (fig. 8).

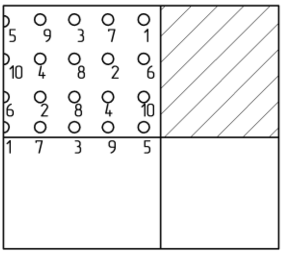

Fig.10

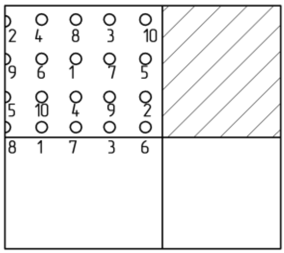

Fig.11

Fig. 10, 11. Order of release of the brought down ore in the panel

On fig. 10 the order of release of the brought down ore in the panel serially through one final aperture prior to the beginning of dilution is shown.

On fig. 11 the chaotic order of release of the brought down ore in the panel is shown.

On above described ways of release the technology of an underground extraction systems of a sublevel caving of ore and adjacent strata with breaking of ore by fans of deep boreholes and delivery of broken ore by loading technics is recommended (fig. 12).

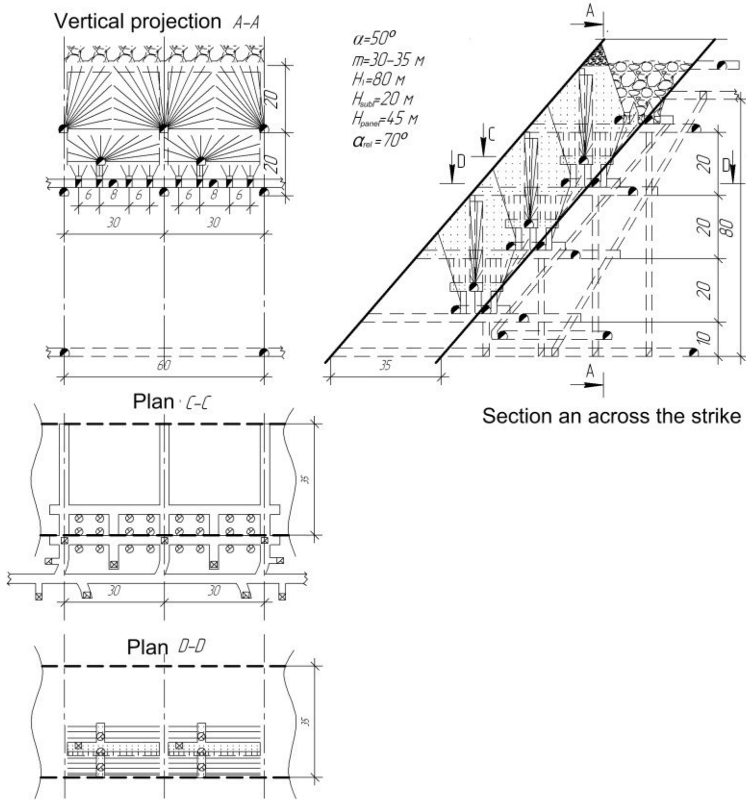

Fig. 12. Mining method of a sublevel caving of ore and adjacent strata with application of thick loading technics; m - power of a deposit, m; H_l - height of level, m; α - a deposit pitch angle, hailstones.

Through 60 m drive orts-arrivals which have connection with haulage gate. Provided that the deposit pitch angle makes 50 degrees, deposit mining will be conducted by three sublevels, on each of which the vertical kerf is formed. Breaking is conducted by fans of deep ascending boreholes on vertical kerf.

On sublevels broken ore haul to ore chutes which incorporate to orts-arrivals. Further ore is overloaded with loading technics.

Conclusions. Thus further the recommended technology of mining leads to the general increase of intensity of working of sublevels. At the expense of increase of intensity of working of a sublevel efficiency of working of reserves as terms of their service decrease increases as a whole.

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DEVELOPMENT OF INTELLIGENT SYSTEMS FOR OPTIMAL PROCESS CONTROL

Suleimenov B. A.
Satbayev University, Doctor of Engineering Sciences, Professor, Head of Department "Automation and Control", Kazakhstan

Kulakova E.A.
Satbayev University, Master of Engineering and Technology, Doctoral Student of Department "Automation and Control", Kazakhstan

Abstract

The search for resource-saving technologies implies not only research in the field of new methods of processing raw materials, but also new approaches to managing technological processes. Another important way of resource saving is the ability to quickly assess the technical condition of equipment, which will allow to timely prevent the emergency state of the main and auxiliary technological equipment. The paper deals with the development of intelligent control systems and diagnostics of processes using fuzzy logic, neural networks and hybrid models for creating control algorithms. The subject of the research is control algorithms developed to determine the key variables of the copper melting process in a liquid bath and algorithms for evaluating the technical condition of turbine units. All algorithms are developed on the basis of PFE matrices compiled by experienced technology experts during a "thought" experiment. The resulting models are carefully studied and analyzed for their sensitivity, stability, and single-valuedness. The absolute error of discrepancy between experimental and calculated variables became the criterion of adequacy.

Finally, the conducted researches have shown high efficiency of the control algorithms, obtained by using the artificial intelligence methods. In comparison with classical methods of building analytical and statistical models, methods based on the knowledge and experience of human experts allow creating optimal control systems for complex technological processes significantly easier, faster and more efficient.

Introduction

One of the main ways of resource saving in the mining and metallurgical industries is the introduction of automated systems of optimal process control (APCS) at industrial enterprises in these industries, which allow more efficient process control with the least loss of material, heat, electricity, labour and other resources of an enterprise.

The development of optimal control systems began in the 60s of the last century. However, so far in the world, few effective automated control systems have been implemented – the main reason is that it is not possible to create adequate mathematical models of technological processes due to the extreme complexity of modern technologies.

The rapid surge in the theory and practice of artificial intelligence (AI) methods in recent years has allowed us to find a simpler and more effective way to develop APCS using modern intelligent technologies. Let's consider the use of AI methods for the synthesis of automated control systems.

1 The concept of creating intelligent control systems

The analysis of works in the field of theory and practice of artificial intelligence has shown that currently there have been developed effective artificial intelligence technologies that are used in various practical applications, including control. However, most authors use these technologies for the development, research, and implementation of local control systems designed primarily to solve problems of stabilizing certain process variables.

We have not been able to find in the literature any examples of using intelligent technologies (IT) for creating optimal process control systems. As noted above, traditional methods of developing sufficiently adequate mathematical models of processes have not allowed the implementation of any significant control system in practice (at least in Kazakhstan).

In this regard, in this work we set the goal to further improve and develop information technologies, with the account of the specific features of technological process control.

Numerous studies conducted by our department, as well as the analysis of publications, have shown that IT can be used in the development of a straight optimal process control model, rather than a model of the technological process itself. That is the considered technologies allow immediate developing control algorithms, in contrast to the traditional chain: development of a process model structure → experimental research on site → model identification → formulation of optimization problem → selection of optimization method → development of an optimal control algorithm. The traditional approach assumes a long (sometimes taking several years), expensive and not always successful way of development of an optimal control system.

Employing IT allows solving similar tasks immediately, and as experience has shown, quite successfully. The fact is that the methods of artificial intelligence involve using the knowledge, experience and intuition of human experts who are familiar with the subject area. In other words, the so-called “ready knowledge” effect is used here. In contrast, the development of a mathematical model (the main component of the system) is a process of creating “new knowledge”, and therefore requires quite a long time for conducting theoretical research, as well as large material and labour costs for conducting empirical research and identifying the model.

Using the main idea of the work (development of a process control model instead of a process model) and developing existing IT methods, we propose the following three-stage procedure for creating optimal process control systems.

At the first stage, a priori studies of the technological features of the control object are carried out; they are based on literary sources, publications in periodicals and factory technical documentation. As a rule, existing technological processes had to go through a long stage of scientific research, pilot and industrial tests before they were implemented into production. There are probably still materials from these studies, as well as attempts to create mathematical models of this process. A thorough analysis of all this information is necessary in order to use it in the development of intelligent control systems.

At the same stage, it is necessary to analyze the process under study as a control object with the identification of input and output, controlled and uncontrolled, managed and unmanaged variables. It is necessary to estimate the response rate of an object, and an object’s type (continuous or discrete), the degree of information entirety of variables, the operating range of the variable change of an object, etc.

After a thorough analysis of the available information, it is necessary to draw up the structure of the future control system, which will greatly facilitate further work.

At the second stage, a model of the control process is developed. With the help of experienced experts (process operators, or engineers and technicians of a workshop or factory), the main control goal (analogous to the target function in optimization problems) is determined; it is usually known, and experienced operators usually strive to achieve it. Then, using the ranking method, the main (according to experts) variables for this object (process) from the general list of all types of variables are determined.

The main task of the second stage is to compile a matrix for planning a complete factorial (CF). The CF matrix and a three-level evaluation of input and output variables are used to create an object (process) control model.

In this case, for three-level factors, the total number of possible combinations of the number of factors for two input variables is $N=3^2=9$, for three variables - $3^3=27$, and so on.

For example, with two input variables, a PFE planning matrix is compiled, shown in table 1. Tables of this type are the basis for the development of intelligent systems, as they focus many years of experience, knowledge and intuition of people who are experts in a particular subject area.

Table 1

PFE planning Matrix

The number of the experiment	X_1	X_2	$Y^э$ expert assessment
1	0,0	0,0	
2	0,0	0,5	
3	0,0	1,0	
4	0.5	0,0	
5	0.5	0,5	
6	0.5	1,0	
7	1,0	0,0	
8	1,0	0,5	
9	1,0	1,0	

Values: 0.0; 0.5; 1.0 mean the minimum, average, and maximum values of the input variables X_1 and X_2. The expert can only use his experience, knowledge and intuition to set the values of the output variable Y^E (control action) in the range from 0,0 to 1,0. Normalization in the range from 0 to 1 of the input and output variables is performed by the formula

$$\bar{x} = \frac{x - x_{\min}}{x_{\max} - x_{\min}}, \quad (1)$$

где $\bar{x}$ - normalized value of the input or output variable; x – running value of the variable; $x_{\min}$, $x_{\max}$ – minimum and maximum value of the variable.

Compiling an experiment planning matrix is much more convenient for experts than drawing up rules for fuzzy products of the type recommended in all textbooks and publications. At the same time an expert does not need to invent infinite terms: “very much”, “very, very little”, “quite sufficient”, etc. He simply puts the value of the output (control) variable in the range from 0.0 to 1.0. And the CF planning matrix can be used for three different methods of creating a control model: expert systems, neural networks, and neuro-fuzzy algorithms.

In contrast to the well-known classical method of experiment planning, drawing up the CF planning matrix with the help of experts significantly speeds up and reduces the cost of this procedure. Experts conduct so-called “mental experiments” instead of expensive, actually conducted active experiments. In addition, it should be taken into account that conducting active experiments under the live production conditions is unrealistic due to the possible emergency situations when changing process variables from their minimum values to their maximum values, and vice versa. In addition, many enterprises simply do not have possibilities to change variables according to the CF planning matrix.

It should be emphasized that Y_i output values are actually controlling variables, so the planning matrix displays the process control model for all the combinations of input variables planned by experts. To calculate values in intermediate combinations of input variables, it is necessary to synthesize a process control model, which is the main task of the second stage.

It should be noted that it is most effective to use well-known mathematical dependencies identified at the first stage of research along with intelligent models. At the same time, it is necessary to make sure that such dependencies adequately reflect certain physical and chemical laws of a particular process.

At the third stage, the created control models are studied. At that the following actions are performed.

The resulting models are carefully studied and analyzed for their sensitivity, stability, and single-valuedness. For this purpose, the control process is modelled for various changes in input variables; curves are drawn for changes in output variables on the account of input variables changes, and their analysis is performed employing experts.

After the study of models obtained by different methods is completed, a comparative analysis for their adequacy is carried out. For this purpose, models are used to calculate the output variables according to the values of input variables taken from the CF planning matrix and compare them with the estimates provided by the expert. After that, a comparison matrix is formed (see Table 2), which allows calculating the value of the modelling error by various means.

The absolute error defined as

$$\delta = 100 \frac{1}{N-1} \sum_{i=1}^{N} \left| Y^E - Y^C \right|, \qquad (2)$$

где Y^Eи Y^C - experimental and calculated values of output variables respectively.

Table 2

Matrix of comparison of calculated and experimental values of the output value

The number of the experiment	X_1	X_2	Y^C model value	Y^E expert value
1	0,0	0,0		
2	0,0	0,5		
...	0,0	1,0		
9	1,0	1,0		

The absolute error is calculated for models obtained using three different approaches, and then their comparative analysis is performed. The model with the smallest absolute error is considered the most appropriate.

The most appropriate model is to undergo simulation tests under the live production conditions. At that, the model input is supplied with real input variables taken from the measuring equipment of the industrial unit, and the simulation results (the output control variable) are compared to the control value actually carried out by an experienced operator-technologist. If the simulation test results are satisfactory, the model is integrated into the industrial controller. Otherwise, the process starts from the beginning - returning to the first stage, and updating all of the model parameters.

Further, the paper shows how the proposed concept is used for the synthesis of an optimal control system for the process of copper melting in a liquid bath.

2 Development of an optimal control system for the process of copper melting in a liquid bath

When considering the technology for processing copper concentrates in a liquid bath furnace (LBF), it is necessary to take into account the fact that this process is relatively new, has not yet been sufficiently studied and has been implemented only at a few facilities of non-ferrous metallurgy enterprises in the world.

2.1 Characteristics of the LBF process as an object of control

The object of control is a LBF industrial unit. The LBF industrial unit is a two-zone caisson furnace with continuous roof charging into the melting zone, side blast into the melt and continuous output of melting products from the reaction zone.

The charge is prepared in advance for the entire campaign based on calculations of the melting mode and data on the composition of processed materials. Once in the melt, the charge is heated and dissolved, the higher sulphides of the charge dissociate and form the simplest sulfide compounds and elemental sulfur. The products of dissociation interact with the oxygen of the blast bubbling the melt, resulting in the formation of matte, slag and sulfurous anhydride rich gases. The matte-slag emulsion gradually delaminates as it descends down. The matte forms a continuous given phase, and the slag enters the reduction zone through the siphon, after which it flows through the second siphon into the electric slag settling tank. Periodically, as the matte accumulates, it is released from the oven into a bucket and poured into the molds, after being weighed.

The slag from the electric settling tank is also released into a bucket and poured into the molds. After solidification, the melting products are taken out of the shop.

The recovery zone is similar in design to the melting zone and is used for studying the modes of processing ores containing zinc and lead; in other cases it is used as a flow container.

The furnace gases are removed from each zone through flue ducts and sent to a system consisting of a cyclone filter, scrubber, CO afterburning and cooling chambers, and enter either the chimney or the system for obtaining elemental sulfur.

From the characteristics of the process of processing materials in the liquid bath furnace at Balkhash Mining-and-Metallurgical Integrated Works, it can be concluded that the control object is a technological process with a complex relationship of its characteristic parameters. Among the features of the LBF at Balkhash Mining-and-Metallurgical Integrated Works the following should be highlighted:

- a small response rate of the object along the channel: blast - the temperature of the melt and waste gases; and a relatively large one along the channel: blast, the composition of the input products - the composition of matte and slag;
- the object can be classified as continuous (continuous feed of charge, blast, continuity of basic transformations (heating, melting, oxidation, recovery of materials, mass and heat exchange) continuous release of slag and periodic release of matte, continuous exhaust of waste gases);
- multidimensionality of the technological process characterized by a large number of input and output variables;
- non-stationary process parameters due to fluctuations in the values of input variables because of insufficient charge averaging;
- incomplete information (the presence of periodically controlled parameters, as well as delays both on the object itself and in the measurement channels, a significant level of interference in the latter, etc.);
- narrow operating range of the melt temperature in the melting zone;
- the processes of processing various types of raw materials studied at the LBF unit have significant differences in the nature of the physical and chemical phenomena.

2.2 Development of models for control the LBF process based on the PFE matrix

The using of intelligent technologies allows us to solve problems of synthesis of optimal control algorithms immediately, and as experience has shown, quite successfully. With the help of experienced experts (operators-technologists, or engineers of a workshop or factory), it is necessary to determine the main goal of management (analogous to the target function in optimization problems), which is usually known and which experienced operators usually strive to achieve. Thereafter, using the ranking method, from the general list of all types of variables, those that, according to experts, are the main ones for this object (process) are determined.

The main task in the development of the control model is creation of matrix for planning a complete factor experiment (PFE). The quality of the PFE matrix significantly affects the efficiency of the entire control system. The PFE planning matrix should reflect the experience, knowledge, and intuition of process operators. The performance of the entire control system will depend on the quality of the PFE matrix.

The task of the automated process control system is determination of the optimal process management modes: Y1, Y2 and Y3, depending on the composition of the initial charge: X1, X2, X3 and X4. As a rule, such calculations must be made once per shift. The survey of the shop's technologists allowed to create a matrix of PFE planning for 81 experiments with a three-level assessment (0,0; 0,5 and 1,0), four input variables: N=3^4=81. For more accurate modeling of the control process, two additional levels were added: 0.25 and 0.75. Since the number of experiments would increase many times, these levels were entered only for the normal conduct of the process. Consequently, combinations were obtained for 89 experiments (table 3).

Table 3

Fragment of the PFE planning matrix

№	Input variable				Output variable		
	copper content in the charge X_1	sulfur content in the charge X_2	silicon oxide content in the charge X_3	magnetite content in the charge X_4	consumption of blast Y_1	the flow rate of concentrate Y_2	oxygen enrichment of the blast Y_3
1	0	0	0	0	0,12	0,87	0,18
2	0	0	0,5	0	0,37	0,62	0,25
3	0	0,5	0	0	0,37	0,62	0,31
4	0	0,5	0,5	0	0,5	0,62	0,62
…	…	…	…	…	…	…	…
86	1	1	0	1	0,43	0,43	0,62
87	1	1	1	1	1	0,68	0,81
88	1	1	0,5	1	0,75	0,56	0,75
89	1	0,5	1	1	0,87	0,62	0,62

It should be emphasized that in table 3, all variables had been normalized in the range from 0.0 to 1.0. The variables had been recalculated in accordance with formula 1.35, while the maximum and minimum values of input and output variables were determined from practice.

The PFE planning matrix is compiled by experienced technologists using a "thought" experiment. Therefore, compiling such a matrix is significantly simpler than according to an active experiment. Table 3 focuses on the long-term experience of technologists at the melting furnace in Balkhash.

The PFE planning matrix can be used in the development of control models in three ways: experiment planning method, fuzzy modeling method, neural network method, and neuro-fuzzy methods. The creation of models for controlling the melting of copper in PV using each of these methods is discussed below.

2.3 Synthesis of a fuzzy control model

The fuzzy model was developed using Matlab graphical tools. Then the membership functions for three input and two output variables were defined. For this purpose, the Matlab membership function editor was used. Then the fuzzy production rules were formed. At the same time, each experiment from the PFE planning matrix corresponded to its own rule. For example, for experiments №1, №2 и №3 , the production rule looks like this:

RULE 1: “IF COPPER CONTENT IN THE CHARGE IS THE MINIMUM” AND “SULPHUR CONTENT IN THE CHARGE IS THE MINIMUM” AND “SILICON OXIDE CONTENT IN THE CHARGE IS THE MINIMUM” AND “MAGNETITE CONTENT IN THE CHARGE IS THE MINIMUM" THEN “THE CONSUMPTION OF BLAST IS 0.12" AND "THE FLOW RATE OF CONCENTRATE IS 0.87” AND “OXYGEN ENRICHMENT OF THE BLAST IS 0.18".

RULE 2: “IF COPPER CONTENT IN THE CHARGE IS THE MINIMUM” AND “SULPHUR CONTENT IN THE CHARGE IS THE MINIMUM” AND “SILICON OXIDE CONTENT IN THE CHARGE IS THE MEDIUM” AND “MAGNETITE CONTENT IN THE CHARGE IS THE MINIMUM" THEN “THE CONSUMPTION OF BLAST IS 0.37" AND "THE FLOW RATE OF CONCENTRATE IS 0.62” AND “OXYGEN ENRICHMENT OF THE BLAST IS 0.25".

RULE 3: “IF COPPER CONTENT IN THE CHARGE IS THE MINIMUM” AND “SULPHUR CONTENT IN THE CHARGE IS THE MEDIUM” AND “SILICON OXIDE CONTENT IN THE CHARGE IS THE MINIMUM” AND “MAGNETITE CONTENT

IN THE CHARGE IS THE MINIMUM" THEN “THE CONSUMPTION OF BLAST IS 0.37" AND "THE FLOW RATE OF CONCENTRATE IS 0.62” AND “OXYGEN ENRICHMENT OF THE BLAST IS 0.31".

The production rules for all 89 experiments are made in the same way.

After Matlab performs all the necessary procedures in accordance with the selected fuzzy output algorithm (for example, the Mamdani algorithm), a fuzzy model of optimal control of the copper smelting process is presented in the graphical interface for viewing the rules (see figure 1). Thus, the interface shown in figure 1 is a model (algorithm) of optimal control, which can be used to simulate various modes for all possible combinations of input variables.

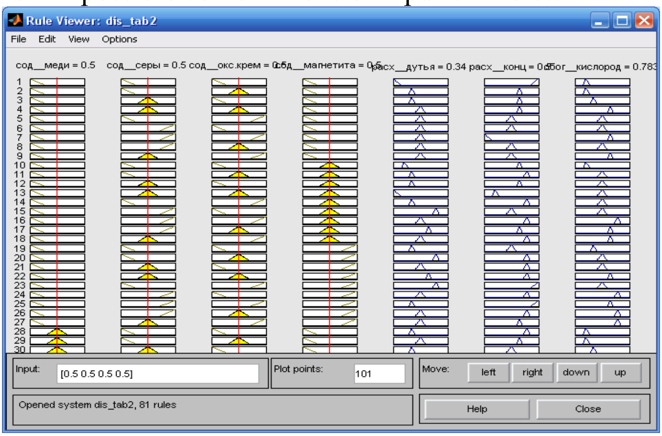
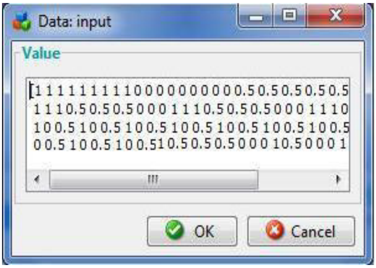


Fig. 1. Fuzzy control model (algorithm)

2.4 Synthesis of neural network model

Instead of fuzzy models, neural networks can also be used to model the process of managing modes in the PV. For training the neural network, the results of 89 experiments from the PFE planning matrix were entered into the Matlab program, an example of such input is shown in figure 2. Output variables (control actions) were entered using the "Datatarget" window (figure 3).



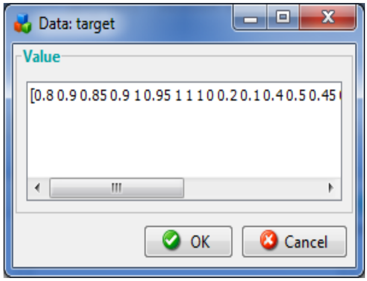


Fig. 2. Creating input data

Fig. 3. Creating target data

At the next step a neural network was created. The input data field specified pre-created data, set the type of neural network, and selected a perceptron (Feed-Forward Back Propa-gation) with 10 sigmoid (TANSIG) neurons of the hidden layer and one linear (PURELIN) neuron of the output layer. The training was performed using the Levenberg-Marquardt algorithm, which implements the TRAINLM function. The error function is MSE, and the number of layers is 2, respectively. The progress and results of the training are shown in figures 4 and 5.

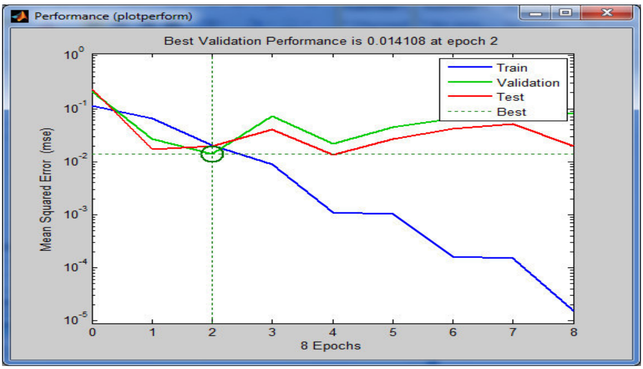


Fig. 4. Neural network learning progress

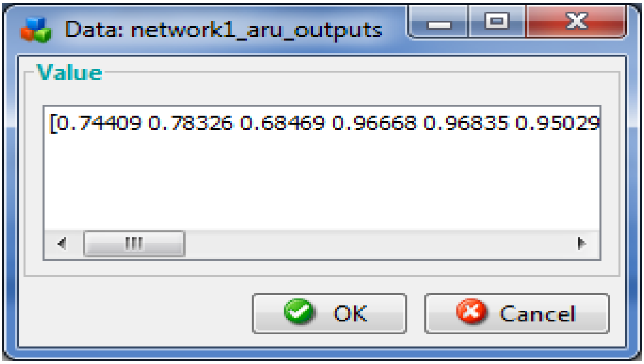


Fig. 5. The result (output) of training a neural network

2.5 Synthesis of neuro-fuzzy control model

Another method of modeling using intelligent technologies is hybrid models, such as neuro-fuzzy networks, which should combine all the advantages of the two methods listed above.

These researches can be performed using MATLAB. To do this, MATLAB has an ANFIS editor that allows you to create or load a specific model of an adaptive neuro-fuzzy inference system, train it, visualize its structure, change and configure its parameters, and use the configured network to get fuzzy inference results

The network presented in figure 6 is a model for managing the process of smelting in a liquid bath (SLB) using neuro-fuzzy algorithms.

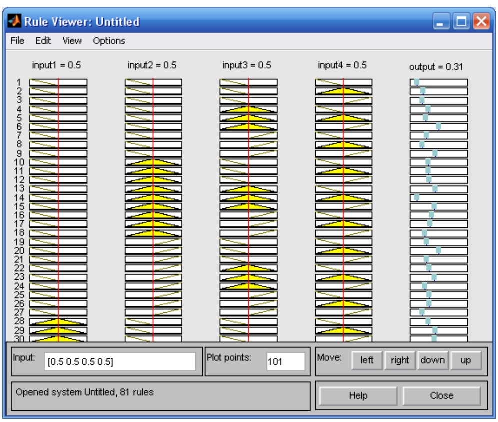


Fig. 6. Neuro-fuzzy model

In the future, this model can be used to calculate output variables for any changes in input variables.

2.6 Results of research of intelligent models

The obtained control models allowed us to simulate various modes of process management depending on the composition of the run-of-mine charge.

The simulation results are listed in table 4.

Table 4

Comparative estimation of absolute error for different methods

Modelling method	Absolute error %		
	Y_1	Y	Y_3
1. Design of experiment	15,53	33,01	12,68
2. Fuzzy algorithms	4,462	4,997	3,612
3. Neural network algorithms	2,26	2,8	2,78
4. Neuro fuzzy networks	1,333	1,002	1,241

The table 4 shows that the experiment planning method cannot be used due to unacceptably high values of absolute errors: from 12% to 33%. Intelligent models showed their advantage: from 1.0% to 5.0%, while the best method is neuro-fuzzy networks (from 1.0% to 1.3%).

Finally, the conducted researches have shown high efficiency of the control algorithms, obtained by using the artificial intelligence methods. In comparison with classical methods of building analytical and statistical models, methods based on the knowledge and experience of human experts allow creating optimal control systems for complex technological processes significantly easier, faster and more efficient.

3 Development and research of algorithms for diagnostics of equipment technical condition

Another important resource-saving tool is the ability to quickly assess the technical condition of equipment, which will allow timely preventing the emergency state of the main and auxiliary technological equipment. Let's consider an example of using AI methods to assess the technical condition of turbine units (TU).

The energy sector is currently characterized by an intensification of the use of capacities and resources of installed equipment. This can be achieved based on intelligent diagnostics of the operational state and modes of equipment use. The increasing responsibility of decisions made on the time of equipment commissioning has tightened the requirements for the quality of identification models, which are based on the information obtained during diagnostics of the TU condition. Their implementation in the conditions of the old forms of maintenance under the PPR system (planned preventive repairs) has become ineffective. There has emerged a problem of insufficient adequacy of diagnostic models and decision-making models for putting TU into repair or reducing the load, due to not using fuzzy information about the condition of equipment, as well as increasing the total uncertainty accumulated during operation.

In order to implement the above three-stage procedure for creating a diagnostic subsystem for the case of turbine units, a survey of operators-technologists and ITR of the turbine shop has been conducted, which showed that the main variables that characterize the state of the turbine unit as a whole are the following:

$X1$- thrust bearing vibration;

X_2 - plummer bearing vibration;

X_3 - thrust bearing babbit temperature;

X_4 - plummer bearing babbit temperature;

X_5 - the axial shift in the direction of the generator;

X_6 - axial shift in the direction of the “chair”;

$X7$ - relative expansion of the high-pressure rotor;

X_8 - relative expansion of the low-pressure rotor;

X_9 - pressure of hydrogen in the generator housing;

X_{10} -temperature of hydrogen in the generator housing;

X_{11}- oil temperature after the oil cooler;

X_{12} -pressure in the discharge chamber of the high-pressure cylinder;

X_{13}- hot steam temperature; X14- hot steam pressure;

X_{15} -vacuum drop in the condenser;

X_{16} -metal temperature in a high-pressure cylinder;

X_{17} -metal temperature in the low-pressure cylinder.

All these variables are controlled by devices, which makes it possible to synthesize an automated system for rapid diagnostics of the technical condition of turbo units.

In accordance with the methodology proposed in this paper, a matrix for planning a complete factor experiment (PFE) is compiled for the synthesis of an intelligent model of diagnostics and forecasting.

However, in this case, it will be necessary to conduct a huge number of "thought" experiments, for example, for a three-level assessment, the number of experimental points will be $N=3^{17}$, which is completely unrealistic to implement.

Therefore, the decomposition of the problem of forming the PFE planning matrix is carried out. For this purpose, the influence of diagnostic features X_1-X_{17} on the state of the turbine unit as a whole, but on its individual main parts is evaluated, which will reduce the dimension of the PFE matrix. Taking into account that the steam turbine is a single-shaft two-cylinder unit designed for direct drive of the generator, it is proposed to consider the turbine unit as a set of the following main elements located on one shaft: a plummer bearing (PB); a high-pressure cylinder and rotor (HPC); a low-pressure rotor and cylinder (LPC); an alternating current generator (ACG); a thrust bearing (TB). In addition, due to the special danger, the including the hydrogen supply system (HSS) in the generator housing is planned. HSS will be considered as a separate element, not connected to the rest by a single shaft (figure 7).

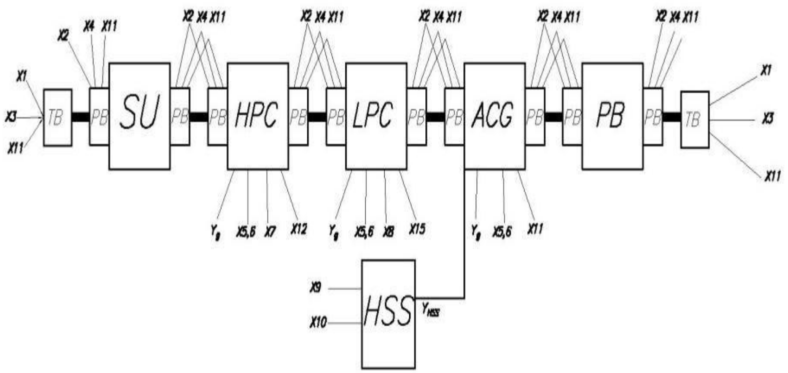


Fig. 7. Turbine unit elements and impact on technical condition of TU

Figure 7 shows that the technical condition of bearings is affected by vibration, babbit temperature and oil temperature, while the technical condition ratings of 10 bearings (Y_{PB} и Y_{TB}), in turn, are diagnostic features for assessing the technical condition of the HPC, LPC and ACG. Considering the fact that is these elements of the turbine are equally affected by the condition of both the plummer and thrust bearings. In this regard, it is proposed to take into account the assessment of the technical condition of only one of the 10 bearings that has the worst rating value (let's denote the value of this assessment in NP).

As a result, taking into account the assessment of the technical condition of only one of the 10 bearings that has the worst rating value (this rating is indicated by Y_g), there is a reduction in the number of diagnostic features per unit for each of the three main elements: HPC, LPC and ACG.

The assessment of the technical condition of the most dangerous hydrogen supply system (Y_{HSS}) depends on the values of the pressure and temperature of the hydrogen in the generator housing. At the same time, Y_{HSS} together with other diagnostic features ($X_{5,6}$, Y_g, and X_{11}) can serve as initial data for evaluating the technical condition of the generator as a whole.

Wherein, the axial shift to the generator (X_5) and the axial shift to the chair ($X6$) are mutually exclusive factors, i.e. the shift can be carried out either in one direction or in the other, so these two factors were combined into one - $X_{5,6}$, which reduced the number of diagnostic features by one for each of the three main elements: the HPC, LPC and ACG.

In addition, the variables X_{16} (metal temperature in the high-pressure cylinder) and X_{17} (metal temperature in the low-pressure cylinder) are used for evaluating the technical condition of the HPC and the LPC only in the process of preparing them for start-up during heating. In the course of normal operation of the turbine unit, they are not even controlled by a person, so they have been excluded from the number of diagnostic features, thereby further reducing the dimension of the problem that being solved.

The variables X_{13} (hot steam temperature) and X_{14} (hot steam pressure) are purely technological, depending on the physical state of the steam coming from the boiler shop. These variables cannot be used to assess the technical condition of a HPC or LPC, so they have been also excluded from the number of diagnostic features.

Thus, taking into account the reduction of diagnostic features, the assessment of each of the main elements of the turbine unit (TU) can be estimated only by four diagnostic features (figure 4.1), consequently, the number of "thought" experiments for each of the parts of the TU (HPC, LPC and ACG) will be $N=3^4=81$, which is quite simple to implement.

As a result of this research, four types of models for diagnosing the technical condition of the turbine unit have been synthesized. Comparative table 5 of these models illustrates that the best result was shown by neuro-fuzzy networks of 0.8% error, which is a very good result compared to traditional methods (the experiment planning method showed 6.7% error).

Table 5

Final comparison table for different methods

Modelling method	Absolute error %
	Y
1. Fuzzy algorithms	1,15101 %
2. Neural network algorithms	1,146136 %
3. Neuro fuzzy networks	0,88102 %
4. Design of experiment	6,75%

Due to the best result of the neuro fuzzy model, all further research will be conducted only on the basis of this model.

3.1 Sensitivity analysis of a neuro-fuzzy model

Sensitivity means that a small change in the input parameters leads to a change in the system's property index that can be detected under conditions of measurement error. The purpose of sensitivity analysis is to compare the impact of various factors on the result of solving the modeling problem.

The integrated circuit sensitivity analysis:

- selection of factors, a slight change in which can have a significant impact on the result from the point of view of the researcher;
- setting the nominal and maximum (lower and upper) values of the selected factors;
- solving the modeling problem in different ranges of source data for all selected limit values of factors;
- building dependencies for the sensitivity of the problem solution for all factors and ranges of changes in the source data.

Figure 8 shows the results of sensitivity estimation when all 4 input variables are changed by a value of 0.02.

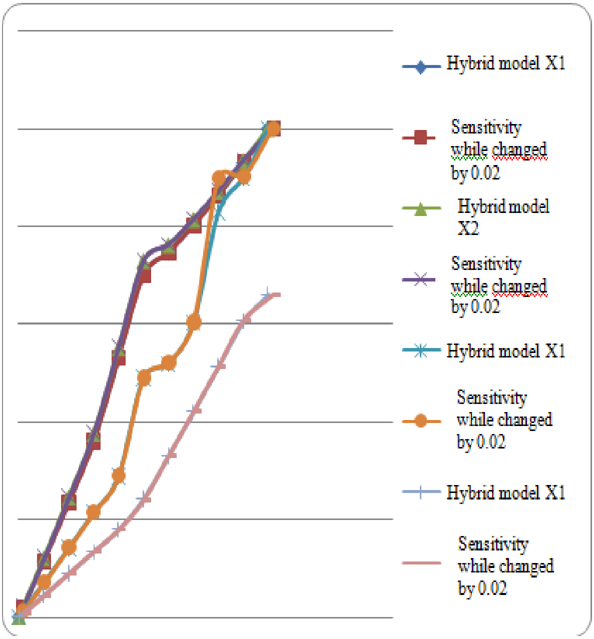


Fig. 8. Estimation of the sensitivity of the neuro-fuzzy model

A slight 0.02 change of input variables affected the output variables as shown in Figure 7. When diagnosing the technical condition of HPC change the values of input variables at 0.02 according to the logic and physics of the technological process should not greatly poulet on the technical assessment of HPC because of their insignificance. Fig. 2 shows that the curves before and after changes in the input variables by 0.02 slightly differ from each other, which fully explains and proves the high stability of the neural network model for diagnosing the technical condition of the HPC. In this case, theЫ neural fuzzy model is most sensitive to changes in the value of the relative expansion of the high-pressure rotor X_3 and X_2 of the axial shift towards the generator or the front chair.

3.2 Analysis of the results of modeling the assessment of the condition of the high pressure cylinder

It is proposed to consider simulations of estimating the state of the HPC (Y) for different rates of input variables: X_1,X_2,X_3 and X_4 (figure 9).

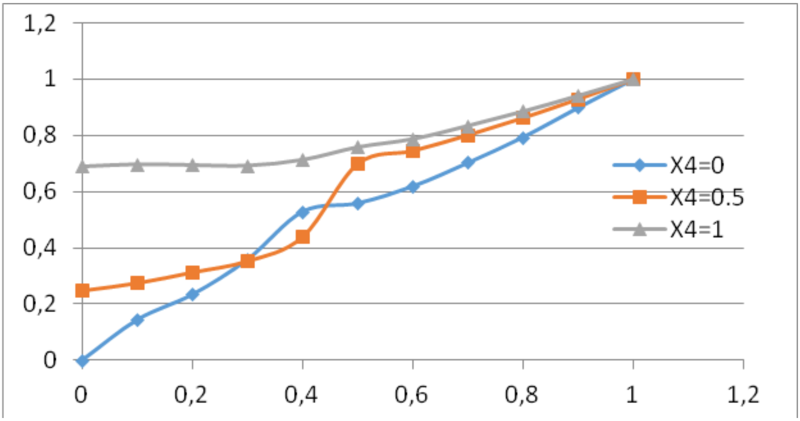


Fig. 9. Simulation Results for evaluating the technical condition of the High pressure cylinder depending on changes in the pressure of the discharge chamber at X_2=0 and X_3=0.5

Figure 9 shows the simulation results for evaluation technical condition of HPC (Y) depending on changes of the general condition of the thrust and plummer bearings (X_1) for various rates of the discharge pressure chamber HPC (X_4), equal to the maximum (X_4=1,0), medium (X_4=0.5) and minimum (X_4=0) rates. The simulation was performed at nominal rates of the axial shift towards the generator or «front chair» (X_2=0) relative expansion of the high-pressure rotor (X_3=0).

The figure 9 illustrates, the maximum value (Y=1) of the technical condition characteristic of the HPC is at the maximum rates of the general condition of the thrust bearing (X_1=1) at any pressure rates in the discharge chamber of the HPC (X_4=0, 0.5, 1.0). This is quite understandable – since the general condition of the thrust bearing equal to (X_1=1) is unacceptable and inevitably leads to a stop of the turbine unit, according to the operating instructions [1]. When the normal pressure in the discharge chamber of the HPC (X_4=0) and when the normal state of the thrust bearing (X_1=0.0) the overall technical condition of the HPC is the best (Y=0.0), and if the general technical condition of the thrust bearing deteriorates to (X_1=0.5), the overall technical condition of the HPC is satisfactory (Y=0.559)

From figure 9 it is also seen that at moderate pressure in the discharge chamber (X_4=0.5) is characterizing the wear of the discharge chamber HPC, and the best rates of the general technical condition of bearing

(X_1=0) general assessment of the technical condition of HPC satisfactory (Y=0.249) and increasing, that is, deterioration of the general technical condition of thrust bearing X_1, the overall assessment of the technical condition of HPC is deteriorating, and is at (X_1=0.5) is the overall technical assessment of HPC is equal to (Y=0,7), this means that the unit is no longer in its normal technical condition and requires increased attention from the operator who controls the turbine unit, and if necessary, a detailed inspection of the unit without stopping it.

Further analysis of figure 8 shows that at maximum pressure in the discharge chamber (X_4=1), and at nominal rates evaluation of the technical condition of the thrust bearing (X_1=0.5), the overall technical condition of HPC is distinctly out-of-band mode (Y=0,69), which is characterized by the poor condition of the discharge chamber of HPC, it is necessary to reduce the pressure in the discharge chamber HPC, by offloading that is, the reduction supplied to the steam turbine in order to prevent further destruction of the discharge chamber. And when the overall assessment of the technical condition of the thrust bearing is equal to (X_1=0.5), the overall technical condition of the HPC is equal to (Y=0.759), which is unacceptable for long operation of the turbine unit in these conditions.

The foregoing discussion has shown that the pressure value, that is, the technical condition of the high-pressure chamber of the HPC (X_4) is not critical, since it is always possible to unload the turbine and thereby reduce this indicator. In this case, the value of the indicator of the general technical condition of the thrust bearing is critical for the overall condition of the turbine unit, because it is impossible to reduce this indicator when the turbine is running.

Figure 10 demonstrates the modeling results of assessment of the technical condition of the HPC (Y), depending on the change in the rates of the axial shift in the line of the generator or front chair (X_2), for different rates of the pressure of the discharge chamber of the HPC (X_4), equal to the maximum (X_4=1.0), average (X_4=0.5) and the minimum (X_4=0.0) value with nominal rates of the relative expansion of the high-pressure rotor (X_3=0.0) and the general condition of the plummer bearing (X_1)

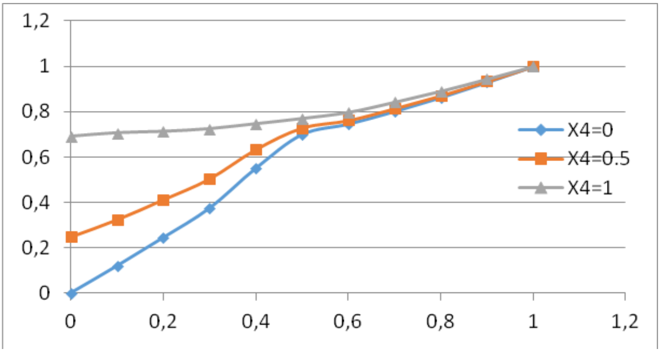


Fig. 10. Simulation results for evaluating the technical condition of the HPC (Y) depending on changes in the rates of the axial shift towards the generator or front chair (X_2) and at X_1=0 and X_3=0

Figure 10 shows that at minimal rates of discharge pressure chamber HPC (X_4=0.0) technical assessment of HPC is deteriorating with increasing axial shift in the direction of the generator or of the front seat (X_2). When (X_2=0.5, X_4=0.0) Y=0.7 a at (X_2=1.0) Y=1.0. Almost the same thing can be seen on the other two curves for estimating the technical condition of the HPC, which have different initial data depending on the pressure rates in the discharge chamber of the HPC. When (X_4=0.5, X_2=0.0) Y=0.249, when (X_2=0.5) Y=0.729 at (X_2=1.0) Y=1.0 and finally at (X_4=1.0, X_2=0.0) Y=0.69, at (X_2=0.5) Y=0.769 and at (x2=1.0) Y=1.0.

The analysis of figure 10 allows concluding that the value of the axial shift towards the generator or the front chair (X_2) is more critical than the pressure in the discharge chamber X_4. Since according to the operating instructions of the turbine unit - when the value of the axial shift in the line of the generator or the front chair increases and it is impossible to normalize it in the shortest possible time, the emergency shutdown of the turbine unit occurs.

During normal operation of the turbine unit, the axial shift is a fairly stable value, but with sharp changes in the parameters of the sharp steam, this indicator goes beyond the set value.

This is illustrated in figure 11

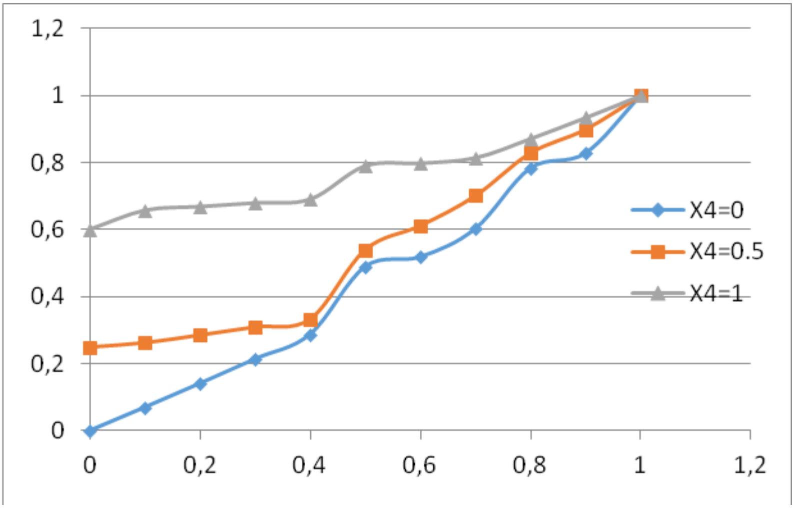


Fig. 11. Simulation results for evaluating the technical condition of the HPC (Y) depending on changes in the rates of the relative expansion of the high-pressure rotor X_3 and with nominal rates X_1=0 and X_2=0

As shown figure 11, which suggest how the overall assessment of the technical condition of the HPC (Y) changes, depending on the change in the relative expansion of the high-pressure rotor (X_3 from 0 to 1) and at the nominal rates of the general state of the thrust and plummer bearing (X_1=0), the axial shift in the line of the generator or the front chair (X_2=0) and at the rates of the pressure in the discharge chamber equal to X_4=0, X_4=0.5 and X_4=1.

Figure 11 shows that for different pressure values in the high-pressure discharge chamber, X_4=0.0, X_4=0.5 and X_4=1 of the initial points of the curves correspond to the value of the general technical status of HPC Y=0, Y=0.249 and Y=0,6. While increasing in the value of the relative expansion of the rotor high pressure X_3, and the values of the pressure in the high pressure chamber (X_4=0.0, 0.5 and 1), the value of the general technical status of HPC (Y) tend to 1 then to the emergency state at X_4=0 (X_3=0.5, Y=0.49), (X_3=1 Y=1) with X_4=0,5 (X_3=0.5, Y=0.54), (X_3=1, Y=1) with X_4=1 (X_3=0.5, and Y=0,79), (X_3=1, Y=1).

These data illustrate that the value of the relative expansion of the high-pressure rotor-X_3 is also critical when evaluating the overall technical condition of both the HPC and the entire turbine as a whole.

That meets the requirements of the operating instructions of the turbine unit. The expansion of the high-pressure rotor is accompanied by the same threats as the axial shift towards the generator or the front chair.

Thus, the simulation results of evaluation of the technical condition of HPC are reasonable in terms of physics of technological process unit and fully reflect the assessment of the experts – the operators of the turbines.

3.3 Recommendations for liquidation of emergency situation HPC

Further adopted the following graduation estimates the degree of closeness of the current state of HPC to the emergency situation:

a - if the value of *Y* ranges from 0 to 0.25 - HPC is in the normal running order;

b - if the value of *Y* ranges from 0.26 to 0.5 - emergency situation of HPC is possible, but more careful monitoring is required, and the necessary preventive actions of the operational personnel are given above;

c - if the value of *Y* ranges from 0.51 to 0.79 - it is a pre-emergency situation;

d - if the value of *Y* ranges from 0.8 to 0.1, an emergency situation has occurred

Depending on the value of the assessment of the technical condition of the HPC, the operational diagnostics subsystem can make one of the following decisions:

- in case (*a*) - do nothing;

- in case (*b*) is to analyze possible causes of the deteriorating condition of HPC: to check the temperature of bearings and the temperature of the cooling water; check circulation lubrication and the operation of the oil pump; check the pressure in the discharge chamber HPC and in front of it; check to see that lines of heating of flanges and studs; check the position of the turbine shaft on special marks;

- in case (*c*) - depending on the results of analysis to produce one of the following: isolate the cause of the temperature rise of the bearings, and eliminate it according to instructions and the above measures; if necessary, reduce the steam flow to the turbine head to the normalization of the pressure in the discharge chamber HPC; if necessary, adjust lines of the heating flanges and studs;

- in the case of (*d*) - act on the order of the shop technologist, shift Manager or Deputy head of the shop for operation or the person replacing him.

Similarly, it is possible to create a PFE planning matrix for evaluating the technical condition of other elements of the turbine unit: the Central heating unit, the PS, and the generator. Faults of this class are detected by the appropriate sensors, and the response to them is specified in the process instructions and can be duplicated on the monitor screen using the rapid diagnostics subsystem.

Thus, the proposed method of evaluating the technical condition of the turbine unit allows predicting the occurrence of emergencies at an early stage.

Conclusion

This paper describes the concept of creating intelligent control systems for technological processes. It presents the results of the development of intelligent algorithms for determining the key variables of the copper melting process in a liquid bath and algorithms for evaluating the technical condition of turbine units.

Experimental algorithms were created using fuzzy logic, neural and hybrid networks, which enable us to formalize the knowledge of experienced technologists.

The check of the adequacy of the algorithms showed that the minimum discrepancy between expert data and model data is inherent in the neuro fuzzy networks algorithm.

The absolute error value for the test sample of expert and model data of both applications (the copper melting process in a liquid bath and for evaluating the technical condition of turbine units) was 1,3% and 0.88%, respectively.

In its turn, this indicator for the classical method of design experiment made 15% and 6.5%. This proves the advantage of intelligent technologies over classical methods in identifying mathematical models of control objects.

Therefore, resulting algorithm can be used as an autonomous expert system or integrated into the overall control system of the described technological processes. This eliminates the influence of the human factor and enable us achieving high technological indicators and predicting the occurrence of emergencies at an early stage.

The arguments that have been presented in this paper suggest that the using of intelligent systems in the control of mining and processing of minerals and in the diagnostics of equipment allows building high-quality control systems reducing the resource consumption of production.

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LOCALIZATION OF POLLUTION OF SOILS AND GROUNDWATERS WITHIN THE INFLUENCE OF THE TECHNOGENIC SWITCHES

Zhomyruk R.V.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Associate Professor, Department of Automation, Electrical Engineering and Computer-Integrated Technologies, Ukraine

Malanchuk E.Z.
National University of Water and Environmental Engineering (NUWEE), Doctor of Engineering, Professor,, Department of Automation, Electrical Engineering and Computer-Integrated Technologies, Ukraine

Stets S.E.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Associate Professor, Department of Automation, Electrical Engineering and Computer-Integrated Technologies, Ukraine

Khrystiuk A.O.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Associate Professor, Department of Automation, Electrical Engineering and Computer-Integrated Technologies, Ukraine

Summary

This scientific article contains research material to address the problem of the storage of man-made wastes and reduce their negative impact on the environment. The results of experimental and field studies of the storage area of technogenic phosphogypsum dumps and the impact of dumps on the environment are presented. Recommendations for environmentally sound storage and disposal of waste are given. The recommendations on the design of protective drainage for the interception of highly mineralized waters with the aim of preventing contamination of soil and groundwater in the area of technogenic dumps are given.

Introduction

Ensuring the ecologically safe existence of all natural objects is one of the major problems of our time. This provision is in no doubt, especially as the adequate solution of this issue and its subsequent implementation will provide not only comfortable living conditions and optimal sanitary and hygienic conditions of their production activities, but also the very existence of the biosphere. That is why environmental security is now regarded as an integral element of national security, which is very relevant for all countries of the world [1].

Ukraine is no exception in this regard, and environmental problems affecting all its regions are no less, and sometimes more acute, than in other countries of the world, as the current state of the environment in our country is beginning to take on negative properties, becoming immediate. a source of threat to the biosphere itself, as well as to the health and even life of its citizens. The main reason for this situation is all the same: excessive man-made load on natural objects without the proper level of environmental responsibility of those who control the development of production, which is increasing the current economic state of the state. Due to this, the rate of environmental degradation, and not only within the direct impact of specific enterprises, begins to exceed (and in some areas has already exceeded) the adaptive capacity of its components [2].

Given the scale and significance of the problem of industrial waste in Ukraine, there is a need to justify the parameters of localization of groundwater contamination and develop engineering methods aimed at improving the environmental situation and reducing the incidence of population within the impact of waste heaps of various industries.

Object and objectives of the study

The object of the research is phosphogypsum dumps near the production site of Rivneazot PJSC. They are a serious threat to the residents of Rivne and the surrounding villages, as well as the entire Goryn river basin. It is a waste of the fourth class of danger, which is represented mainly by phosphorus oxides accumulated in phosphogypsum dumps.

Laboratory studies of phosphogypsum have shown that in terms of dry matter, it contains 94% CaSO4, 1.8 undigested apatite, 1.8 phosphoric acid, 0.22 silicic acid, 1.92 insoluble residue, up to 1% iron and aluminum oxides. [3].

The dumps cover an area of 58 hectares and their total amount is 15.3 million tons [4]. The solution to the problem of industrial waste disposal in Rivne region depends to a large extent on how successfully this problem will be solved at this site.

For these environmentally hazardous dumps and most similar sites, it is common that the waste is stored outdoors. Due to the filtration of precipitation through this object, harmful substances get into the ground water, which leads to contamination of the soil adjacent to the object territory. For a long time such storage, not only dumps or waste heaps of mining, but also adjacent territory become environmentally dangerous. This factor should be taken into account when designing measures that will ensure environmentally-safe operation of open storage facilities for mining and other industries.

Therefore, finding ways to prevent contamination of soil and groundwater from the effects of phosphogypsum dumps, processing and disposal of phosphogypsum are important and urgent tasks.

Results of experimental and field studies.

The analysis of the object of study, which included topographic surveying, determination of chemical, mechanical composition of soil and groundwater, water-physical properties of soil of the adjacent territory was carried out in the work. [4, 5]. The level of morbidity of the population living in the territories adjacent to the waste heaps is analyzed [6]. As a result of the studies, the degree of soil and groundwater contamination was determined.

It has been confirmed that phosphogypsum dumps cause soil and groundwater contamination that spreads towards the river. This is evidenced by the nature of the distribution of pollutants in the soil: less value of the mass fraction of dry salt residue in the upper soil layers (depth 1-3 m) and greater value at the level of groundwater deposition (depth > 3 m) [5].

According to experimental studies, not only the phosphogypsum dumps themselves, but also the adjacent territory are the source of contamination. Due to the wind erosion of the dumps and the transfer of groundwater contamination, the soil contamination in the radius of 1 km around the dumps exceeded the limit values. Nitrate content in the soil exceeds the maximum permissible concentration and ranges from 30 to 90 mg/dm^3 (MPC - 45 mg/dm^3). The content of chromium in the soil is 3.0-6.0 mg/dm^3 (MPC - 1.0 mg/dm^3) [7]. The content of manganese in groundwater within the study area in all samples exceeds the MPC (0.1 mg/dm^3). The area with solid manganese content in groundwater of 50-300 mg/dm^3 is allocated directly on the territory of the landfill site. Iron content in groundwater - is in the range of 2000-1000 mg/kg (MPC - 0.3 mg/dm^3). In groundwater, petroleum products were detected and amounted to 4.3 mg/dm^3 (MPC - 0.3 mg/dm^3). The content of lead, zinc, copper, cadmium, nickel, cobalt, nitrite in groundwater is generally higher than the MPC [8, 9]. The results of chemical analysis of water samples taken from the site show the mineralization of water 8.3 mg/dm^3 [5].

Analyzing the results of the research, we can conclude that in the experimental site there is filtration of highly mineralized water from the territory of phosphogypsum dumps, which leads to contamination of soil, groundwater and waters of the Goryn river [10]. As a consequence, it has a negative impact on the health of the population: the incidence rate in the phosphogypsum waste heaps has increased by 30% compared to 1999 and 2019 [6].

Given the results of experimental and full-scale studies, finding ways to prevent contamination within the impact of man-made dumps is an extremely important task.

Recommendations for environmentally safe storage and disposal of phosphogypsum dumps

On the basis of full-scale, experimental researches and mathematical modeling, recommendations for environmentally safe storage and disposal of phosphogypsum dumps [13] have been developed, which include three stages:

engineering scheme for intercepting contaminated water from the territory of phosphogypsum dumps;

coating of phosphogypsum dumps with a protective polyethylene film, followed by powdering with a fertile soil layer and planting;

processing of phosphogypsum into building materials.

Engineering scheme for intercepting contaminated water from the territory of phosphogypsum dumps.

The scheme is presented in Fig. 1. In the area around the dumps it is proposed to have a collector-drainage network (1, 2) which will intercept and divert contaminated groundwater to treatment facilities (14) of Rivneazot PJSC, which are located at a distance of 5 km from the territory of the object of study. In order to intercept contaminated water coming from the territory of the phosphogypsum dumps, it is proposed to arrange hunting channels (3) along the perimeter of the object. From the feed channels, the solution is fed to the pump station (5) into the storage pool (7), which is located on the dumps themselves (4). The pumping station can operate in two modes: main and emergency. The main mode is the supply of water to the sump basin, the emergency mode is connected with the possible damage to the sump basin, then the contaminated water is supplied from hunting channels directly to the treatment facilities of Rivneazot PJSC.

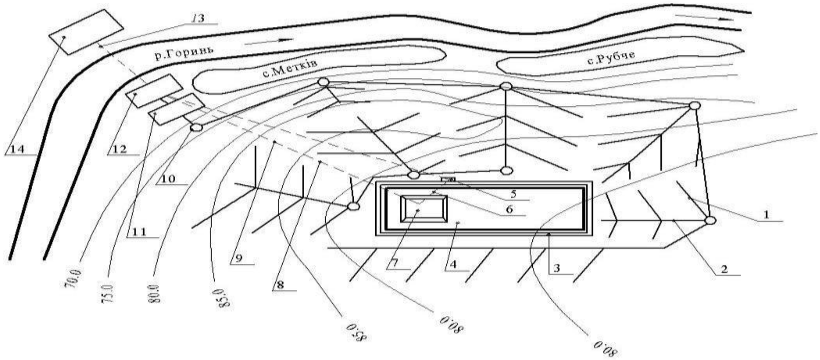

Fig. 1. Engineering network of contamination localization within the impact of phosphogypsum dumps: 1 - drains; 2 - collector; 3 - hunting channels; 4 - dumps of phosphogypsum; 5 - pumping station; 6, 13 - pressure pipeline; 7 - storage pool; 8, 9 - pressureless pipeline; 10 - drainage well; 11 - precast pool; 12 - pumping station; 14 - treatment plants

To calculate the parameters of the drainage-collector network for intercepting highly mineralized water coming from the phosphogypsum dumps, it was used by O. Oliynyk [14]. The basis of the calculation is a scheme of a bounded layer, which in the plan has one external rectilinear border (Fig. 2). To satisfy the boundary condition on the power circuit $H = H_1 = const$ at $X = 0$, when positioned on the right side at a distance L from the axis of the runoff, it is necessary to place a symmetric source of equal intensity q on the left distance from that axis at the same distance. Thus, in order to find a solution to this problem it is necessary to summarize the solutions for the drain and the source [15]. We limit ourselves to adding up the equation for a two-layer soil, given that by replacing the corresponding coefficients it is easy to get a solution for soils with more layers.

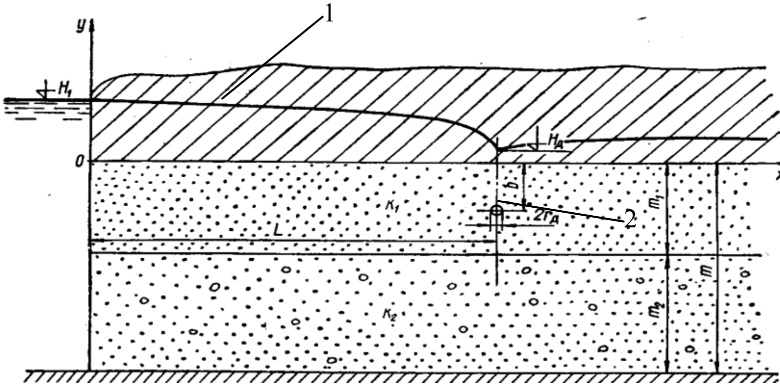

Fig. 2. The design scheme of unilateral drainage inflow: 1 - curve of depression; 2 - drainage

Since the solution of the problem for the runoff placed at the point with coordinates (*L, -b*) and having a flow *q*, is expressed by the equation.

$$H(x,y)=\frac{q}{4\pi r_1}\left\{\ln\left[(x-L)^2+(y+b)^2\right]+\ln\left[(x-L)^2+(y-b)^2\right]\right\}+$$
$$+\frac{q}{4\pi r_1}\sum_{n=1}^{\infty}c'_n\left\{\ln\left[(x-L)^2+(y+2nm_0-b)^2\right]+\ln\left[(x-L)^2+(y+2nm_0+b)^2\right]+\right. \quad (1)$$
$$\left.+\ln\left[(x-L)^2+(y-2nm_0+b)^2\right]+\ln\left[(x-L)^2+(y-2nm_0-b)^2\right]\right\}+C_1$$

where m_0 is the total largest divisor of m_1 and m_2

The solution of the problem for the source located at the point *(-L, -b*) and having a flow rate of -q will be as follows

$$H(x,y)=-\frac{q}{4\pi r_1}\left\{\ln\left[(x+L)^2+(y+b)^2\right]+\ln\left[(x+L)^2+(y-b)^2\right]\right\}-$$
$$-\frac{q}{4\pi r_1}\sum_{n=1}^{\infty}c'_n\left\{\ln\left[(x+L)^2+(y+2nm_0-b)^2\right]+\ln\left[(x+L)^2+(y+2nm_0+b)^2\right]+\right. \quad (2)$$
$$\left.+\ln\left[(x+L)^2+(y-2nm_0+b)^2\right]+\ln\left[(x+L)^2+(y-2nm_0-b)^2\right]\right\}+C$$

Summing up expressions (1) and (2), we find the final solution of the problem:

$$H(x,y)=\frac{q}{4\pi r_1}\{\ln\frac{\left[(x-L)^2+(y+b)^2\right]\left[(x-L)^2+(y-b)^2\right]}{\left[(x+L)^2+(y+b)^2\right]\left[(x+L)^2+(y-b)^2\right]}+$$
$$+\sum_{n=1}^{\infty}c'_n\ln\frac{\left[(x-L)^2+(y+2nm_0-b)^2\right]\left[(x-L)^2+(y+2nm_0+b)^2\right]}{\left[(x+L)^2+(y+2nm_0-b)^2\right]+\left[(x+L)^2+(y+2nm_0+b)^2\right]}\times \tag{3}$$
$$\times\frac{\left[(x-L)^2+(y-2nm_0+b)^2\right]\left[(x-L)^2+(y-2nm_0-b)^2\right]}{\left[(x+L)^2+(y-2nm_0+b)^2\right]\left[(x+L)^2+(y-2nm_0-b)^2\right]}\Bigg\}+C$$

In order to satisfy the condition $H = H_1$ at $X = 0$, the constant C must be taken to be equal to H_1. The pressure H_D.on the contour of the tubular drain with radius r_D is determined from the expression (3), substituting into it $x = L - r_D$ and $y = - b$:

$$H_{Д}=\frac{q}{4\pi r_1}\left\{\ln\frac{r_{Д}^2(r_{Д}^2+4b^2)}{(2L-r_{Д})^2\left[(2L-r_{Д})+4b^2\right]}\right.+$$
$$+\sum_{n=1}^{\infty}c'_n\ln\frac{\left[r_{Д}^2+4(nm_0-b)^2\right](r_{Д}^2+4n^2m_0^2)^2}{\left[(2L-r_{Д})^2+4(nm_0+b)^2\right]\left[(2L-r_{Д})^2+4n^2m_0^2\right]}\times \tag{4}$$
$$\times\frac{\left[(r_{Д}^2+4(nm_0+b)^2\right]}{\left[(2L-r_{Д})^2+4(nm_0+b)^2\right]}\;\}+H_{1.}.$$

From formula (4), given that $r_D < m_1$ and $r_D << L$, we obtain for the consumption $q = q_D$ per unit length of the drainage:

$$q_{Д}=4\pi r_1(H_1-H_{Д})\left\{\ln\frac{16L^2(L^2+b^2)}{r_{Д}^2\;(r_{Д}^2+4b^2)}+\right. \tag{5}$$
$$\left.+\sum_{n=1}^{\infty}c'_n\left[\ln(1+\frac{L^2}{(nm_0-b)^2}+2\ln(1+\frac{L^2}{n^2m_0^2})+\ln(1+\frac{L^2}{(nm_0+b)^2})\right]\right\}^{-1}.$$

In the paper using the theory of O.Oliynyk [16], some partial cases of drainage arrangement in two-layer soil are also considered, in particular at $r_2 = 0$ and $r_1 = r_2$.

For $r_2 = 0$

$$q_{Д}=\frac{r_1(H_1-H_{Д})}{\dfrac{L}{m_1}+\dfrac{1}{2\pi}\ln\dfrac{m_1}{2\pi r_{Д}\sin\dfrac{\pi(2b+r_{Д})}{2m_1}}}, \tag{6}$$

$$q_Д = \frac{r_1(H_1 - H_Д)}{\frac{L}{m_1} + \frac{1}{\pi}\ln\frac{m_1}{\pi r_Д}}. \tag{7}$$

To fully intercept the soil flow, a condition is required $H_Д^{/} = 0$, that is, the head over the drain was zero.

$$H_Д^{/} = H_Д - \frac{4\pi r_1\left(\frac{4\pi(L^{/} - L)}{m_0} + 2\ln\frac{m_0}{4\pi r_Д^{/}} - 2\ln\sin\pi B - 2\ln\frac{\sin\pi B}{4}\right)^{-1}}{q_Д^{/}} = 0; \tag{8}$$

For the n-th drainage, the dependence (8) can be rewritten

$$H_Д^{n} = H_Д^{n-1} - \frac{4\pi r_1\left(\frac{4\pi(L^{n} - L^{n-1})}{m_0} + 2\ln\frac{m_0}{4\pi r_Д^{n}} - 2\ln\sin\pi B - 2\ln\frac{\sin\pi B}{4}\right)^{-1}}{q_Д^{/}} = 0; \tag{9}$$

performing the necessary mathematical operations in equation (9) we obtain

$$H_Д^{/} = H_Д - \frac{4\pi r_1\left(\frac{4\pi(L^{/} - L)}{m_0} + 2\ln\frac{m_0}{4\pi r_Д^{/}} - 4\ln 2\right)^{-1}}{q_Д^{/}} = 0; \tag{10}$$

$$H_Д^{/} = H_Д - \frac{4\pi r_1\left(\frac{4\pi(L^{/} - L)}{m_0} + 2\ln\frac{m_0}{16\pi r_Д^{/}}\right)^{-1}}{q_Д^{/}} = 0; \tag{11}$$

$$H_Д^{n} = H_Д^{n-1} - \frac{4\pi r_1\left(\frac{4\pi(L^{n} - L^{n-1})}{m_0} + 2\ln\frac{m_0}{16\pi r_Д^{n}}\right)^{-1}}{q_Д^{n}} = 0; \tag{12}$$

where $q_Д^n$ - drainage intensity of the nth drainage, *m^3/s;*

$H_Д^n$ - head over nth drainage, *m*;

$r_Д^n$ - the radius of the cross-section of the nth drainage, *m*;

Ln -distance from the nth drainage to the power source, *m*;

bn -depth of the nth drainage, *m*.

Using the obtained dependencies, the necessary parameters for designing the drainage-collector network along the contour of the phosphogypsum dumps were calculated. The calculation scheme is presented in Fig. 3. For the complete interception of contaminated water drainage will be arranged on the sole of the waterproof layer. In order to prevent contamination of soil and groundwater, the pressure above the drained H_D must be equal to 0. In order to maximally intercept drainage water, the drainage radius should be such that it can pass the drainage flow.

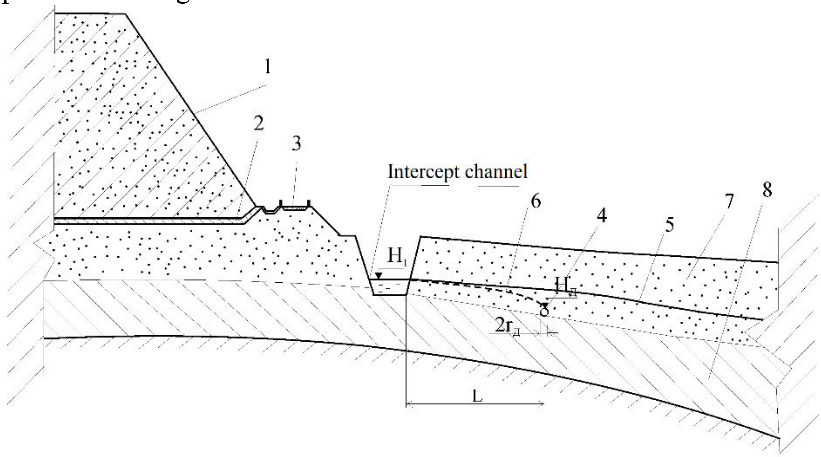

Fig. 3. Scheme for calculating the drainage line device for intercepting soil flow from the array of phosphogypsum dumps: 1 - phosphogypsum dumps; 2 - concrete curtain; 3 - the road; 4 - drainage; 5 - natural groundwater level; 6 - the level of groundwater after drainage; 7 - loops; 8 - loam

For more efficient interception of groundwater, horizontal drainage can be enhanced by vertical wells [17], which will be unloaded into drains (Fig. 4).

Strengthening of drainage by vertical wells is recommended when the soil mass from which contaminated water is removed is represented by soils with low filtration coefficient.

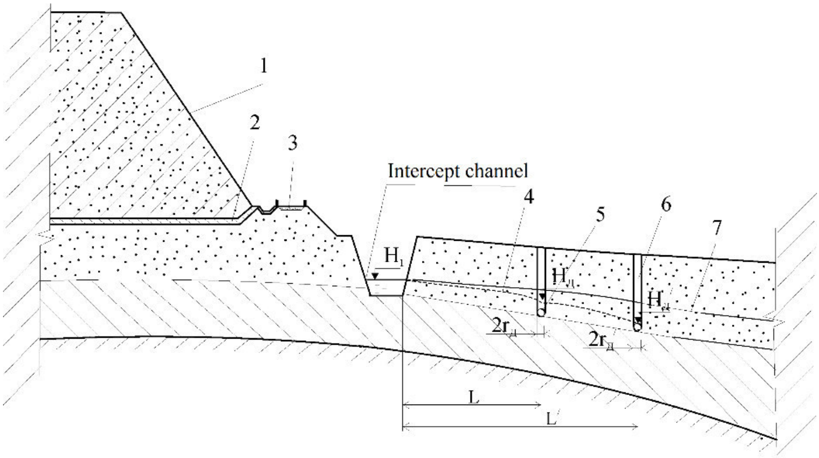

Fig. 4. The scheme of interception of contaminated waters by drainage with reinforced vertical wells: 1 - phosphogypsum dumps; 2 - concrete curtain; 3 - the road; 4 - the level of groundwater after drainage; 5 - drainage; 6 - well; 7 - natural groundwater level

In the construction of tubular drains, as one of the main and defining components of drainage systems, a particularly important step is the arrangement of drainage filters, which provide an increase in the flow of water and reliable protection against siltation. The reliability and durability of drainage systems is mainly determined by the quality of construction and the efficiency of drainage filters [17].

The systematic analysis of the latest research results and the current level of knowledge on the problem of sludge drainage protection indicate that one of the most effective and reliable drainage filters are bulk ready filters made of organic or a mixture of organic and synthetic fibrous materials (production wastes). The main advantages of them compared to thin-layer artificial are: a significant increase in the inflow of water to the drainage, the ability to maintain its protective properties longer due to the less susceptibility to the processes of colmatation (mechanical and hoarding) and the porosity regeneration property, cheapness, availability in sufficient quantity, technological availability , adaptability to soil, environmental friendliness and more.

Such protective and filtering materials include straw of cereals (wheat, rye, rice, barley, corn), peat, flax, sawdust, chips (especially effective coniferous), textile waste (fibers of cloth, carpet production mainly synthetic). All these materials can be rationally used to protect the drainage from mud.

Coating of phosphogypsum dumps with a protective plastic film, followed by powdering with a fertile soil layer and planting

In the second stage, it is recommended to cover the phosphogypsum dumps with a protective film, followed by powdering the fertile soil layer with the landing of vegetation, which will prevent wind erosion and contamination of adjacent territories [19].

Polymer films as a material for anti-filtration devices of hydraulic structures have several advantages: practical impermeability of the material itself and high resistance of polyethylene and polyvinyl chloride films to aggressive influence of common chemical reagents, high deformative ability of films and high material capacity. The advantages of polymer films should also be attributed to the fact that the formation of water retaining elements in them depends little on the local conditions of construction.

Film anti-filtration devices are reliable in operation throughout the life of the structure. The reliability of operation is determined primarily by the properties of the polymer film element. These properties should provide protection against impacts that are possible during both construction and operational periods.

In our case, not only punctures, cuts, but also changes that would lead to a violation of the integrity of the films and, consequently, a loss of water tightness in the term shorter than the service life of the structure, are inadmissible in film anti-filtration elements.

In the absence of mechanical damage in the film due to its low porosity, water movement through the film is possible only in the form of diffusion of water molecules and soluble substances in it. Diffusion water losses, however, are quite small and in our case acceptable. Soil life of up to 100 years of soil coverage. The vegetation that is recommended for planting on a meter layer of soil - small bushes with sowing grasses.

Recycling phosphogypsum into building materials

The third stage is designed for a more long-term perspective - the processing of phosphogypsum into building materials [20] (wall blocks, overlapping panels, binders with a strength of more than 500 concrete) with simultaneous decontamination of phosphogypsum from harmful elements and removal of rare-earth metals composition of phosphogypsum - up to 1%. Prospects of such works have been proved by both Ukrainian scientists and scientists of other countries. There are currently several phosphogypsum processing technologies for building materials. According to preliminary calculations, the profitability of such production is 5.7 times higher than the standard profitability.

Conclusions

The result of long-term storage of waste from the production of mineral fertilizers at PJSC "Rivneazot" is the formation of technogenic waste heaps of phosphogypsum, which amount to 15.2 million tons and require a constant increase in the removal of storage areas.

Toxic waste is stored in the dumps, which leads to contamination of soil, surface and groundwater, adversely affecting the health of the population. The main factor that causes soil contamination in the territory of phosphogypsum dumps is the filtration of contaminated water. This is evidenced by soil contamination, which is below the level of groundwater 5 to 10 times higher than on the surface. And over time, this figure increases.

Experimental and field studies have shown that the content of nitrates, chromium, manganese, iron, lead, zinc, copper, cadmium, nickel, cobalt, nitrites and petroleum products in soil and groundwater within the study area exceeds the concentration in all samples.

Recommendations for environmentally safe storage and disposal of phosphogypsum dumps are developed, which include three steps:

- engineering scheme for intercepting contaminated water from the territory of phosphogypsum dumps;
- coating of phosphogypsum dumps with a protective polyethylene film, followed by powdering with a fertile soil layer and planting of vegetation;
- processing of phosphogypsum into building materials.

The paper proposes a method of calculating the drainage-collector network for intercepting highly mineralized water coming from man-made dumps. But the recommendations on the arrangement of the engineering network depending on the soil conditions of the territory of storage of waste of mining or other production are given.

The complex of measures considered allows to solve the problem of coordination of relations between production, on the one hand, and nature - on the other, will allow to reduce excessive technogenic load on natural objects, to provide protection of soils, ground and surface waters from the receipt of pollution, and will allow to get economic effect from introduction of waste processing technologies.

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MINING OF UNDERGROUND DEPOSITS IN DIFFICULT GEOLOGICAL CONDITIONS

Serhii Pysmennyi
Kryvyi Rih National University,
Ph.D. (Engineering), Associate Professor, Ukraine

Kanay Rysbekov
Institute of Geology, Petroleum and Mining engineering
Satbayev University, PhD (Engineering), Associate professor,
Kazakhstan

Serhii Chukharev
National University of Water and Environmental Engineering
(NUWEE), Ph.D. (Engineering), Associate Professor, Ukraine

Branko Gluščević,
University of Belgrade, faculty of Mining and Geology,
PhD (Engineering), Associate Professor, Serbia

Abstract

The article studies stability of intervening pillars at underground selective mining of complex structured ore bodies of Kryvyi Rih iron ore basin applying open stope systems that will enable the increased useful mineral component content in the mined ore mass. When calculating an exposure span, the current methods of determining room system constructive elements do not consider dirt thickness. So, it is essential to develop methods of determining room system constructive elements for mining complex structured ore bodies to provide stability of stope exposures. When mining a mine block, stoping is suggested to be fulfilled gradually from the footwall to the hanging wall of the ore body by room-and-pillar methods leaving a dirt or ore inclusion in the block. This enables decrease in concentrating tension and compression stresses in the middle part of the dirt or ore inclusion resulting in its 1.5 – 2.0 times greater stability. There is also determined that the stope stability is impacted by the horizontal thickness of the inclusion, the hardness ratio and the order of stoping in a mine block. Application of open stope methods in selective mining of complex structured ore bodies of Kryvyi Rih iron ore basin enables increased quality of the mined ore mass without significant capital and operating costs and results in better environment of the region.

Introduction

Kryvyi Rih iron ore basin is represented by thick deposits of rich and lean ores with dirt or ore inclusions (DOI) of 2 to 15 m [1-3]. The Protodiakonov hardness ratio of DOI is 8-6, that is on average by 4-6 units higher than that of the main massif. DOI in rich ores make nearly 300 mln t of which up to 800 thousand t are mined annually [4-6].

Underground mining of complex structured ore bodies (CSOB) is conducted by bulk-caving and open stope methods resulting in 3-6% less iron content in the mined ore and 1.2-1.5 times greater than standard ore losses. Thus, the amount of the mined ore with the iron content of over 62% does not exceed 60% that leads to loss of the world sales market.

To increase sales, mining enterprises apply resource-saving technologies enabling increase of volumes of mined ore with the iron content of 60% to 80%. This is achieved through use of a complex approach considering the concept of the environmental-economic system management and taking into account availability of functioning elements applying the organizational-technical control theory. The results of studying ore raw material mining and processing enable concluding that concentration efficiency indicators depend greatly on operational information and current technological processes. In most cases, when developing nondestructive control methods, electromagnetic, ultrasonic and radiometric techniques are used. However, all these approaches result in not only greater mining costs but also deterioration of the regional environment [7–10].

To increase mining of non-diluted ore at Kiruna mine (Sweden), selective mining with sorting ore mass underground and its transporting by main shafts is used. To support main openings, the mine applies modern methods of controlling the state of the rock massif disturbed by underground mining. It should be noted that their implementation at Kryvbas underground mines will lead to deterioration of labour conditions and increased mining costs due to complicated mining, geological and technical conditions [11-15].

Development of complex structured deposits is highlighted in many investigations dealing with determining regulations of extraction indicators, manifestations of rock pressure, sequence of mining and determining parameters of basic constructive elements of mining systems [16-18]. Efficiency of mining CSOB is proved to be impacted by mining geological and technical conditions, as well as by the order of stoping, rock pressure, mining intensity, number and stability of pillars, level height, mutual arrangement of main strike stopes and pillars.

When applying room-and-pillar methods, the iron content in the mined ore mass increases as compared with bulk caving systems but conditions of further mining deteriorate due to changes in rock massif stresses caused by concurrent appearance of tension and compression stresses [19-21].

Methods. According to [22-24], to provide stability of stopes considering changes of stress concentration, it is practicable to apply open stope methods taking into account physical and mechanical properties of DOI and sequence of mining a block.

To settle the problem of iron content increase and ore loss decrease when mining deposits represented by complex structured ore bodies, it is reasonable to apply the method suggested in [25-28]. The authors suggest modernization of traditional concentrating through hydrometallurgical and chemical processing that enhances efficiency of concentration for the account of using other energy kinds. This direction of modernization is based on processes of force impacts on a substance during disintegration in the activator and does not consider processes connected with mining useful minerals by underground methods.

The critical analysis of works dealing with mining and processing useful minerals enables the following conclusions:

1. Most authors suggest increase of the iron content in the mined ore at the expense of building an underground or surface mining and concentrating complex that will result in increased mining costs and lost world markets.

2. Increase of the iron ore content in the mined ore at the first stage may be achieved through applying resource-saving selective mining aimed at excluding the concentration process. At that, mining of ore bodies with horizontal thickness of dirt or ore inclusions of less than 12 m is suggested to be conducted by the traditional underground method involving concentrating combines.

Research tasks. Thus, when mining CSOB, the necessity arises to enhance the resource-saving technology which provides the iron content increase at selective mining of deposits depending on mining systems applied. That is why, it is essential to determine impacts of dirt inclusion thickness and a mining system on the mined ore mass volumes.

Material and results of research

To increase mined ore volumes that directly impact economic indicators, it is reasonable to mine complex structured ore bodies by the selective open stope method according to the developed classification [4,18,28]. Application of the selective open stope system enables significant enhancement of quality of the useful component in the mined ore as compared with the bulk caving system and decrease of mining costs when applying the mining-and filling system.

The suggested mining open stope system provides for a certain sequence of mining operations depending on mining and geological conditions of CSOB. Mining within a mine unit is performed from the hanging wall to the footwall and consists of two stages, Fig. 1: *stage I* - ore is mined from the hanging wall with the dirt inclusion left in the stope as a pillar; *stage II* - the remaining ore is mined from the footwall depending on the order and sequence of mining operations.

To obtain high values of ore mass extraction from mining by open stope systems, it is necessary to provide stability of exposures and pillars and, consequently, stability of DOI for the whole period of mining the stope of SCOB. That is why, when mining complex structured ore bodies by open stope systems, it is essential to specify constructive elements of the stope (which are not yet completely defined [29-30]) and substantiate the minimum permissible thickness of the dirt or ore inclusion that will provide stability of stopes and the ore body.

When applying stope systems depending on the stage and order of mining operations in a mine block, there occur different loads. Depending on the load action on DOI, a field of tension or compression stresses is formed in it.

Further deeper mining of reserves of rich iron ores results in deterioration of mining and geological conditions and, consequently, caving of the hanging wall ores at exposure creation. In such conditions it is reasonable to change the procedure of mining blocks with development of mining operations from the footwall to the hanging wall.

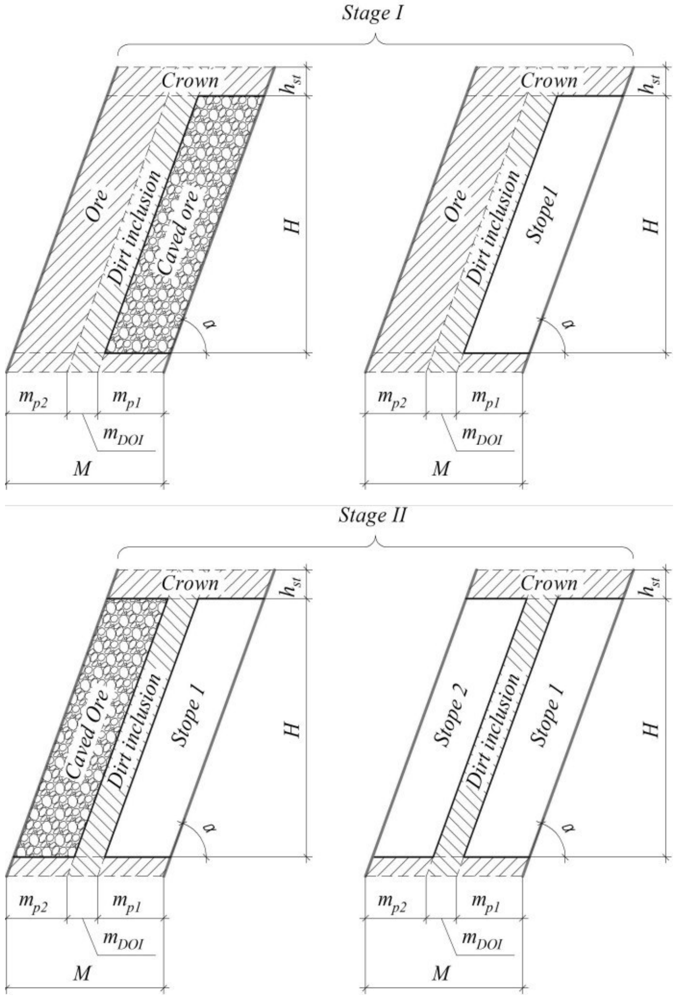

Fig. 1. The basic diagram of mining complex structured ore bodies in stable ores by open stope systems

Mining a block represented by CSOB from the footwall to the hanging wall changes the character of stresses and, consequently, produces a significant impact on DOI stability, Fig. 2.

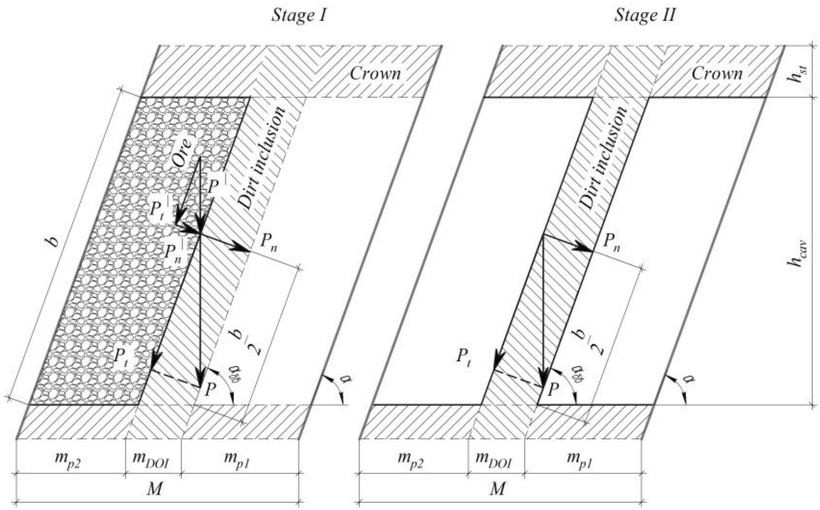

Fig. 2. The computational diagram for determining constructive elements of the room-and-pillar mining method when mining a stope according to *option 2*

Thus, at *stage I* after the exposure at the footwall is created (stope 1), on the contour of DOI there occur tension stresses. After breaking the ore massif and its further drawing from stope 2 (see Fig. 1), the tension stresses on the contour of DOI with stope 1 decrease and there appear compression stresses on the other side of DOI.

Maximum tension stresses on the contour of DOI and stope 1 will appear after stope 2 caving. That is why, when determining the permissible length of the exposure, it is essential to take into account weight of the caved rocks that will impact tensile stresses on the DOI – stope 1 contour. Fig. 3 demonstrates that DOI is impacted by the caved ore in the footwall triangle.

Maximum stresses occurring in the pillar presented as sandwiched beam impacted by the caved ore weight are calculated by the expression [5, 29]

$$\sigma_{max} = \frac{6 \cdot \left(M_x + M_r\right)}{l_{DOI} \cdot m_{DOI}^2}, \tag{1}$$

where σ_x is maximum stresses, t/m; M_x is the maximum bending moment value at part z of the DOI exposure length along the axis x, t/m; M_r is the maximum bending moment caused by caved rocks, t/m; l_{DOI}

is the maximum permissible length of the dirt or ore inclusion exposure, m; m_{DOI} is normal thickness of the dirt or ore inclusion, m.

The maximum bending moment of the triangular shape caused by caved rocks is calculated by the formula

$$M_r = A \cdot z + M_a - \frac{q_r \cdot z^2}{6} = \frac{3 \cdot q_r \cdot l_r}{20} \cdot \frac{l_r}{2} - \frac{q_r \cdot l_r^2}{30} - \frac{q_r \cdot l_r^2}{24}, \qquad (2)$$

where q_r is weight of the caved ore in the triangle of the footwall per 1 m of DOI length, t/m^2; l_r is the inclined length of the caved rocks at the footwall, m.

After formula (2) is simplified, the following expression for determining the maximum bending moment is obtained

$$M_r = \frac{2 \cdot q_r \cdot l_r^2}{3}. \qquad (3)$$

Maximum stresses occurring in DOI and calculated according to formula (1) considering the maximum bending moment of the caved ore (3) are determined by the expression

$$\sigma_{max} = \frac{6\left(\frac{q \cdot l_{BOI}^2}{24} + \frac{2 \cdot q_r \cdot l_r^2}{3}\right)}{l_{DOI} \cdot m_{DOI}^2} = \frac{q \cdot l_{BOI}^2 + 16 \cdot q_r \cdot l_r^2}{4 \cdot l_{DOI} \cdot m_{DOI}^2}. \qquad (4)$$

It should be taken into account that the length of the dirt or ore inclusion equals the inclined length of the caved ore, so expression (4) will look like

$$\sigma_{max} = \frac{q \cdot l_{DOI} + 16 \cdot q_r \cdot l_{DOI}}{4 \cdot m_{DOI}^2}. \qquad (5)$$

The maximum permissible length of the DOI exposure presented as a beam considering the maximum bending moment of the caved ore weight is determined by the expression

$$l_{BOI} = \frac{4 \cdot \sigma_{max} \cdot m_{DOI}^2}{q + 16 \cdot q_r} = \frac{4 \cdot K_f \cdot f \cdot K_{str.w} \cdot m_{DOI}^2}{\left(a_{II} \cdot m_{DOI} \cdot \gamma_{DOI} + 16 \cdot q_r\right) \cdot K_{st}}, \qquad (6)$$

where K_f is the conversion rock hardness into stress ratio; f is the hardness ratio of rocks of the dirt or ore inclusion (Protodyakonov scale); $K_{str.w}$ is the ratio of structural rock weakening caused by fractures (accepted from 0.65 to 0.95); K_{st} is the rock hardness ratio (accepted 1.5-2.0).

The caved ore weight in the triangle of the footwall per 1 m of DOI length

$$q_r = \frac{h_{st} \cdot в \cdot \gamma_o}{2 \cdot K_r} = \frac{h_{st}^2 \cdot \gamma_o \cdot tg\alpha}{2 \cdot K_r} = \frac{h_{st} \cdot \gamma_o \cdot sin^2 \alpha}{2 \cdot K_r \cdot cos\,\alpha}, \qquad (7)$$

where h_{st} is the height of the caved ore layer (height of a level or sublevel), m; $в$ is the caved ore width in the upper part of the stope of stage II of mining, m; γ_o is the volume weight of ore, t/m^3; K_r is the primary ore loosening factor; α is the dirt or ore inclusion dip angle, degrees.

On inserting (7) into (8) we obtain

$$l_{DOI} = \frac{4 \cdot m_{DOI}^2 \cdot K_f \cdot f \cdot K \cdot K_r \cdot cos\,\alpha}{\left(a_{II} \cdot m_{DOI} \cdot \gamma_{DOI} \cdot K_r \cdot cos\,\alpha + 8 \cdot h_{st} \cdot \gamma_o \cdot sin^2 \alpha\right) \cdot K_{st}}. \qquad (8)$$

The results of the calculations enable building dependencies of the exposure span of the dirt or ore inclusion (Fig. 3) and width of the stope of stage II of mining.

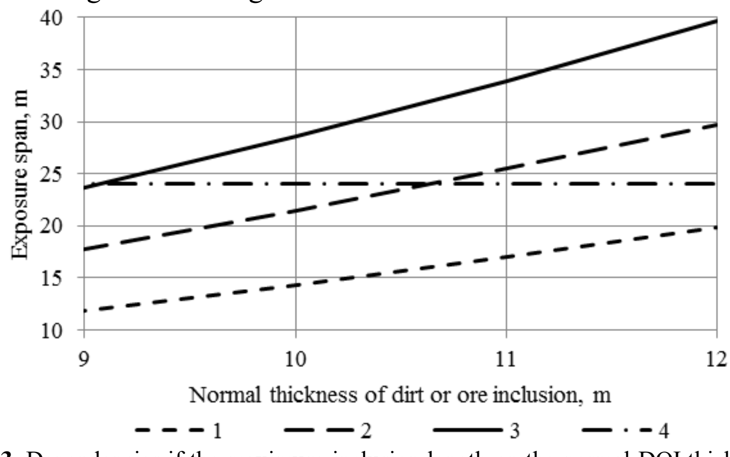

Fig. 3. Dependencies if the maximum inclusion length on the normal DOI thickness and hardness at the width of the stage II stope of 25 m (option 2): 1-5 is the at DOI hardness of respectively 8, 12, 16; 6 is the permissible length (according to ROMI)

The dependencies in Fig. 3 demonstrate that increase of thickness of DOI from 9 to 12 m with its hardness of 12 (Protodiakonov scale) causes increase of the exposure length from 12 to 40 m. It should be noted that, according to the technique developed by the Research Ore Mining Institute (ROMI), the maximum stable length of an exposure for given design conditions should not exceed 24 m. thus, according to the calculations performed, if a stope is created with the exposure span of 24 m and with DOI hardness of 8, it will be destroyed regardless of the inclusion thickness.

The diagrams in Fig. 4 demonstrate that increase of the width of the stope along the strike from 25 to 60 m at normal thickness of the dirt or ore inclusion of 10 m decreases the maximum span of the exposure from 36 to 15 m with the DOI decreased hardness from 16 to 8.

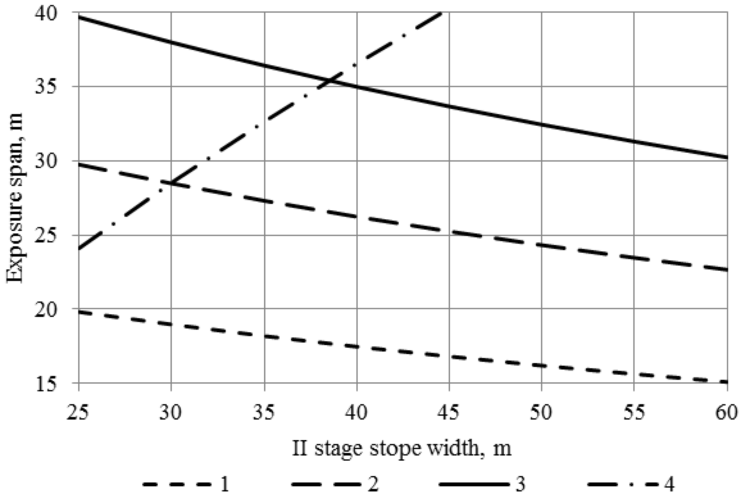

Fig. 4. Dependencies of the maximum inclusion length on the stope width and thickness of DOI rocks at the normal thickness of the dirt or ore inclusion of 10 m (option 2): 1-5 is the at DOI hardness of respectively 8, 10, 12, 14, 16; 6 is the permissible length (according to ROMI)

ANSIS-16.0 - based mathematical model of the change of fields of equivalent stresses in the rock massif around stopes at different stages of mining operations when mining the deposit from the footwall to the hanging wall is given in Fig. 5: initial – without mining operations; intermediate – formation of a stope in the footwall; final - formation of a stope in the hanging wall.

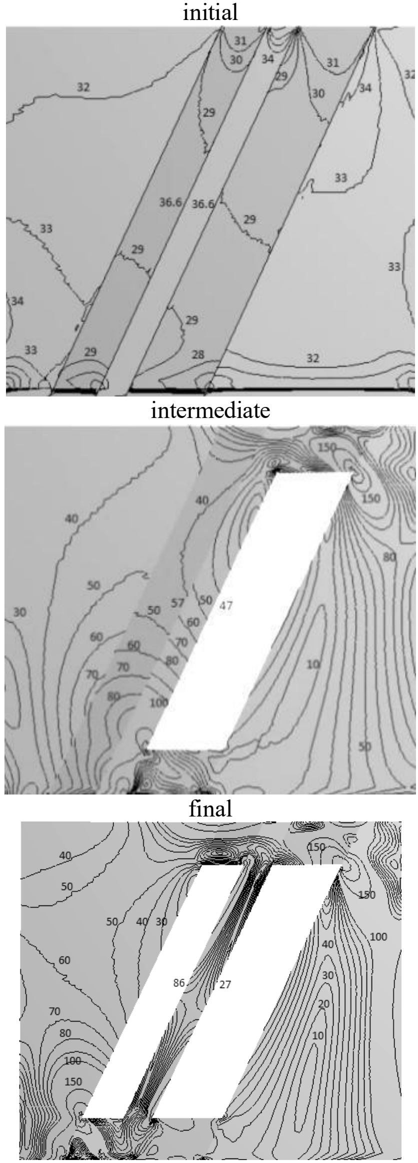

Fig. 5. The results of mining CSOB from the hanging wall at the compressive strength of the dirt inclusion of 160 MPa

Model studies registered a field of equivalent stresses in the massif around stopes and in the middle of the dirt inclusion at different stages of mining operations (Fig. 1, option 2). There were conducted 9 series of studies that differed from each other by physical and mechanical properties of the rock massif and DOI, other parameters (mining depth, level height, thickness) remained unchanged. The results of the studies enable building dependencies of changes of equivalent stresses in the middle part of the dirt inclusion, Fig. 6.

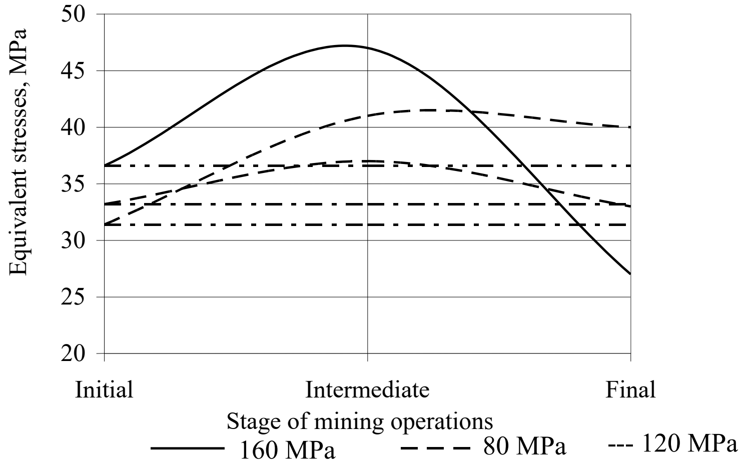

Fig. 6. Dependencies of equivalent stresses changes in the middle part of the dirt inclusion on the stage of mining operations, ultimate strength of DOI rock at ultimate strength of ore of 90 MPa and the level height of 90 m

The diagrams in Fig. 6 show that the character of equivalent stresses changes in DOI in its middle part is only impacted by the order of mining operations in a mine block. At that, further loading beyond the yield point increases the proportional limit according to Hooke's law [4, 5].

Thus, when the compressive strength is greater than 120 MPa (rock hardness of 12, Protodiakonov scale), the I and II stage stopes will be stable. It should be noted that mathematical modeling cannot practically reproduce the rock massif and that is why the stope exposure length should be decreased by the stability safety factor.

Conclusions

According to the study results, selective mining of complex structured ore bodies enables increasing the iron content in the mined ore mass. However, high indicators of mining can be achieved when applying certain mining systems in current mining and geological conditions. Ore bodies containing dirt inclusions of 4 to 12 m thick should be mined by mining systems used at underground mines of Kryvyi Rih basin.

Modeling results enable determining that at the level heights of 90 m (the span of the inclined exposure of 58.9 m) and the stope width of 25 m at DOI ultimate strength less than 120 MPa the exposure remains stable. When the ultimate strength exceeds 120 MPa, the exposure span will be instable, DOI and stopes will be destroyed.

Thus, application of selective mining methods when mining stopes presented by CSOB will enable not only enhancing mined ore mass volumes and, consequently, retaining the world market but also decreasing mining and processing costs.

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RATIONALE OF THE STRATEGIC DEVELOPMENT OF THE ENERGY INDUSTRY OF UKRAINE WITH USE OF RESOURCE-SAVING TECHNOLOGIES

Kushnir N.B.
National University of Water and Environmental Engineering (NUWEE), Candidate of Sciences (Economics), Professor, Department of Enterprise Economics, Ukraine

Moshchych S.Z.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Senior Lecturer, Institute of Postgraduate Education, Ukraine

Veretin L.S.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Senior Lecturer, Institute of Postgraduate Education, Ukraine

Okseniuk R.R.
National University of Water and Environmental Engineering (NUWEE), Assistant, Department of Development of Deposits and Mining, Ukraine

Summary

Energy is an important factor that has a direct impact on industrial relations, social development of society, the state of the environment. The scarcity of energy resources requires assessing them as a factor of economic security and taking them into account when formulating national and entrepreneurial strategies. The concept of energy security as the availability of energy for industrial and domestic consumption has become widespread in scientific works. At the same time, the driving force of any process is a strategic vision that builds a system of goals and develops concrete measures aimed at overcoming problem areas in the trajectory of achieving effective solutions.

The modern post-industrial period needs a new stage of energy development - ecologically regulated, during which the organizational and economic mechanism should be formed as a result of ecological regulation. This will ensure the transition to sustainable development, characterized by economic and social progress while reducing the man-made impact of energy on the environment, which meets the needs of the knowledge economy.

Introduction

Nature has provided mankind with an inexhaustible supply of energy, and taming and transforming it into forms suitable for use have given human civilization over the past centuries an unprecedented advance in material production and, consequently, in the spiritual development of society. Improvements in technology and growth in production are helping to better meet people's needs, but this process is contributing to increasing environmental pollution. The economic consequences of numerous natural disasters and man-made disasters, global epidemics are growing to alarming proportions, which, in turn, slows down both the development of the world economy as a whole and the solution of demographic, energy and food problems.

The issues of environmental efficiency in general and energy in particular are being investigated by scientists O. O. Veklych, T.P. Galushkina, O.D. Gnatkovich, L.S. Grinov, P.M. Gritsyuk, V.S. Kravtsiv, L.I. Maksimov, L.G. Melnik, O. Yu. Popova, S.M. Rogach, V.V. Sabadash, P.M. Skripchuk, A.Ya. Sokhnich, Y. Yu. Tunic, M.A. Khvesik, E. V. Khlobistov, L. M. Cherchik, R. Costanza, H. Daly, J. Dixon, J. Farley, R. de Groot, N. Hanley , J. Krutilla, D. Pearce and others. In recent years, considerable work has been done in Ukraine to address both scientific, technical and organizational and economic problems of improving energy supply. In particular, the directions of increasing energy efficiency were explored in their works by A. Borisenko, S. Denisyuk, D. Deregan, M. Kovalko, E. Krykavsky, M. Ksenofontov, V. Kuratchenko and others.

The interaction of human society and nature in the process of development of energy sources was not optimal at all stages of development. The rapid increase in energy consumption creates the risk of rapid depletion of fuel and energy resources. In addition, the dynamics of global energy consumption in the last century sufficiently indicate its exponential growth, which is a potential threat of an increase in excess heat emissions that can disrupt the thermal balance of the planet and lead to catastrophic changes in its climate. Therefore, the development of national energy using the methods of environmental regulation will contribute to the sustainable development of the fuel and energy complex.

The purpose of the work is to develop theoretical and methodological approaches for the strategic development of national energy, taking into account the requirements of the European Community and to provide scientifically sound recommendations for its improvement, taking into account environmental, economic and social aspects.

In order to achieve this goal, it is envisaged to fulfill such tasks, namely to identify the types of energy development by the intensity of impact on the environment; identify trends in national energy; analyze the conditions for achieving the strategic priorities of national economy development; to develop scientific and practical recommendations for the development of the national energy complex to the requirements of the European Energy Community.

1. Current state of energy efficiency of Ukraine

Energy development has a decisive influence on the state of the economy in the country and the standard of living of the population. The aim of the welfare state, which, according to the Constitution, is Ukraine, is to provide conditions for increasing the well-being of citizens. One of the most important components of well-being in civilized states is to provide citizens and companies with the necessary energy resources. The key to achieving this goal should be a reliable, economically sound and environmentally sound meeting the needs of the population and the economy in energy products.

Ukraine is one of the energy scarce countries in the supply of basic primary energy, which necessitates significant volumes of their imports. The issue of Ukraine's energy independence has been particularly aggravated with the onset of Russian hybrid aggression against our country. Aggressive economic activities, the constant blackmailing of Russian state-owned corporations, and the threat of disruption of energy supplies from Russia (natural gas, nuclear fuel, oil and petroleum products) have forced us to look fundamentally at the issue of energy independence.

A key challenge that needs urgent response is the unsatisfactory technical condition of Ukraine's energy sector, which continues to deteriorate as a result of the aging of fixed assets. Most of the generating assets and energy networks are worn out and inefficient. The vast majority of thermal power plant units has exceeded the physical wear and tear and requires major upgrading or replacement; most nuclear power plant units are nearing the design lifetime. The power balance of Ukraine's power grid is characterized by a lack of regulatory capacity, which causes irrational use of existing capacity and a high level of losses. Emissions of dust, sulfur oxides and nitrogen by thermal power plants of Ukraine are several times higher than the corresponding norms of developed countries. A large proportion of the backbone and distribution network facilities has fulfilled their resources and is in need of modernization.

Inefficient use of energy resources, heat loss in buildings that do not have thermal insulation and electricity in the grids, energy consumption is turning a huge loss for Ukraine and affects the level of national energy security, economic development, cost of production, ecology, as well as elementary, the cost of living for citizens. According to experts, only inefficient energy consumption for heating residential and public buildings in Ukraine generates losses of $ 3 billion annually (or 3% of national GDP). Equally large losses are causing the country and the use of outdated and inefficient technologies in industry.

Today, with a high level of import dependency, Ukraine is the most energy-intensive country in Europe. Low efficiency of use of fuel and energy resources causes high level of GDP energy intensity in Ukraine. By international standards, Ukraine's economy is one of the most energy-intensive economies in the world due to its large share of energy-intensive sectors, outdated and inefficient technologies, extremely worn-out fixed assets, inefficient energy transformation and supply systems, and energy-efficient building stock.

Instead of providing the extensive development that Ukraine's economy has been moving through for decades, the energy sector needs to move to effectively ensure sustainable economic development. Maintenance of economy and social sphere of the country with the main types of energy sources (electric and thermal energy, motor and boiler-fuel types, as well as natural gas) and raw materials for the needs of the chemical and metallurgical industry (coking coal, oil and gas products), is entrusted to complex energy sources of Ukraine.

2. Strategic directions of national energy development

Diversification of energy sources, enhancement of Ukraine's transit potential and development of alternative energy all require a sound energy infrastructure development policy.

However, Ukraine's state energy policy is characterized by a mismatch of economic and political interests at different levels of government. In the face of political instability, the role and sensitivity to the threats to the fuel and energy complex as a guarantor of the country's economic security is increasing. Along with other components of economic security, energy security forms the basis of economic relations, creating the conditions for GDP production.

The forecast of socio-economic development of Ukraine (ensuring GDP growth of Ukraine twice by 2035) is based on the need for radical technological renewal of the national economy and social sphere, implementation of corresponding large-scale investment projects. In this case, the forecast of economic development proceeds from the need to maximize the use of internal potential in the realization of these ambitious goals, which forms the prospect of loading the metallurgical, construction, transport and energy sectors of the national economy.

The dynamics of the development of individual sectors of the national economy and the structure of GDP formation are shown in Figs. 1 and fig. 2. The structure of output is shown in fig. 3.

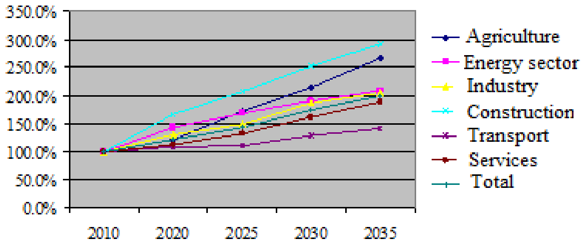

Fig. 1. Forecast of the dynamics of GDP formation by individual sectors of the national economy

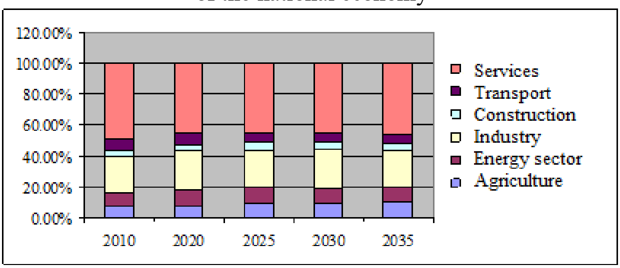

Fig. 2. Structure of GDP formation by sectors of national economy

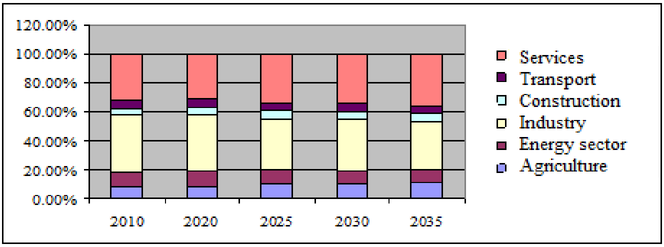

Fig. 3. Structure of output by sectors of the national economy

At the same time, Ukraine's international commitments and sustainable development goals impose on the national economy and the energy sector a number of restrictions on the need for innovative renewal of the energy sector, expansion of the use of renewable energy sources, reduction of energy intensity of the national economy, environmental impact, etc. The overall forecast energy balance of Ukraine for the period up to 2035, taking into account these limitations, is given in Table 2.

Table 2

Forecast balance of consumption of fuel and energy resources
for the period up to 2035

FER consumption, ppm	2020	2025	2030	2035
Coal	37.69	38.37	37.27	33.78
Natural gas	37.33	33.57	33.20	34.17
Petroleum products	13.97	14.86	15.74	16.48
Nuclear energy	25.31	25.38	27.39	32.86
Biomass, biofuels and waste	6.38	8.91	11.85	13.10
Solar energy	0.37	0.56	0.70	0.84
Wind energy	0.21	0.32	0.43	0.54
Hydraulic energy*	0.93	1.02	1.21	1.25
Environmental energy	0.78	1.42	1.86	2.40
Net exports	-1.03	-1.29	-2.15	-2.58
Total, incl.	121.92	123.12	127.49	132.84
not energy use	4.72	4.96	5.16	5.31
energy use	117.20	118.17	122.33	127.53
including RES	8.66	12.23	16.05	18.12
Share of RES in consumption of primary resources,%	7.4	10.3	13.1	14.2
GDP, billion USD United States (PCA 2005)	457	540	638	761
Energy consumption, kg AD / thousand US dollars USA	0.27	0.23	0.20	0.17
Final consumption	78.89	80.84	85.13	88.91
Share of RES in gross final consumption,%	11.0	15.1	18.9	20.4

* the forecast for 2020-2035 does not take into account the energy received from the HPP
** IEA data

Priorities for the development of the electric power industry will be optimization of the structure of generating capacities, taking into account the peculiarities of attraction to the energy balance of renewable energy and development of power supply networks, which implies the reduction of transformation rates and the approximation of high-voltage networks to the consumer, increasing the flexibility of the system through the implementation of network principles.

The development of thermal generation will be determined by the availability and cost of coal and natural gas reserves, as well as the development of energy technologies for the use of biomass and other fuels. Due to the projected high cost of natural gas conservation, a significant proportion of coal generation is expected to be saved. At the same time, the requirements for limiting the negative impact of energy on the environment will require a large-scale modernization and renewal of thermal power generation capacities, which will lead to a relative stabilization of the total installed capacity of coal TPPs at the existing level.

Of course, increasing energy independence is impossible without modernizing the national energy infrastructure, attracting new modern efficient and energy-saving technologies. Increasing domestic and foreign investment in energy is the most effective solution to this problem. For this purpose it is necessary to work on demonopolization of the energy sector, development of competition, implementation of European norms and market principles of work in the energy sector. A transparent and fair energy market with clear rules of the game is the key to the progressive development of Ukrainian energy.

The development of renewable (or green) energy is not only a popular global energy trend these days; it is also of great global importance. The fossil fuels that underlie energy production in the world today have limited reserves that will sooner or later be depleted. Therefore, not only Ukraine but also all countries that care about their future are tasked with balancing electricity production so that they do not depend solely on temporarily available resources.

For Ukraine, active use of renewable energy sources is especially important, as our country imports part of the traditional fuel and energy resources. Thus, an increase in the share of “green energy” fits into the overall national strategy for diversification of energy sources and will help to strengthen energy independence.

3. Innovative model of development of fuel and energy complex of Ukraine

Our research has confirmed the systemic crisis of the fuel and energy complex, the development of which, if ignored, will threaten the entire economic complex of Ukraine and the existence of the state as a whole. As a variant for overcoming this crisis in the environment of scientists, specialists, experts, in recent years, a consolidated opinion was developed about the need for restructuring of the FEC of Ukraine by applying an innovative model of its development.

It should be noted that today the innovative direction of development is the basic strategy for business, where knowledge together with social capital creates competitive advantages of individual countries and regions to a greater extent than their natural resources [5]. Innovation processes are becoming a major source of economic growth, especially in the context of the current paradigm of sustainable development and scarcity of natural resources, including energy [62]. Quality technological and organizational change is the basis of innovation.

Strategic directions of innovation activity are legislatively determined for the FEC of Ukraine modernization of power plants; new and renewable energy sources; the latest resource-saving technologies; protection and rehabilitation of the person and the environment.

Considering the technological level of the fuel and energy complex, the total need for innovative financing (according to experts) annually ranges from 8 to 12 billion UAH. Undertakings will not be able to obtain such financial resources without their own foreign investments.

Therefore, the problem of creating an attractive investment climate in the fuel and energy complex of Ukraine for the activation of innovative processes has become of particular importance.

As noted by the specialists of the National Institute for Strategic Studies, in order to increase the investment attractiveness of the innovative market of FEC of Ukraine, it is necessary to carry out a consistent set of measures, which are summarized in Table 1.

Table 1

Measures to create an attractive one investment climate in FEC of Ukraine

And the stage is until 2021	Stage II - by 2025	Stage III - by 2035
1. Formation of a support system for the implementation of energy technologies and management systems (stimulation of the introduction of innovative technologies), the creation of a system of scientific and technical centers for technology development, research and development of personnel potential of energy (grants, scholarships, financing of study abroad). 2. Provision of financial support for basic and applied research.	1. Ensuring the participation of domestic research institutions in EU scientific and technological programs and bringing Ukraine closer to the European scientific space.	1.Creation of national energy companies (power engineering in hydropower, energy production and supply companies) and support of their entry into foreign markets.
3.Approval of the methodology for estimating the cost of capital that can be taken into account for the formation of tariffs for products and services for different branches of FEC	3.Approval of the methodology for estimating the cost of capital that can be taken into account for the formation of tariffs for products and services for different branches of the FEC	
4. Formation of a national energy technology transfer network, state support for technology transfer (technology buy-out), localization of energy technology production.	3. Formation of system of support of venture activity and transfer of energy technologies, development of the Ukrainian engineering companies.	2. Implementation of pilot projects for carbon capture and storage, recycling of household waste, etc..
5. Introduction of new specialties for the energy sector and new training programs for training specialists in educational institutions in order to prepare staff to work in the energy markets in the following areas: - energy economy (energy trading, day-ahead markets, balancing markets, ancillary services markets); - “energy technologies” (modern innovative technologies in energy); - “project analysis in energy”	4. Ensuring the first graduation of specialists in new educational programs by higher education institutions.	3. Improvement of training programs for training specialists in accordance with the needs of competition of Ukrainian companies in the European energy markets.

(business planning of project implementation); - "energy security" (risk assessment and response to energy security threats, international energy markets). 6. Introduction of a system of advanced training of specialists in the issues of implementation of energy management systems, use of renewable energy equipment. 7. Updating of the material and technical base of scientific institutions and higher educational establishments providing training for FEC		

Source: author by [1]

An objective indicator of the effectiveness of innovative activities in the fuel and energy industry, taking into account the requirements of environmental regulation, should be to achieve certain levels of baseline indicators that characterize the efficiency of the fuel and energy and its environmental impact. As an option for possible evaluation, we propose a set of parameters, shown in table 2, formed using data from the National Institute for Strategic Studies.

Achieving these levels of benchmarks requires more than just boosting investment activity. Ultimately, the environmental regulation of FEC (as well as other sectors of the economic complex) requires the construction of a new system of relationships in the chain: central government - regional government (local government) - the entity.

However, the dominant role of central authorities in the implementation of the system of environmental regulation and control of this process has objectively encountered insurmountable difficulties in the current conditions, namely: lack of effective and objective environmental monitoring; poor control over the implementation of state environmental programs and a formal approach to monitoring the implementation of regional environmental programs.

At the same time, regional authorities and local self-government (unlike similar structures in EU Member States) do not consider the environmental issues of their territories as absolutely priority issues, focusing mainly on socio-economic issues, the state of housing and communal services, and employment.

The innovative model of the development of the FEC of Ukraine, taking into account the requirements of environmental regulation, requires a change in the system of relations between the authorities, strengthening the competence, activity and capacity of local authorities to solve environmental problems created by the entities in the respective communities, and Euroadaptation of the FEC consistent investment from business entities, as well as EU budget support and financial assistance.

Table 2

Baseline performance indicators the functioning of the FEC and its environmental impact

Indicators	2020	2025	2030	2035
1.Energy intensity of GDP, kgc / $ 1 of GDP	0,27	0,23	0,20	0,17
2. Fuel costs for TPP for electricity generated, gp / kWh.	384	367	353	334
3. The level of residual resource of FEC fixed assets,%.	30	50	60	80
4.Particle shunting power generation capacity of the fuel power plant to the total installed capacity,%.	12	14	16	18
5. Share of losses in the distribution grids,%.	11	10	9	8
6. Share of energy trading in the stock exchange,% of domestic consumption, including electricity, coal, oil, gas and other fuels.	25	50	60	70
7. Share of renewable energy sources in gross final energy consumption,%.	11	15	18	20
8. Share of local alternative fuels in local fuel and energy balances,% to total consumption.	10	15	18	20
9. Reduction of CO_2 emissions by end-use,%, from 2010.	>5	>10	>15	>20
10. Reduction of specific emissions in CO_2 equivalent at production of 1 kWh,%, from 2010	>5	>10	>15	>20

11. Reduction of specific emissions in CO_2 equivalent in production 1 Gcal,%, from 2010.	>5	>10	>15	>20
12.Particle capacity in thermal generation that meets EU environmental requirements (SO_2, NOx, ash emissions),%.	20	40	80	100

Source: author by [1]

Increased investment in environmental programs, effective control and monitoring, application of environmental management at the sectorial, regional levels and directly by economic entities will provide the process of reforming environmental regulatory tools for the Euroadaptation of national energy.

Conclusions

In the context of Ukraine's energy dependence on Russian supplies and the steady increase in energy prices, the country's burgeoning developing economy suffers significant losses, leading to a decrease in production levels and a slowdown in socio-economic development.

Therefore, the issue of reducing energy dependency through the formation of an effective energy conservation program and the development of alternative energy in Ukraine should be considered as strategically important, which need urgent solution.

Ukraine has significant nuclear and renewable energy potential that can and should be used to stimulate innovative development of the country's economy, energy security and global goals.

Renewable energy plays a critical role in Ukraine's achievement of strategic energy goals, but it also needs to take into account the potential available in the field of energy efficiency, which, in particular, can be used to reduce natural gas consumption.

Given that many power and heat power facilities are outdated and demand is projected to grow, Ukraine's grid needs to build new, more efficient and environmentally friendly facilities.

This requires the formation of an appropriate balance of energy generation capacities: nuclear power, hydropower, solar and wind, biomass energy, other renewable energy sources with the lowest levels of greenhouse gas emissions.

Taking into account the peculiarities of the use of renewable energy sources, in particular the energy of the sun and wind, which are caused by natural conditions, it is necessary to harmonize and balance the frequency and volume of electricity produced at solar and wind power plants into the unified energy system of Ukraine.

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SUBSTANTIATION OF OPTIMAL PARAMETERS OF THE MAN-MADE DEPOSITS

Peregudov V.V.
professor, Doctor in technical science, director,
SE «SDI «Krivbassproekt»

Hryhoriev I.J.
Associate prof., Ph.D. in technical science, deputy director,
SE «SDI «Krivbassproekt»

Hryhoriev Y.I.
Ph.D. in technical science, senior lecturer, Kryvyi Rih National University

The aim: The current states of open mining works at domestic enterprises, as well as the situation in the market of mineral resources require the search for new approaches to open mining works. This approach is a comprehensive development of mineral resources, which contributes to the improvement of technical and economic performance indicators for the mining enterprises. Purposeful formation of man-made deposits with the necessary parameters and their further development is one of the main directions of complex development, and the definition of these parameters and the study of their relationships is the scientific task of this publication.

Methods: The methods of patent search, analysis of literary sources were used to study the technology of formation and development man-made deposit, regression analysis and mathematical modeling of the main parameters of man-made deposits.

Object of research: man-made deposits of bulk type

Subject of research: the relationship of the parameters of man-made deposits

Scientific novelty. The dependences of the optimum values of the capacity and production capacity of the man-made deposit, which provide the best technical and economic indicators of the formation and development of the man-made deposit, are investigated in the work. It is proved that the specific cost of formation and development of man-made deposit are directly dependent on its capacity and inverse - on the production capacity. It was found that the capacity of the man-made deposit has a greater influence on the specific cost of formation and development of the man-made deposit, than its production capacity. Mathematical modeling of the main parameters of the man-made deposit was performed.

Practical significance. The results of studies of the main parameters of man-made deposits can be used by design organizations and mining enterprises in the design. Mathematical dependencies will allow to determine more fundamentally the main parameters of man-made deposit and will increase the accuracy of technical decisions of design institutes.

Keywords: man-made deposit, cost, capacity of man-made deposit, production capacity of man-made deposit, mathematical model, regression analysis.

Introduction

The urgency of the problem. As a result of the activities of the mining and development industries, billions of cubic meters of waste in the form of overburden, sludge, slag, ash, etc., have accumulated on the planet's surface, which exacerbates the environmental burden, and therefore the problem of their utilization is of global importance. At the same time, Ukraine has a strong mining industry and ranks seventh in the world in terms of mineral production, so the problem of waste disposal is of the first importance for Ukraine. More than 2 billion tonnes of rock mass is mined annually from the Ukrainian subsoil, 60-70% of which is stored in stock dumps. However, the level of utilization of production waste reaches only 12-15%, while in the advanced countries of the world it reaches 80%. The tendency for the use of secondary resources is observed in the USA, Japan, Canada, Great Britain, France, Germany, South Africa and other industrialized countries.

For the Kryvyi Rih iron ore basin, which is one of the largest mining regions in the world, the problem of integrated deposits` development and involvement in the recycling mining wastes is becoming increasingly important.

According to various estimates, the dumps and tailings ponds of KryvyiRih mining and processing plants contain up to 13 billion tonnes of overburden and up to 6 billion tonnes of waste of poor iron ore, at the same time the annual economic damage from environmental pollution is estimated at 300 mln dollars. Therefore, it is possible to state with certainty that there is a substantial raw material base of secondary mineral resources [1, 2].

At the same time, the prospect of development of mining is characterized by an increase in the output of minerals with a constant decrease in their quality and complication of the conditions of exploitation of natural deposits. Therefore, in future, deposits of low quality ores, comparable to those of man-made deposits, will be involved in the development. Therefore, undoubtedly, the accumulated mining wastes, which may be man-made deposits, will eventually become one of the important sources of mineral resources.

However, as a result of unsystematic storage of mineral resources, the costs of further development of these mineral objects are significantly increased, as the work scope on re-excavation of the rock mass for the rocks output of the required type increases. In most cases, the development of mineral objects thus formed is associated with large quantitative and qualitative losses, and therefore, in most cases, their exploitation becomes economically impractical.

The aim. Investigate the dependencies and optimize the parameters of the man-made deposit of the bulk type, which would ensure the maximum efficiency of its development.

Scientific tasks. 1. to carry out an analysis of the theory and practice of formation and development of the man-made deposit in conditions of complex development of subsoil;

2. determine the main parameters of the man-made deposit, which have the greatest impact on the economic factors of technological processes;

3. identify the relationship between the main parameters of the man-made deposit, develop a mathematical model and optimize the parameters.

Analysis of research in the formation of man-made deposits

Taking into account external and internal factors in the works [1-3], the authors proposed different technologies of formation and development of man-made deposits, each of which is characterized by different advantages and disadvantages. At the same time, the technical and economic indicators of man-made deposit development will largely depend on the main parameters of the man-made deposit. The first difficulty in this aspect is the lack of a common approach to the list of the main parameters of the man-made deposit.

Secondly, the rational values of the main parameters of the man-made deposit will be determined by a combination of several factors: the cost of land acquisition, the method of formation of the man-made deposit, the equipment, the development system and the physical and mechanical properties of the man-made raw materials.

In the works [3-6] the problem of complex development of the man-made deposits is investigated, the basic terms and concepts are laid, the ways of formation of the man-made deposits are proposed. A number of schemes of selective storage of temporarily off-grade minerals have been found [6]. In addition, methodological principles of purposeful formation of the man-made deposits with specified parameters have been identified [7], which can be reduced to the following regulations:

1. independent storage and development of temporary substandard ores and associated minerals in space and time (according to the regime of mining works in the open pit and market conditions);

2. minimum areas of alienated land;

3. minimum volumes of over-excavation in the development of the man-made deposit and reduction of quantitative and qualitative losses of mineral resources;

4. minimum distances of transportation of temporarily substandard raw materials and associated minerals during the storage and development of the man-made deposits.

However, these selective storage technologies provide the opening of the deposit after its filling by the passage of the surface trenches. Fundamentally different from the schemes described above, but in accordance with the above mentioned principles, the technology of formation of the man-made deposits with the system of tunnels and ore chutes, described in the works [8]. The described technology involves the discovery of the man-made deposit as an object of development, still in the formation stage. So, during the filling of the man-made deposit, the cargo transport connection of every stage with the surface of the bottom of the man-made deposit is secured by laying the fixing of horizontal and vertical workings. The developed technology allows to reduce the cost of development of the man-made deposit and to increase the completeness of mineral output by simplifying the access of extractive equipment to the mineral. Despite this, this technology has several disadvantages:

1. The formation and development of the man-made deposit in this way involves significant capital expenditures during the construction phase. At the same time, 17% of them are the cost of laying the fixing system, which is explained by the complexity of the installation work and the high cost of the materials, which were used.

2. Fixing, that is in the thickness of the man-made deposit, especially horizontal tunnels, require periodic metrological control, because they are in a constantly tense state.

3. The installation of the vertical timbering system requires only a bulldozer peripheral method of dumping. It is characterized by almost uniform distribution of oversize over the series of the man-made deposit. It brings the next problem.

4. In this technological scheme with the placement of vibratory feeders at the bottom of the vertical workings, there is the probability of the backing with the oversize. Therefore, the proposed flow chart requires a uniform granularity without oversize. The technology proposed in [9] is devoid of these problems. According to it, the formation of the man-made deposit is performed by known storage technologies, and the types of minerals can be separated in plan and according to the height. After the release of a certain bank of the man-made deposit on the design boundary or complete filling of the man-made deposit, the ore chute can be formed on it in the form of an open trough.

Depending on the design capacity, the size of the man-made deposit and the angle of inclination of the bank, the open ore chute can be mobile or stationary. The stationary ore chute is placed in the inclined trench. The mobile ore chute is mounted on supports, which are on the bank of the man-made deposit. To reduce of an angle of the open ore chute, its surface may be lined with steel or other materials. At the bottom of the ore chute a hopper-loader with a vibrating feeder is placed. The method of the man-made deposit development is in the removing from the face of the necessary grade of mineral by the wheel loader and delivering it to the storage capacity at the mouth of the ore chute. From there, over the open ore chute, the man-made raw materials are transferred to the vibrating feeder under the action of gravity and bypassed to the rail or conveyor vehicles, located along the horizon of the daylight surface.

Determination of the main parameters of the man-made deposits

The basic parameters of the man-made deposit include the size of the base, the number of stages, the angle of inclination of the bank, the capacity and production capacity of the man-made deposit. The angle of inclination of the bank will depend on the physical and mechanical properties of the rocks, the number of the stages - on the angle of inclination and the size of the basis of the man-made deposit. The capacity and production capacity of the man-made deposit must also be in a specific interdependence.

Minimum of specific costs for the formation and development of the man-made deposit was adopted as the criterion for assessing the efficiency of formation and development of the studied man-made deposits. It is revealed that the capacity of the man-made deposit influences the specific cost of its formation and development. Thus, as the volume of the man-made deposit increases, the specific costs of its exploration decrease, while the specific costs of the formation of the man-made deposit increase. This is explained by the increase in the distance of the overburden transportation during the dumping, which depends on the number of stages in the man-made deposit (fig. 1).

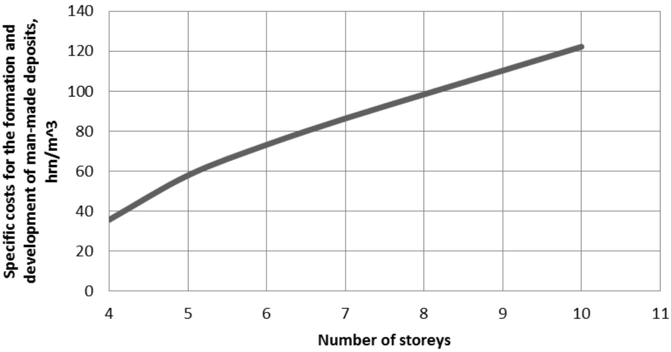

Fig. 1. The specific costs for the formation and development of the man-made deposit from the number of stages dependence diagram

In addition, it was found that the cost of forming and development of the man-made deposit is also affected by the range of transportation of dry mineral raw materials through the man-made deposit. This is due to the fact that with increasing the length of the basis of the man-made deposit increases the haulage of man-made raw materials to the open ore chute and accordingly reduces the annual operational productivity of the pneumatic wheel loader.

The distance of transportation of man-made raw materials through the man-made deposit is calculated as the average weighted distance from the ore chute to the rational limit of the transportation distance by a pneumatic wheel loader of 500 m. This points to the necessity of choice of a rational mechanization complex to develop the man-made deposit. In turn, the cost of the complex will directly affect the cost of formation and development of the man-made deposit as a whole.

In the course of investigations the simultaneous influence of the capacitance of the man-made deposit and production capacity during its development on the cost of forming and development of the man-made deposit was identified. At the same time, it is obvious that the production capacity of the man-made deposit depends on its capacitance.

Therefore, it was decided to investigate the cumulative impact of these factors on final cost. For this purpose two cases were considered: at a fixed capacitance of the man-made deposit; at a constant production capacity.

It should be noted that the criterion for the choice of technological complex was the equipment utilization ratio, which should remain constant for both cases, which ensures the objectivity of cost comparison for both cases.

Research of production capacity of the man-made deposit

For research of the first case, the man-made deposit with a square base shape with a capacitance of 16473456 m3 was considered. It is clear that with the constant capacitance of the man-made deposit, its production capacity will vary depending on the period of development of the man-made deposit.

In order to ensure the rational use of technological equipment (maximize the utilization ratio), a rational mechanization complex was selected in the course of the research to develop the man-made deposit by a complex being a part of a pneumatic wheel loader and a vibrating feeder.

It is established that with different productivity of the man-made deposit, the capital and operating costs for its development, namely depreciation and costs for ongoing repair and maintenance of equipment, depending on the models of the loader and the vibrating feeder, will change.

The results of the research were presented in the form of a diagram (Fig. 2), which shows that with the increase of the working life and, consequently, the decrease in the productivity of the man-made deposit, the specific costs of its formation and development increase.

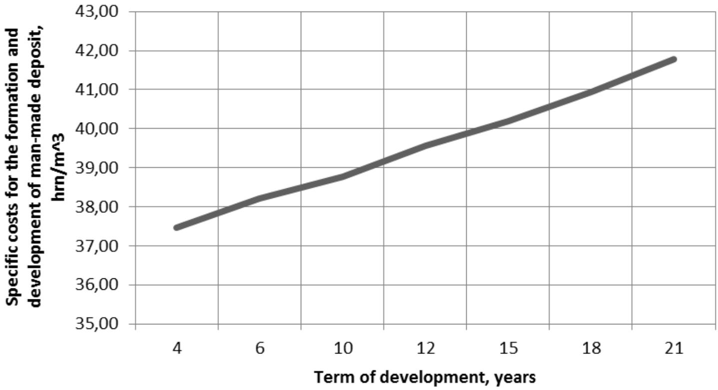

Fig. 2. The dependence diagram of specific costs for the formation and development of the man-made deposit at its production capacity at a fixed capacitance

Investigation of the capacitance of the man-made deposit

For the second case, a constant production capacity for each of the studied man-made deposits at the level of 18200000 m^3/year was adopted. Accordingly, with constant productivity of the man-made deposit, the term of its development depends on the capacitance.

Proceeding that the production capacity for each of the man-made deposits under study is constant at different capacitances, the following complex of technological equipment was selected for their development: one loader CAT988K with a capacity of 2862720 m^3/year and one vibration feeder ПЕВ 2А-4*15 with productivity 2044800 m^3/year.

It is established that the cost of forming and development of the man-made deposit with a given productivity will also be affected by operating costs for its development, namely consumables in contrast to the case described above. The obtained results are presented in the form of a diagram (Fig. 3), which shows that with increasing the capacity of the man-made deposit and, accordingly, the period of its development, the specific costs for the formation and development of the man-made deposit are increasing.

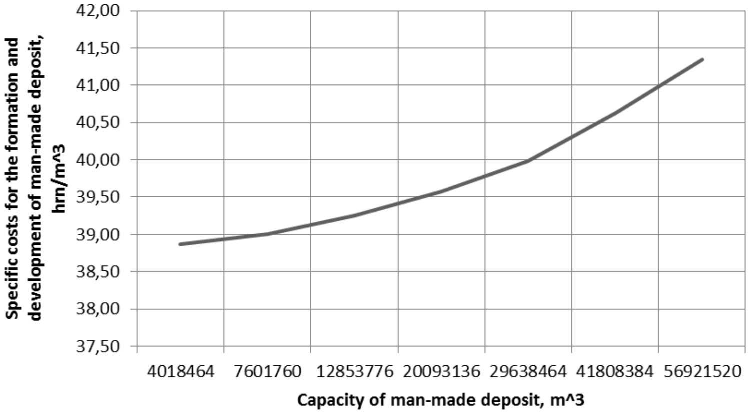

Fig. 3. The dependence diagram of specific costs for the formation and development of the man-made deposit with different capacity at its constant production capacitance

Development of mathematical model of the main parameters

The obtained results allowed us to compile an array of data for the permanent capacitances and production capacity of the man-made deposit and to investigate the cumulative influence of these parameters on the cost of formation and development of the man-made deposit.

For this purpose, a least square regression analysis for a two-argument function was performed, changed to the canonical form, a constraint system was added, and mathematical model according to the formula (1) was constructed and visualized as a three-dimensional diagram (fig. 4).

$$C_{TP}^{num} = 41{,}02 + 4{,}31 \times 10^{-8} \times V_{TP} - 1{,}13 \times 10^{-6} \times A_{TP} \to \min$$

$$\begin{cases} V_{TP} = [4 \times 10^{-6}; 57 \times 10^{6}] \\ A_{TP} = [0{,}78 \times 10^{6}; 4{,}12 \times 10^{6}] \end{cases} \quad A_{TP} = [0{,}78 \times 10^{6}; 4{,}12 \times 10^{6}] \quad (1)$$

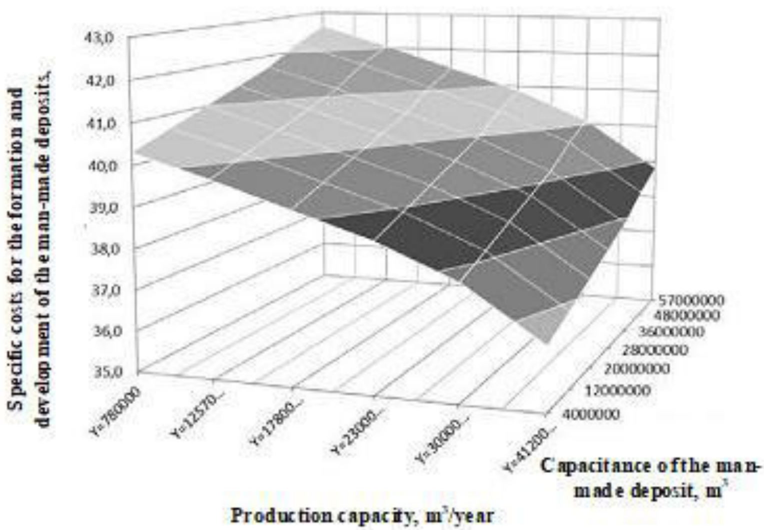

Fig.4. Cost of formation and development of the man-made deposit - its capacitance and production capacity three-dimensional interdependence diagram

The obtained mathematical model reflects the area of dependence of the cost of formation and development of the man-made deposit on its capacitance and production capacity, taking into account the interdependence of the latter.

Conclusions and directions for further research.

Thus, the main parameters of the man-made deposit were identified and their relationship was established. For the method of formation and development of the man-made deposit, investigated in the previous studies, the influence of capacitance and production capacity on the economic indicators of mining works was established. It is proved that the specific cost of formation and development of the man-made deposit are directly dependent on its capacitance and inverse - on the production capacity. At the same time, it was found that the capacitance of the man-made deposit has a greater influence on the specific cost of forming and development of the man-made deposit, than its production capacity.

Further research will be directed to a detailed study of the relationship between the parameters of the elements of the development system of the man-made deposit for various technologies of its formation and development.

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IMPLEMENTATION OF TECHNOLOGIES FOR CRISIS MANAGEMENT OF ENTERPRISES

Kostrychenko V.M.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Associate Professor, Department of Economy of Enterprise, Ukraine

Ignatyuk I.Z.
National University of Water and Environmental Engineering (NUWEE), Ph.D., Associate Professor, Institute of Postgraduate Education, Ukraine

Summary

Modern enterprises operate in the conditions of increased turbulence of the business environment, permanent both positive and negative effects of external and internal factors. The destabilizing effect of certain factors leads to a decrease in the crisis resistance of the enterprise. Therefore, crisis management should be aimed precisely at preventing the enterprise from losing its resilience, preventing or overcoming the crisis of development. To do this, businesses need to have a variety of anti-crisis management technologies.

The subject of the research is theoretical and methodological foundations of introduction of anti-crisis stability management technologies of the enterprise as a process of implementation of anti-crisis procedures applied to a particular enterprise in order to ensure its anti-crisis stability.

The methodology of the research involves reflection of discussion aspects in the interpretation of enterprise crisis resistance, conducting a critical analysis of methodological approaches to its assessment, consideration of a set of basic technologies for managing enterprise crisis resistance, offering synchronization of the type of enterprise crisis resistance and management tools of crisis management technologies.

The purpose of the study is to develop an algorithm for the implementation of management technologies, depending on the level of crisis resistance of the enterprise. The main stages of such an algorithm should be the following: quantitative and qualitative assessment of enterprise crisis resistance, individual selection and implementation of anti-crisis management technologies.

According to the conducted research, active use of various management technologies is required to prevent a decrease in the crisis resistance of the enterprise, as well as to formulate its tactics and strategy for overcoming crisis processes. When choosing anti-crisis management technologies, it is suggested to take into account the stage of the crisis process in accordance with the type of anti-crisis stability of the enterprise and to choose anti-crisis tools that are adequate to these technologies.

Introduction

In today's conditions of increased instability and uncertainty of the market environment, the risk for the enterprise to be in crisis is increasing. This is due to the fact that the enterprise as a socio-economic system develops cyclically against the background of permanent dynamics of the external economic environment. In such circumstances, the role of anti-crisis enterprise management as a systematic process of preventing or overcoming a crisis in order to preserve, restore the pre-crisis state and subsequently improve the potential for successful functioning of the enterprise market, in particular mining industry, is actualized. The need for anti-crisis monitoring of the state of the mining enterprise and the development of its anti-crisis strategy poses the key task of assessing the anti-crisis stability of the enterprise. Particularly acute for enterprises is the problem of choosing and implementing a rational management technology, as well as the skillful use of an arsenal of anti-crisis tools.

Common approaches to the nature and use of anti-crisis management technologies have been studied by domestic and foreign scientists. Theoretical and applied aspects of the problem of economic and anti-crisis resilience, as well as the methodological principles of anti-crisis resilience are described in the writings of economists.

At the same time, as numerous studies show, technologies that allow to eliminate the negative manifestations of crises in an enterprise are of high priority in the current practice of crisis management, but they do not provide the conditions for ensuring its crisis resistance. The negative impact of the factors of the economic environment contributes to the increase of crisis processes, which actualizes the study of management technologies in the direction of regulation of crisis resistance of the enterprise.

The purpose of the study is to develop an algorithm for the implementation of management technologies, depending on the level of crisis resistance of the enterprise.

In order to achieve this goal, the following tasks were solved in the research: to consider specific features of sustainable development of mining enterprises; to open up discussion aspects in the interpretation of enterprise crisis resistance; to conduct a critical analysis of methodological approaches to its evaluation; to characterize modern technologies of crisis management of enterprise; to establish compliance with the type of anti-crisis resilience of the enterprise and tools of anti-crisis management technologies.

The research contains elements of scientific novelty, in particular, the proposed methodological imperatives in the assessment of enterprise crisis resistance, as well as the algorithm of implementation of enterprise crisis management technology.

Setting objectives, the main text

The functioning of the enterprise in the current market environment is accompanied by the complexity of counteracting the enterprise to different groups of factors. In order to ensure the sustainable development of the enterprise in the long term, its stability must be constantly monitored. The problem of defining and ensuring anti-crisis resilience of the enterprise is of particular relevance in a crisis economy and fierce competition in the market, which is characterized by non-deterministic factors of influence.

Mining enterprises are permanently exposed to external and internal factors during their operation. The destabilizing influence of individual factors leads to a decrease in the crisis resistance of these enterprises. Therefore, crisis management should be aimed precisely at preventing the loss of a firm's stability, prevention or overcoming the crisis of development. To do this, there must be a variety of anti-crisis management technologies in the mining enterprise's arsenal. This actualizes the choice of adequate anti-crisis technology of enterprise management, which should depend on the level of economic stability and the direction of its dynamics.

Specific features of sustainable development of mining enterprises

In the Ukrainian economy, the mineral resources sector occupies one of the central positions, which determines the high importance of the state of mineral reserves and the dynamics of its development. In Ukraine, which is rich in various minerals, about 90 of their species have been explored and 8,000 fields are being developed. The average level of use of minerals is no more than 15% of the technologically possible potential; the productivity of labor in the industry is 3 times lower than the level reached in the developed countries. The annual investment in mining is less than 1% of the revenue it receives [1, p.45].

In modern conditions, complex processes of institutional, technological and organizational transformations take place at the mining enterprises. They cover the material and technical base; affect the volume of production and sales of products, economic and financial condition of enterprises. According to the analysis of mining and quarrying indices, the industrial production index in 2016 was 99.8% compared to 2015. In 2017, the industry reduced its production volumes by 5.5% compared to the previous year. The year 2018 was marked by a slight improvement, when the growth rate of production was 102.4% compared to the previous year. The share of unprofitable enterprises in industry for 2015-2018 was within the range of 27.1-28.2%. However, the crisis processes currently unfolding in the global and national economies are likely to lead to an increase in the number of unprofitable enterprises. For comparison, in 2010, after the crisis of 2008-2009, this figure reached 41.3%.

Taking into account the specific features of mining enterprises, it is possible to determine the factors that influence their crisis resistance (Table 1).

Table 1

Factors of influence on the crisis resistance of mining enterprises

Specificity of the mining enterprise	Factors of influence on the crisis resistance of the enterprise
1. Exhaustive reserves of mineral deposits	- limited life cycle of the enterprise; - significant costs of liquidation of the enterprise; - the need for investment in the development of new fields.
2. The location of the enterprise is often in undeveloped areas	- significant costs of creating a complete infrastructure.
3. Release of a monoproduct	- high dependence on market conditions; - low maneuverability in the range.
4. Information insufficiency of information on mining and technological conditions	- significant costs for further exploration of the field; - the risk of commercial inefficiency of the project.
5. High capital intensity of the production process	- significant costs for construction and technical equipment of the enterprise.
6. Dangerous working conditions	- significant costs of safe working conditions.
7. Significant environmental impacts of mining	- significant costs of environmental protection and mitigation measures.

Source: generated by authors based on materials [1, p.47]

At present, the competitiveness and sustainability of most domestic mining companies is ensured by the relatively low cost of consumed energy, material and labor resources, whose prices are on average 4 times lower than the world. In the context of an open economy, it will inevitably equalize the cost of consumed resources to the world level, which will take Ukrainian producers out of competitive production.

The above facts confirm that mining enterprises, by meeting the economic needs at the present stage, threaten the ability to satisfy them in the future, have low rates of development and insufficient level of competitiveness in the world markets, which directly affects the anti-crisis stability of these enterprises.

Under such conditions, mining companies are not only forced to make new technical, technological and organizational decisions that must ensure effective qualitative transformations, but also to introduce modern technologies for managing their anti-crisis stability.

Discussion aspects in the interpretation of enterprise crisis resistance

Review of well-known scientific works on the interpretation of the essence of crisis resistance showed that in modern economic science, this concept is often identified with financial stability. Generalizing the existing interpretations of the concept of "crisis crisis", we can say that the authors, often formulating definitions, often use the following definitions as a basis: it is the ability of the enterprise, it is a complex category, it is a characteristic of the activity or condition of the enterprise, and more. The most common definitions of enterprise crisis resistance are those in which the emphasis is placed on the fact of a crisis situation and decision making on its exit [2; 3; 4; 5; 6]. According to this approach, anti-crisis resilience is the ability of an object to return to a state of economic equilibrium after it has been withdrawn from that state by the negative effects caused by the crisis [5, p.56]. In continuation of the same approach, another definition can be made: anti-crisis resilience is a characteristic of the internal state of the enterprise and its position in the business space, at a proper level of which in the event of a crisis, the enterprise does not reduce its real value or is able to restore it quickly with minimum costs [6, p.324].

However, in our opinion, companies that focus on preventive crisis management in the context of permanent changes and uncertain economic environment demonstrate that crisis is resilient. It is this understanding of the crisis resistance of the enterprise formulated in scientific works [7; 8; 9]. Noteworthy is the interpretation of crisis resistance as the ability of an enterprise to prevent the cardinal impact of the crisis on its financial and economic activities through the use of anti-crisis management technology, taking into account the specifics of the industry and the formation of reserve funds during the phase of lifting the life cycle of the enterprise [9, p.6]. According to this approach, anti-crisis resilience is the ability of an enterprise to withstand crisis influences, and in the event of a crisis, to restore the pre-crisis state in the shortest possible time; anti-crisis resilience is a target function of enterprise management for crisis prevention [8, p.51]. Undoubtedly, the prevention of the negative impact of environmental factors, as well as the obligatory formation of reserve funds in enterprises in order to insure against possible unforeseen circumstances is an indispensable condition for its anti-crisis stability [7, p.38].

Summarizing the scientific views on this category, in our opinion, the anti-crisis stability of the enterprise can be interpreted as such a characteristic of the internal state of the enterprise and its position in the external economic environment, at a proper level of which the enterprise is able to prevent a crisis situation, and in the event of its occurrence can quickly to recover its real value by maneuvering available resources and making crisis decisions.

Methodical approaches to assessing the crisis resistance of an enterprise

Crisis resilience of the enterprise plays an important role in ensuring the long-term functioning and effective development of the enterprise in the face of challenges produced by the competitive environment, uncertainty and threat of crisis situations. Therefore, there is a need for permanent evaluation of crisis resistance in order to make rational decisions on the functioning and development of the enterprise, which will allow to achieve the set strategic goal and the desired result with minimal negative consequences.

Anti-crisis resilience assessment is a set of actions that allow to establish the level, dynamics and tendencies of change of indicators characterizing the results of the enterprise activity, that is, it provides for the interpretation of indicators describing the economic phenomenon. Exploring the methodological principles of anti-crisis resilience assessment, scientists propose different scientific approaches, which explains the existence of alternative methods and different methods of appropriate assessment.

Summarizing the scientific debate on the evaluation of economic and anti-crisis sustainability of the enterprise, we have proposed methodological imperatives that can be formulated in this way [10, p.104]: first, methods for assessing anti-crisis stability need to adapt to the peculiarities of functioning in conditions of transformational economic environment; second, anti-crisis resilience studies should be conducted using analytical and evaluation procedures; thirdly, they need to distinguish the anti-crisis resilience subsystems, which are associated with the provision of certain areas of activity of the enterprise; fourthly, a rational system of assessment of the anti-crisis stability of the enterprise should be formed; Fifth, a quantitative and qualitative assessment of the level of the enterprise's anti-crisis resilience indicator should be carried out; sixth, a certain type of anti-crisis resilience should identify the stage of the crisis process, which will allow to rationalize the anti-crisis tools of the enterprise.

Critical analysis of existing methods for assessing the crisis resistance of the enterprise [2; 3; 4; 11; 12] showed that there are different visions of such calculations in the scientific environment. The authors propose different principles of construction of the methodology, the list of components of the crisis stability of the enterprise and a set of indicators of crisis resistance of the enterprise, in which the systematic reflection of external and internal, financial and non-financial indicators is not always present.

In our opinion, the best way to evaluate the crisis response of an enterprise can be built on the basis of a comprehensive approach and taking into account the principles of forming a balanced scorecard. Such a system is a strategic planning and management tool that serves to set strategic goals and evaluate the effectiveness of an enterprise from the point of view of strategy implementation with the help of certain key performance indicators [13, p.247]. The scorecard itself is formed in the context of four projections: finance, customers, internal business processes, training and staff development. Based on this approach, we have proposed a system of indicators for evaluating the crisis resistance of the enterprise, combined, according to projections, within the following functional components: financial, marketing, operational, investment-innovation, management and information, personnel.

An enterprise, having information on the level and dynamics of anti-crisis stability, can decide on the configuration of technological schemes of anti-crisis management.

Modern technologies for crisis management of enterprise

Broadly speaking, technology is a set of actions or sets of operations that are used in various fields. Management technology is a constant creative process of maintaining a sustainable mode of functioning of the economic system by making and implementing economic decisions. At some stage of evolution, technology was seen as information and means of storing it, as activity and knowledge about cause and effect, as the variability of resources and processes of search [14, p.58].

The meaningful content of the concept of "control technology" can be revealed using content analysis. It is by the results of such a study [15] that a general definition was formulated: management technology is a purposeful, orderly, coordinated set of management procedures that are implemented within management functions and provided with a choice of appropriate management tools and methods [15, p.74].

In Western management there are three main groups of technologies: technologies of planning of management decisions; technologies of implementation of management decisions; technologies for changing the structure of the organization. In our country, the most common way is thematic-sectorial grouping of technologies by relevant sections of management (strategic management, personnel management, business process management, logistics management, etc.), by management functions (planning, organization, motivation, control), or by major management perspectives (finance, customers, internal processes, training and development) [16].

The analysis of the application of management technologies in economics and finance shows that today the priorities are shifting in the direction of transition from special management technologies to complex self-replicating ones. Integrated technology for managing financial and economic processes is a set of purposeful, hierarchically ordered, coordinated in time and resources, rational management procedures that correspond to the state of the external and internal environment of the economic entity and are implemented within the range of management functions through the use of management tools [17, p.327].

The following features are inherent in the complex technology of financial and economic process management: strategic orientation; extensive composition of procedures; dependence of the content of management procedures on the phase of economic entity development; coverage of several management functions; requirements for high competence of personnel for the use of complex technology [17, p.327]. The above specificity confirms the possibility of using such technologies in crisis management of the enterprise.

Anti-crisis management technology means a set of sequential measures taken to prevent, prevent, overcome a crisis and reduce the level of negative consequences of its implementation. It demonstrates the peculiarities of the behavior of the management subject in a crisis situation, reflecting the patterns and specifics of the choice and implementation of a certain sequence and interrelation of operations on the development of anti-crisis solutions [18, p.382].

All anti-crisis management technologies can be grouped into the following active, reactive and predictive anti-crisis management technologies. A concise description of these technologies is shown in Table 2.

Table 2

The set of anti-crisis management technologies

Technology groups	Intrinsic characteristic of technology
Active management technology	It assumes the transition to action when the threshold of rational understanding of the threat of crisis phenomena is reached and to determine on this basis the nature of emerging problems, selection and testing according to the criteria of strategic development of the most acceptable options for operational response to emerging threats.
Jet control technology	Displays conditions under which the rate of change in the management object is low and strategic discrete deviations from the established trends are rare; at the initial stage of response to a threat, an idea of the nature of the problem is formed and a consistent strategic response is made in the form of an attempt to apply operational measures, saving resources.

Continuation of table. 2

Predictable crisis management technology	Built on the pre-emptive action before a crisis threat, on recognizing the nature of the threat and selecting options for combining strategic and operational responses; based on forecasting, which allows the company to take the necessary steps before a possible threat can cause enough tangible damage to the enterprise.

Source: Authored by authors [18].

The modern arsenal of control technologies has a significant number of their types depending on the degree of centralization, the order of operations and procedures, the degree of division of labor, object, subject, functional orientation, the level of automation, structuring of the control object, the level of control [17, p. 325].

In summary, the composition of technologies used in economic management and the main types of crisis management technologies for economic processes are shown in table 3.

Table 3

Modern anti-crisis technologies of economic process management

Technologies used in economic management	The main types of crisis management technologies for economic processes
Strategic Management	Customer Relationship Management (CRM)
Business Planning	Benchmarking
Marketing Management	Strategic planning
Financial Management	Outsourcing
HR	Balanced Scorecard (BSC)
Corporate Governance	Controlling
Quality management	Change management
Information Technology	Key competencies management
Operational management	Business Process Reengineering
Production management	Consumer segmentation
Logistics technologies	Quality management
Structuring and organization of business processes	Loyalty management

Source: generated by the authors on materials [14, p.64-65; 17, pp.324-325; 19].

The monitoring of the implementation and use of various management technologies, which has been conducted for a long time by the consulting company Bain & Company, showed the level of popularity and effectiveness of the use of management practices. The most widely used management technologies in the world are customer relationship management, benchmarking, employee engagement surveys, strategic planning, outsourcing, a balanced scorecard, mission and vision development [19].

In domestic practice, the high level of use has the tools of customer relationship management (66.7% of respondents) and budgeting technology (70.6%) [20, p.153]. This indicates an active search for methods of enterprise adaptation by domestic management, identification of new sources for obtaining competitive advantages and directions of development of Ukrainian business.

At the same time, Ukrainian enterprises are under-utilizing the management technologies of innovative management. In particular, only 39% of respondents admit that their businesses have innovation management, 31% - benchmarking, 32% - knowledge management, 22.5% - change management, 21% - scenario planning. In the conditions of turbulent changes of the external and internal environment, the tasks of development and implementation of management technologies are actualized, which requires constant benchmarking, scenario planning and systematic change management [20, p.154-155].

The dynamic changes that take place in the business environment require the use of management technologies that are capable of ensuring the anti-crisis stability of the economic system.

The algorithm of introduction of technologies of crisis management of enterprise

Management of crisis resistance of the enterprise involves the construction of appropriate technological schemes of crisis management. Therefore, management technology involves an orderly sequence of implementation of a set of certain operations in the development and use of anti-crisis tools. In the process of such activity, it is important not only to choose the management technologies that best meet the nature of the problems, the level and type of crisis resistance of the enterprise, but also to ensure their implementation.

The description of the steps in our proposed algorithm allows us to demonstrate the procedural features of the implementation of anti-crisis management technologies of the enterprise (Table 4).

Modern methodological approaches to the process of establishing the level of anti-crisis stability should take into account that the procedures for the study of anti-crisis stability of the enterprise are divided into two groups, namely: analytical and evaluation. Analytical procedures involve the use of such economic tools that can quantify the level of crisis resistance of the enterprise. Evaluation procedures provide a qualitative assessment and compliance with the level of the overall integral index of anti-crisis firm stability of the Harrington scale, as well as determining the type of anti-crisis stability [21, p.60].

Table 4

An algorithm for the introduction of anti-crisis resilience management technologies

Stage	Contents of the stage
Step 1. Quantification of ACP	1. Definition of ASP subsystems (components). 2. Formation of a system of estimates. 3. Calculation of individual indicators of the ASA, their evaluation and standardization. 4. Determination of the weight of single and key performance indicators in ASA subsystems. 5.Calculation of key performance indicators for each subsystem and their interpretation. 6. Calculation of the integral indicator of the ASA.
Step 2. Qualitative assessment of the UAV	1. Establishment of compliance with the level of the overall integral index of the FAS of the Harrington scale. 2. Determination of the type of ASP.
Step 3. Synchronization of the type of ACP and solutions in crisis management technologies	1. Identification of the stage of the crisis process according to the type of ASA. 2. Establishing an appropriate type of crisis management.
Step 4. Individual selection of crisis management technologies	1. Formation of a set of anti-crisis technologies of enterprise management. 2. The choice of management technologies, depending on the level of ASP.
Step 5. Implementation of ASA management technologies	1. Creation of a system of information and intellectual support for crisis management of the enterprise. 2. Arrangement of anti-crisis tools for providing the ASA. 3. Implementation of ASA management technology. 4. Evaluation of the effectiveness of the implementation of the control technology of the ASA.

Source: Developed by the authors

The evaluation process does not end with the calculation of the overall integral indicator of the enterprise's crisis resistance. Next, the compliance with the indicator level of a certain stage of the crisis process, which determines the type of crisis management [22, p.230] (table 5).

Table 5

Compliance with the type of anti-crisis stability of the enterprise type of anti-crisis management

Type of crisis resistance	Stage of the crisis process	Type of crisis management
Absolutely	Crisis phenomenon	Preventive
Enough	The crisis situation	Wellness
Unstable economic condition	Pre-crisis state	Stabilization
Low	Crisis	Rehabilitation
Catastrophic	Crisis	Liquidation

Source: Developed by the authors

The above identification creates the basis for the application of rational anti-crisis management technologies and making effective business decisions by the enterprise (Table 6). Given that the crisis process in the enterprise is able to move from individual crisis phenomena to a crisis situation, which can develop into a crisis state, in the practice of crisis management, different methods can be used both in relation to individual parameters of the crisis phenomenon and in relation to the crisis situation as a whole [22 , p.231].

Tactical and strategic methods are used in managing the crisis resistance of the enterprise. Tactical methods, including downsizing, remediation, monitoring, controlling, are aimed at rapid improvement of financial performance of the enterprise, ie overcoming the consequences of the crisis. However, in most cases these methods are not sufficient to address the root cause of the crisis - an inefficient management system. This requires the use of longer-term strategic measures, such as diversification, regularization, reengineering, restructuring, mergers, liquidation, which are aimed at improving the quality characteristics of enterprises [23, p.118-119].

Table 6

Solutions developed within the framework of anti-crisis technologies

Type of crisis resistance	Contents of anti-crisis solutions
Absolutely	aimed at ensuring crisis resistance and preventing the emergence of crisis phenomena, as well as further development of the enterprise
Enough	aimed at restoring anti-crisis stability and preventing the emergence of crisis situations at the enterprise
Unstable economic condition	aimed at stabilizing the enterprise and avoiding bankruptcy proceedings

Continuation of table. 6

Low	aimed at withdrawing a company from a crisis state, achieving a pre-crisis state; aimed at minimizing the negative effects of the crisis, preventing the liquidation of the enterprise
Catastrophic	are directed to proceed with the procedure for liquidation of the company against which the bankruptcy case is initiated

Source: Developed by the authors

In the process of crisis management of the enterprise there is a problem of making complex management decisions and not only the choice but also the introduction of the most effective management technology. This necessitates the use of information technologies capable of enhancing the intellectual level of the decisions being made, their timeliness and adequacy. It is necessary to take into account the peculiarities of using information technologies in accordance with the types of crisis management.

Creation of information-intellectual support system for crisis management of the enterprise enables the managers of the enterprise to effectively manage certain areas of activity of the enterprise, to coordinate the activities of structural units, quickly find information about changes in the internal state of the enterprise and changes occurring in the external environment, to track trends, to have them, comfortable display, and evaluate the effectiveness of management decisions made in the environment withdrawal of the enterprise from crisis [24, p.142-143].

Determining the selection and application of anti-crisis technologies for managing the anti-crisis stability of the enterprise is a careful selection and justification of practical anti-crisis tools. Combining anti-crisis solutions can be different, given the type of anti-crisis stability and the corresponding stage of the crisis process [22, p.231].

Loss of anti-crisis stability by the enterprise necessitates the development and implementation of a set of strategic, tactical and operational economic decisions. At each stage of the crisis process, it is necessary to choose the dominant individual anti-crisis solutions or a combination of them, which will ensure rapid recovery of anti-crisis stability or minimize the negative consequences of the crisis [22, p.231].

Strategic managerial decisions include drastic restructuring at the enterprise, change of directions of financial flows into product groups or target segments, transformation of structure, entry into new regional markets, expansion or reduction of activity, change of assortment policy [22, p.231].

Tactical management decisions are related to the development of new models of enterprise behavior in the market, change in pricing policy, organization of promotions, updating of material and technical base, optimization of work of the team, etc. Operational management decisions are made in the process of daily work of the labor collective and are related to the organization of work of direct executors, providing them with resources, front of work, information [22, p.231].

However, most Ukrainian businesses in management practice are focused on monetary (financial) business valuations. This indicates a low level of strategic orientation of enterprises, the desire to focus activities on achieving acceptable levels of short-term financial indicators (in particular, profit, sales). As a result, there is an insufficient use of modern anti-crisis management technologies by domestic enterprises.

Conclusions

1. In today's conditions of increased turbulence of the economic environment, the increase of negative processes in the functioning of industrial enterprises increases the threat for them to be in crisis. Given the current economic trends and specific features of mining companies, it is possible to predict the gradual loss of their anti-crisis stability.

2. Crisis resistance of the enterprise can be interpreted as such a characteristic of the internal state of the enterprise and its position in the external economic environment, at a proper level of which the enterprise is able to prevent a crisis situation, and in case of its occurrence can quickly recover its real value, maneuvering available resources and accepting anti-crisis solutions.

3. The methodology for assessing the crisis crisis of the enterprise is based on a balanced scorecard, which is formed in the context of four projections: finance, customers, internal business processes, training and staff development. In the system we offer, the indicators of anti-crisis resilience assessment of the enterprise are combined,

according to projections, within the following functional components: financial, marketing, operational, investment-innovative, management-informational, personnel.

4. The overall level of application by Ukrainian enterprises of modern anti-crisis management technologies remains relatively low. The problem remains with the use of information and virtual technologies by the enterprises of crisis management, which are able to increase the intellectual level of the decisions made.

5. The introduction of anti-crisis resilience management technologies of the enterprise should be carried out according to the appropriate algorithm, which provides for individual selection of anti-crisis management technologies. This should take into account the stage of the crisis process in accordance with the type of anti-crisis resilience of the enterprise and select anti-crisis tools that are adequate to these technologies.

In the future, the scientific and practical interest is the development of anti-crisis strategies and the design of technological management schemes, taking into account the level of anti-crisis stability of the enterprise and possible scenarios for its change.

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THE RESEARCH OF INERTIAL CONVEYOR TRANSITIONAL CHUTE OSCILLATIONS INFLUENCE ON ITS TECHNICAL AND ECONOMIC INDICATORS

Serilko L. S.
National University of Water and Environmental Engineering, Ass. Prof., PhD of Technical Sciences, Associate Professor of Theoretical Mechanics, Engineering Graphics and Mechanical Engineering the Department, Rivne, Ukraine

Lyashuk O. L.
Ternopil Ivan Puluj National Technical University, Professor, Doctor of Technical Sciences, Head of Automobile Department, Ternopil, Ukraine

Sasyuk Z. K.
National University of Water and Environmental Engineering, Ass. Prof., PhD of Agricultural Sciences, Associate Professor of Theoretical Mechanics, Engineering Graphics and Mechanical Engineering the Department, Rivne, Ukraine

Serilko D. L.
National University of Water and Environmental Engineering, Ass. Prof., PhD of Technical Sciences, Associate Professor of Construction, Road, Reclamation, Agricultural Machines and Equipment Department, Rivne, Ukraine.

Abstarct

The construction of an inertial conveyor for bulk materials transportation has been elaborated in order to increase productivity and reduce the energy intensity of the bulk materials transportation process. The inertial conveyor consists of a chute, which performs reciprocating motion on a fixed basis and actuating the reciprocating movement of the chute. The chute can perform not only a reciprocating motion, but also a movement in a direction that is perpendicular to the axis of the conveyor in a horizontal plane. It has been proposed to reduce the friction force between the load and the chute at a time when the speed of the chute relative to the load will be directed in the opposite direction. The analyses of known methods of solving nonlinear differential equations, which describe the motion of mechanical systems in the presence of dry friction has been done. A mathematical model of a bulk material particles motion over a surface that performs a harmonic motion has been elaborated. A construction of efficient inertial conveyor has been created. The influence patterns of structural and kinematic parameters of the inertial conveyor on the indicators of bulk materials transportation process by this vehicle have been experimentally investigated. A mathematical model of the flow of a bulk material in a rectangular chute that performs high intensity transverse oscillations in the plane of greatest inclination has been developed. It has been established experimentally that, in the presence of such oscillations, the material retains the shape of the chute and therefore its motion can be regarded as the motion of a solid on an inclined surface. The averaging method obtained an approximate solution of the equations of this motion, which satisfactorily agrees with the results of experimental studies.

The obtained dependences of the average velocity of the bulk material on the surface of the chute from the angle of inclination to the horizon and the coefficient of friction of the material on this surface can be applied in engineering calculations for solving specific practical problems.

Keywords: dynamic system, vibration transportation, inertial conveyor, transportation of bulk materials, transverse oscillations, friction force, vibration, oscillation frequency.

Introduction

A continuous improvement of the technical level of new mechanisms and machines is a characteristic feature of the modern development of world mechanical engineering.

The improvement of the constructions of various conveyors (belt, screw, inertial, etc.) is one of the directions of bulk materials transportation means development, which provides high productivity, technical and economic efficiency. The conducted analysis of designs of various vehicles constructions shows that in mining enterprises, for the transportation of bulk materials (coal, ore, etc.) at a distance of 20 to 100 meters, it is advisable to use inertial conveyors with a constant load pressure to the bottom of the chute [1]. These conveyors have considerable stability because the oscillation amplitude of the conveyor chute is constant.

Many domestic and foreign companies (KNAPP, SIAT, Blume, Librawerk, “Steklopak”, “Siberian Machine-Building Company”, “Potok-TM”, “Project Invest”) are engaged in research aimed at creating new-generation conveyors [1]. The widespread popularity of such studies is justified by the following factors: simplicity of construction, tightness, the possibility of combining the process of transportation with technological operations (sifting, drying, cooling), transportation of various materials of the construction industry (gypsum, sand, clinker, crushed stone, energy consumption), low chute wear compared to vibrating conveyors.

The inertial conveyors that transport goods due to the harmonious nature of the chute movement are particularly prominent. They have been used for transportation of various bulk and artificial loads, especially those where there is an undesirable mode of work with the tossing of material particles (heavy metal chips, hot wet sugar, etc.). Otherwise, when the inertial conveyor is in operation, there is no contact between the load and the rotating parts, such as in a screw conveyor.

On the surface of the inertial conveyor chute the load moves cyclically, that is, in one rotation of the crank chute performs a straight-line movement, and the movement back and forth is carried out at different speeds. Due to this, the load moves on the surface of the chute with some average speed, on which the performance of the conveyor depends.

The kinematic characteristics of the chute movement, depending on the angle of crank rotation are shown in Fig. 1.

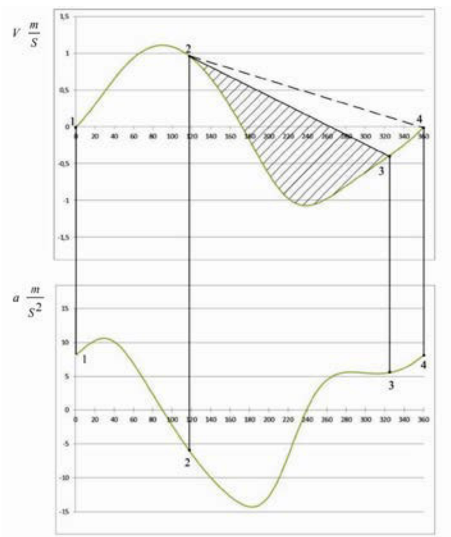

Fig. 1. The dependence of the velocity and acceleration of the inertial conveyor chute on the position and speed of the crank turn

Analysis of recent research and publications. In works[1, 2] the issues of kinematics and dynamics of inertial conveyors have been considered, which allow to judge its performance and energy costs for load movement and other operational characteristics.

The studies [3,4] have been dedicated to the development of the construction and study of the inertial conveyor using linear asynchronous electric drive. These mechanisms have been used to transport wet sugar. Because, unlike vibration conveyors, they work without tossing loads, which leads to sugar segregation.

In scientific works [5,6], the results of studies of kinematic and dynamic parameters of inertial conveyors with elastic units have been presented. They can be used to select the optimal parameters when creating inertial conveyors, which are characterized by compactness and reduced material consumption and energy consumption.

The problems of studying the oscillations of dry friction systems, which arise in the study of the operation of inertial conveyors, are a typical example of the problems of nonlinear mechanics, which require special methods of analysis to be solved. These include: small parameter method, harmonic balance method, asymptotic methods and numerical methods [7-10].

The main disadvantage of inertial conveyors is the reduction of the velocity of the material being transported when the conveyor belt has been reversed due to friction. The conversion of dry friction into a binder allows the reduction of friction resistance in some selected direction, which is a prerequisite for creating a new generation of conveyor systems.

The main directions of the research are to solve the equations of bulk material particles motion on the surface of the chute, which performs transverse oscillations and the amplitude and frequency of which varies according to laws of different types. As a result, this action will optimize the construction of kinematic parameters of the inertial conveyor.

Among the publications on the issue of vibration displacement of solids and bulk materials, it is worth mentioning the works of Blehman I., Dzhanelidze G., Zayika P. [11,12].

The aim of the study formulation. The aim of the study is to establish the laws of bulk material particles motion on the surface, which performs a harmonic movement and to create on their basis constructions of high-performance inertial conveyors. In the framework of the work it is planned to develop a mathematical model of the movement of bulk material particles on the surface of the chute and determine the movement velocity of this material (conveyor performance), which make it possible to substantiate the rational construction and kinematic parameters of this vehicle. Also, get simple dependencies to determine the average movement velocity of the bulk material in a rectangular chute, which transversely oscillates in the plane of the bottom of the chute, perpendicular to the line of greatest slope.

The outline of the main research material. The construction of an inertial conveyor for bulk materials transportation has been elaborated in order to increase productivity and reduce the energy intensity of the bulk materials transportation process. The inertial conveyor consists of a chute, which performs reciprocating motion on a fixed basis and actuating the reciprocating movement of the chute. It should be mentioned that, the chute can perform not only a reciprocating motion, but also a movement in a direction that is perpendicular to the axis of the conveyor in a horizontal plane. It has been proposed to reduce the friction force between the load and the chute at a time when the speed of the chute relative to the load will be directed in the opposite direction [13].

The scheme of the proposed inertial conveyor is shown in Fig. 2

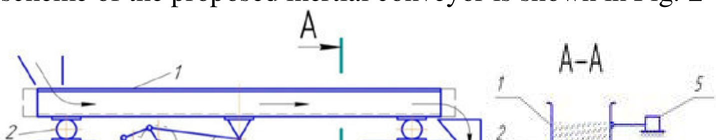

Fig. 2. Intertial conveyor
1 – chute; *2* – ball; *3* – base;*4* – reciprocating drive; *5* – vibrator

The inertial conveyor (Fig. 2) has a chute 1 that can move both along and across the axis of the conveyor on the balls 2 on a fixed base 3, drive reciprocating motion 4, vibrator 5 for the implementation of transverse vibrations of the chute and a vibrator control that is not shown in the figures.

The principle of an inertial conveyor operation is as follows: chute 1 moves to the right by actuator 4 and the load on the chute begins to move with the chute. When the velocity of the chute 1 reaches the maximum value, the vibrator 5 is switched on and the chute begins to transverse. Thus, the friction force between the material and the chute will in most cases be directed in the transverse direction. Because the frequency of transverse oscillations is much greater than the frequency of longitudinal oscillations of the chute, but the friction force is always directed in the opposite direction from the vector of relative velocity of the material movement on the surface of the chute and now its longitudinal value will have a minimum value. Inertia, the material will move on the surface of the chute until the chute itself begins to move in the required direction. At this point in time, the vibrator shuts off and the cycle repeats.

Therefore, the studied inertial conveyor allows increasing productivity and improving the energy performance of the material transportation process [14].

The scheme and the general view of elaborated experimental setup are presented in Fig. 3 and Fig.4.

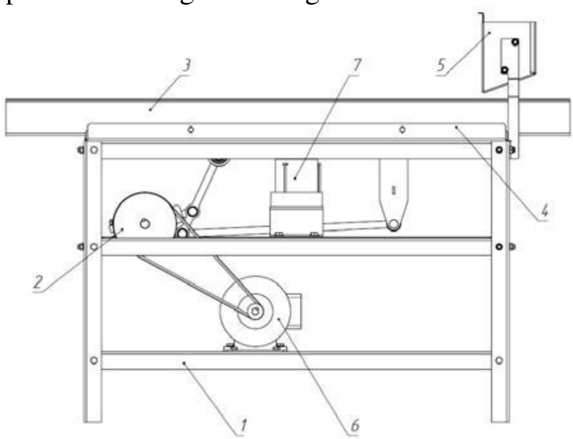

Fig. 3. The scheme of experimental setup:
1 – frame; *2* – chute drive; *3* – chute; *4* – guide of chute; *5* – material feed hopper; *6* – DC electric motor; *7* – vibrator.

Fig. 4. General view of the elaborated inertial conveyor

The motion of the particle located on the surface of the groove of the inertial conveyor should be considered to determine the velocity of the bulk material [12].

The chute moves in a horizontal plane along the axis of the conveyor by law $S_x = S(t)$ and across the axis of the conveyor by law $S_y = A\sin\omega t$ (Fig. 5).

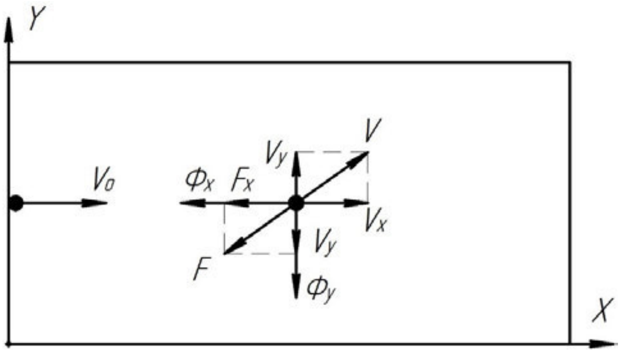

Fig. 5. The scheme of a particle motion of material on the surface of the chute

The force of gravity $\vec{P}$, the surface reaction $\vec{N}$, the friction force $\vec{F}$ and inertia forces $\vec{\Phi}_x$, $\vec{\Phi}_y$ are acting on the particle.

$$\Phi_x = -m\ddot{S}_x \tag{1}$$

$$\Phi_y = -m\ddot{S}_y \sin \omega t \tag{2}$$

Since the friction force directed in the opposite direction of the vector velocity point, then expand it into two parts F_x and F_y

$$F_x = f_N \frac{V_x}{\sqrt{V_x^2 + V_Y^2}} \;; \tag{3}$$

$$F_y = f_N \frac{V_y}{\sqrt{V_x^2 + V_y^2}} \tag{4}$$

where V_x=x – the projection of the velocity vector of the particle on the axis V_y=y, – the projection of the velocity vector of the particle on the axis y.

The motion law of the chute S_x depends on the geometric and kinematic characteristics of the inertial conveyor drive.

The differential equations of a particle motion of the bulk material, in projections on the axis x and y will be the following

$$\begin{cases} \ddot{x} = -\ddot{S}_x - fg\sqrt{\dfrac{V_x}{V_x^2 + V_y^2}} \\ \ddot{y} = A\omega^2 \sin \omega t - fg\sqrt{\dfrac{V_y}{V_x^2 + V_y^2}} \end{cases} \tag{5}$$

As the system of equations (5) is essentially nonlinear, so it cannot be integrated into quadratures and its periodic junction cannot be found in a closed form.

By solving the obtained system of equations by numerical method, the velocity of a particle motion of material on the surface of the chute under the given initial conditions of motion should be determined [15].

The obtained dependences of the coordinates and velocities of the particles of the bulk material on time, allow to determine the performance of an inertial conveyor with specified geometric and kinematic characteristics. The dependences of the velocity of a particle motion of the material on the horizontal surface on the frequency and amplitude of oscillations are given in Fig.6 [16].

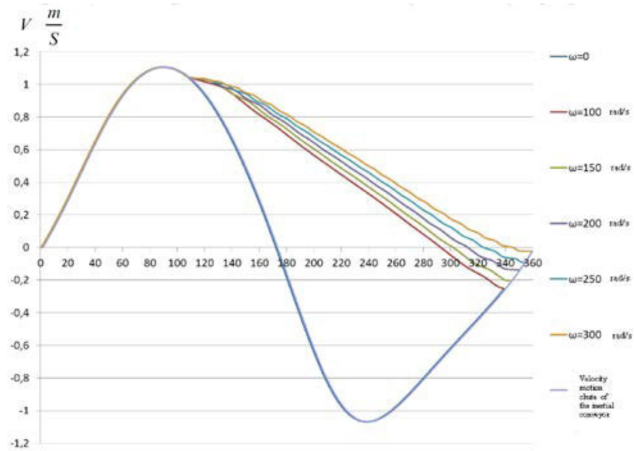

Fig. 6. The dependence of the velocity motion of the material on the chute surface of the inertial conveyor on the position of the crank and the vibration amplitude of the vibrator. (n=100 rev/min, ω=150 rad/s)

The dependence of the load motion on the chute surface of the inertial conveyor on the frequency and the oscillation amplitude of the vibrator is given in Fig. 7 (n=100 rev/min).

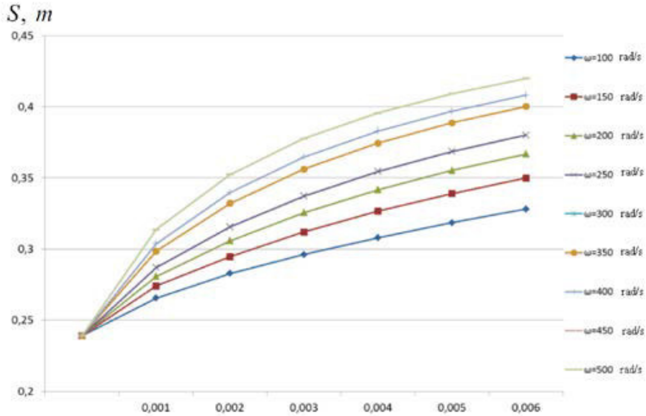

Fig. 7. The dependence of the load motion on the chute surface of the inertial conveyor on the frequency and the oscillation amplitude of the vibrator (n=100 rev/min)

The dependency response surface of load motion along the surface of the inertial conveyor chute from the frequency and oscillation amplitude of the vibrator is given in Fig.8.

The flow of bulk material in the chute, which is inclined at an angle α to the horizon, has been considered.

It has been assumed that the layer of bulk material with the height h is in a rectangular chute, which is located at an angle α to the horizon (Fig. 9) while $\alpha<\varphi$, where φ - the sliding friction angle of the bulk material on the chute surface.

In the general case, to determine the velocity distribution of particles motion of bulk material in the chute, it is necessary to write the differential equations of motion of a continuous medium, continuity, barotropy and boundary state. Solving the differential equations obtained with partial derivatives, under certain boundary conditions, that is associated with considerable mathematical difficulties and it is possible to make them only under certain assumptions.

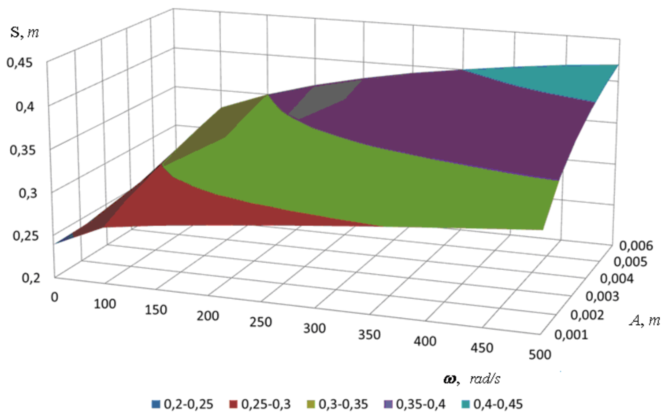

Fig. 8. The dependency response surface of load motion along the surface of the inertial conveyor chute from the frequency and oscillation amplitude of the vibrator at crank rotation speed n=100 rev/min

Consider the transverse harmonic oscillations of a high-intensity chute , where A - oscillation amplitude, m; ω - oscillation frequency, s^{-1}.

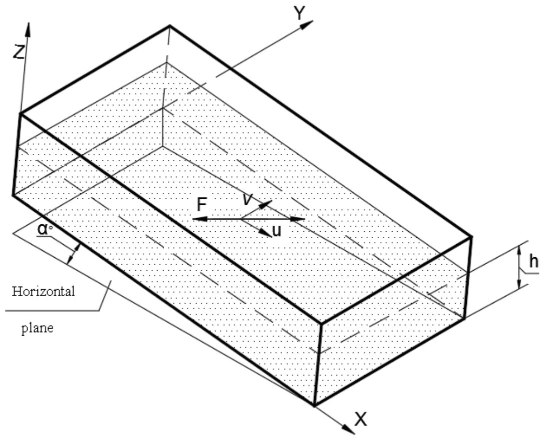

Fig. 9. The calculating scheme of the chute with a layer of bulk medium

Experimental studies (A=0,001 m, ω=140 s^{-1}) show that the bulk material does not change the rectangular shape, that is, during the period $T = \frac{2\pi}{\omega} = 0{,}045s$ the particles of material do not have time to crumble.

In this case, the motion of the bulk medium in the chute can be considered as the motion of a solid body, which has the shape of a parallelepiped (Fig. 9).

Since the force of friction between the body and the bottom is always directed in the opposite direction of motion, its projections (F_x and F_y) on the coordinate axis will be

$$\begin{cases} F_x = fN_z \cdot \dfrac{u}{\sqrt{u^2+v^2}} \\ F_y = fN_z \cdot \dfrac{u}{\sqrt{u^2+v^2}} \end{cases}$$

where $u = \dot{x}$ - velocity of the body, m/s, v - the velocity of the chute in the transverse direction, m/s.

As N_z=mg $\cos\alpha$, so

$$F_x = fmg\cos\alpha \cdot \frac{u}{\sqrt{u^2+v^2}}$$

The differential equation of motion of this body in the projection on the axis *x*

$$m\ddot{x} = mg\sin\alpha - fmg\sin\alpha \cdot \frac{u}{\sqrt{u^2+v^2}}. \tag{6}$$

In our case, the chute performs transverse oscillations by law $y = A\sin\omega t$

The projection of the chute velocity on the *y*-axis will be $v = \dot{y} = A\omega\cos\omega t$.

Then the differential equation of motion of the material on the surface of the chute will look like

$$m\ddot{x} = mg\sin\alpha - fmg\cos\alpha \cdot \frac{\dot{x}}{\sqrt{\dot{x}^2 + (A\omega\cos\omega t)^2}} \tag{7}$$

The resulting equation does not integrate into quadratures and its solution cannot be found in general form, but it is possible to get an approximate solution [7].

Dimensionless quantities

$$\tau = \omega t; \xi = \frac{x}{A}; \dot{x} = \frac{dx}{dt} = \frac{d(\xi A)\cdot\omega}{d\tau} = A\omega\xi',$$

where ξ' derivative of ξ in dimensionless time τ

$$\ddot{x} = \frac{d(\ddot{x})}{dt} = \frac{d(A\omega\xi')\cdot\omega}{d\tau} = A\omega^2\frac{d\xi}{d\tau}.$$

Equation (7) takes the form

$$A\omega^2\frac{d\xi'}{d\tau} = g\sin\alpha - fg\cos\alpha \cdot \frac{A\omega\xi'}{\sqrt{(A\omega\xi')^2 + A^2\omega^2\cos_2\tau}} \tag{8}$$

The notation is entered

$$\mu = \frac{fg\cos\alpha}{A\omega^2}; \quad y = \frac{g\sin\alpha}{f\cos\alpha} = \frac{\tan\alpha}{f}.$$

Finally (6) in dimensionless form

$$\frac{d\xi}{dt} = \mu\left(y - \frac{\xi'}{\sqrt{\xi'^2 + \cos 2\tau}}\right). \tag{9}$$

Because in our case

$$A\omega^2 >> \frac{gt}{\cos\alpha}.$$

Therefore, μ-value in this equation is a small parameter and the averaging method to solve the equation can be used. It should be assumed that velocity is constant over a period of oscillation v [7].

$$\nu = \langle \xi' \rangle = \tfrac{1}{2\pi}\int_0^{2\pi} \xi'(\tau)d\tau \tag{10}$$

The expression under the sign of the radical from equation has been transformed as (9)

$$\sqrt{\xi'^2 + \cos^2\tau} = \sqrt{\xi'^2 + (1-\sin^2\tau} = \sqrt{\nu^2 + 1 - \sin^2\tau} =$$
$$= \tfrac{1}{k}\sqrt{1-k^2\sin^2\tau},$$

where, $\int_0^{\frac{\pi}{2}} \frac{1}{\sqrt{1-k^2\cos^2\tau}} \cdot d\tau = K(\nu)$ - a complete elliptic integral of the first kind whose modulus is equal to k.

Finally it will be

$$\frac{d\nu}{d\tau} = \mu\left(\gamma - \frac{2\nu}{\pi}\cdot k(\nu)\cdot K(k)\right); \tag{11}$$

$$\varphi(\nu) = \frac{2\nu}{\pi}k(\nu)\cdot K(k). \tag{12}$$

Tables for finding elliptic integrals [17] can be used to determine the value of a function $\varphi(\gamma)$ (12), which has been shown in Table 1 and in Fig. 10

Table1

The value of function dependency $\varphi(\gamma)$ versus the dimensionless velocity v

□	0,0	0,1	0,2	0,3	0,4	0,5	0,6	0,7	0,8	0,9	1,0
$k(v)$	1	0,995	0,981	0,958	0,928	0,894	0,857	0,819	0,781	0,743	0,707
K(k)	∞	3,7	3,04	2,67	2,43	2,26	2,13	2,035	1,96	1,9	1,85
φ(ν)	0	0,234	0,379	0,488	0,575	0,643	0,698	0,743	0,780	0,809	0,834

v	1,0	1,1	1,2	1,3	1,4	1,5	1,6	1,7	1,8	1,9	2,0
$k(v)$	0,707	0,673	0,640	0,610	0,581	0,554	0,530	0,507	0,486	0,466	0,447
$K(k)$	1,85	1,82	1,78	1,76	1,74	1,72	1,70	1,69	1,68	1,67	1,66
$\varphi(v)$	0,834	0,855	0,870	0,888	0,901	0,911	0,918	0,927	0,934	0,940	0,945

Continuation of table.1

v	2,0	2,1	2,2	2,3	2,4	2,5	2,6	2,7	2,8	2,9	3,0
$k(v)$	0,447	0,430	0,414	0,399	0,385	0,371	0,359	0,347	0,336	0,326	0,316
$K(k)$	1,66	1,65	1,645	1,64	1,63	1,63	1,625	1,62	1,62	1,615	1,61
$\varphi(v)$	0,945	0,950	0,953	0,958	0,960	0,964	0,966	0,968	0,970	0,972	0,974

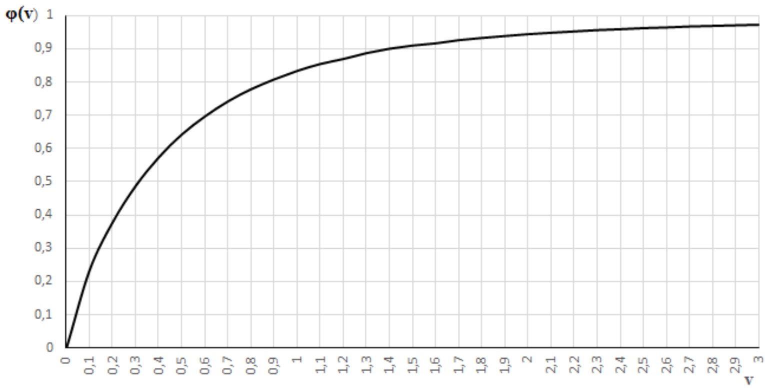

Fig. 10. The Graph of function dependency $\varphi(\gamma)$ versus the dimensionless velocity v

From the form of equation (12) and table 1 it is clear that when γ<1 there is a steady-state regime in the system, while $\gamma = \frac{\tan\alpha}{\tan\varphi} > 1$, a>1, so the motion will occur in the absence of transverse oscillations and will be equally accelerated, because the projection of the gravitational force on the x-axis will be greater than the friction force.

Fixed values of the average velocity ν have been found from the equation

$$\frac{2v}{\pi} \cdot k(v) \cdot K(k) = \gamma . \quad (13)$$

Given the values of $\gamma = \varphi(\gamma)$ in Table 1, the dimensionless average velocity of the bulk material in steady motion has been determined.

Table 2 shows the dimensionless average velocity ν of the bulk material motion on the chute surface, which performs transverse oscillations, from the inclination chute angle α to the surface and the friction coefficient f, and in Fig. 11. the response surface has been shown.

Table 2

The dependence value of the average velocity of the bulk material on the chute surface, from the inclination chute angle α to the surface and the friction coefficient f

f / α°	10	12	14	16	18
0,35	0,318	0,447	0,631	0,94	1,7
0,4	0,257	0,348	0,471	0,642	0,912
0,45	0,212	0,285	0,376	0,491	0,653
0,5	0,168	0,242	0,313	0,4	0,505

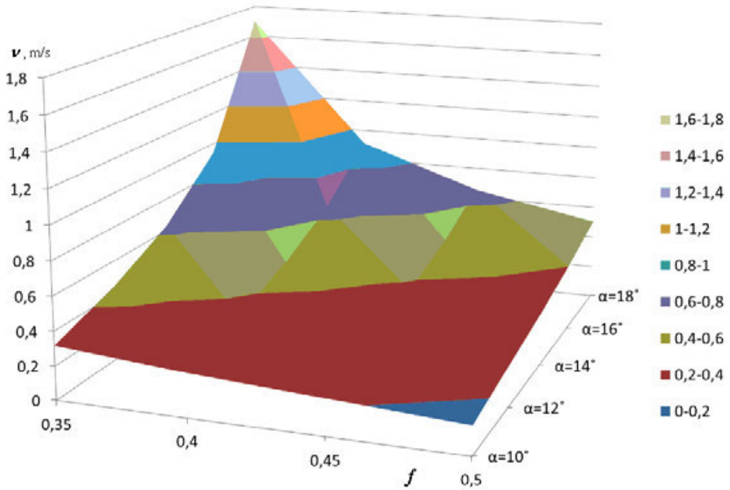

Fig. 11. The dependency response surface of the average velocity motion of the bulk material on the chute surface, which performs transverse oscillations, from the inclination chute angle α to the surface and the friction coefficient f

The dimensional average velocity U has been determined from equation

$$U = A\omega v. \tag{14}$$

Table 3 shows the results of theoretical and experimental studies of the bulk material motion on the chute surface (the material is barley, the coefficient of friction f=0,5 , oscillation amplitude A=0,001m, oscillation frequency ω=140 s^{-1}).

Table 3

The results of theoretical and experimental studies of the bulk material motion on the chute surface

α°	10	12	14	16
U_T, *m/s*	0,023	0,034	0,044	0,056
U_E, *m/s*	0,02	0,029	0,035	0,054
δ, %	15	17,2	12,8	7

A comparative analysis of the theoretical results confirmed their satisfactory agreement with the experimental data, in contrast to [2], where the average values of the velocity motion of a particle have been significantly different from the experimental data [18].

Conclusions

As a result of conducted theoretical researches, it has been found that by providing the chute an inertial conveyor of transverse oscillations during periods of time when the load slides on the surface of the chute, the performance of the vehicle is significantly improved. Thus, at the frequency of transverse oscillations ω=500 *rad/s* and the amplitude of oscillations A=0.005 *m*, the productivity of the conveyor is increased by 20%.

Currently, the program and methodology of additional experimental studies have been elaborated. Due to given actions the adjustments will be made to the construction of the inertial conveyor and rational geometric and kinematic parameters will be proposed, both as the chute motion driver, as well as the frequency and amplitude of the chute transverse oscillations, which ensure an efficient process of transporting bulk materials through this device.

On the consequences of provided studies of the bulk material motion in a rectangular chute, which performs transverse oscillations with high intensity, the dependence of the average velocity of the material on the inclination chute angle to the horizon, the sliding friction coefficient and the oscillation parameters have been obtained.

These dependencies can be used in engineering calculations of the motion of flowing bulk material while solving specific practical problems.

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INVESTIGATION OF THE MECHANISM OF DUST FORMATION DURING TRANSPORTATION AND TRANSSHIPMENT OF RAW MATERIALS OF A CEMENT-MINING ENTERPRISE USING DISCRETE-ELEMENT MODELING

Sakhno S.
Kryvyi Rih National University, Ph. D., Assoc. Prof., Ukraine

Korolyova J.
Kryvyi Rig Cement PrJSC, Specialist of Heals and Safety, Ukraine

Yanova L.
Kryvyi Rih National University, Ph. D., Assoc. Prof, Ukraine

Pischikova O.
Kryvyi Rih National University, Ph. D., Assoc. Prof, Ukraine

Abstract

The subject of the study - investigation of dust formation processes in the zone of cement flour transfer from one conveyor to another.

Research methodology – the study of the behavior of particles of different dispersion during loading from one conveyor to another using the Discrete Element Method.

The goal – determination of places of localization of dust emission and the behavior of dust in these places to improve dust collection processes.

Conclusion of the study. The proposed method made it possible to identify the features of the interaction of particles of different dispersion both among themselves and with the surrounding space. The method designated areas where the most significant dust emission occurs and to which special attention should be paid when developing dust cleaning measures. The research results apply to various sectors of the mining and processing industries.

1. Introduction

Cement production is one of the most essential and most developing industries. In the technological process of cement production, raw materials used many bulk materials: limestone, clay, chamotte, sand, marl.

Cement production is associated with some environmentally harmful technological processes [1-3]. During the transportation of bulk materials by both conveyors and pneumatic transport, during their unloading, sorting, significant dust formation occurs. Dusting occurs when drying bulk materials, burning clinker in furnaces, crushing and grinding raw materials and clinker, packing in bags, and shipping to the consumer. As practice shows, a significant residual amount of dusty ingredients of a high degree of dispersion released into the atmosphere.

Reducing dust generation requires significant energy consumption. In this regard, it is crucial to determine localization sites and the mechanism of dust formation. This article discusses the features of dust formation during the overloading of raw flour from one conveyor to another.

2. Object of study

Three digital models have developed to study the dust emission processes occurring in the raw mix overload zone. Modeling performed in the Rocky Academic software system.

The first digital model made following the parameters of the zone of transfer of the raw mix from conveyor No. 74 to conveyor No. 49 of Kryvyi Rig Cement PrJSC and modeled the movement of particles when they fall. To save computing resources, this model considered the laminar motion of particles larger than 10 mm only.

The second digital model developed based on the first. However, it models the behavior of finely divided particles of raw mix with a size of less than 0.1 mm.

The third digital model designed to study the behavior of fine particles of a raw mix in turbulent conditions.

Characteristics of materials and boundary conditions that accepted in the calculations shown in table 1.

Table 1

Characteristics of materials and boundary conditions for models 1-3

Parameter	Value	Parameter	Value
Raw mix		**Transfer hopper**	
Bulk Density	1300 kg/m^3	Density	7850 kg/m^3
Young's Modulus	1e+8 N/m^2	Young's Modulus	1e+11 N/m^2
Poisson's Ratio	0,3	Poisson's Ratio	0,3
Air		**Belt**	
Velocity	0 m/s	Velocity	2 m/s
Density	1,617 kg/m^3	Density	1400 kg/m^3
Viscosity	1,846e-5 Pa·s	Young's Modulus	1e+11 N/m^2
Temperature	300º K	Poisson's Ratio	0,3

An analysis of the results obtained in the study of models will provide an understanding of the processes of dust emission in the feed flour overload zone, which can further used to select the optimal parameters of the dust collection system.

3. Results of the research

Digital model No. 1. This model (Fig. 1) designed to study the movement of particles of the raw mix when it falls from conveyor № 74 to conveyor № 49. The model made by the drawings of the Kryvyi Rig Cement PRJSC raw mix transfer zone and includes a feed and a receiving conveyor, as well as a loading conveyor bunker.

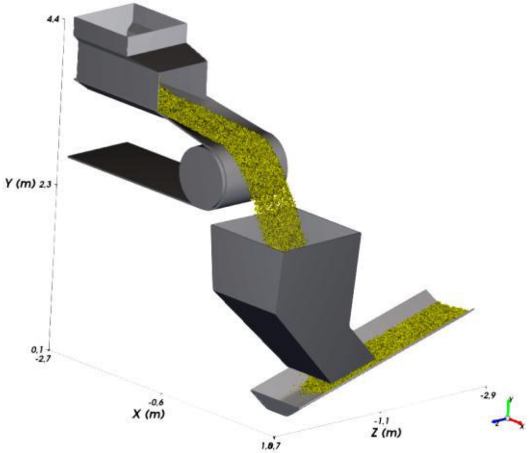

Fig. 1. General view of the digital Model № 1

A feed mix of 280 t/h supplied to feed conveyor № 74. The feed conveyor installed horizontally. Receiving conveyor № 49 in the horizontal plane is mounted perpendicular to the feeding conveyor, and in the vertical plane at an angle of 15°. The width of the conveyors is 1 m.

To reduce the computational resource consumption in the model were considered only particles with a size of 10-40 mm. In the calculation, particles in the form of a polyhedron with arbitrary orientation in space used. The particle size distribution showed in Figure 2.

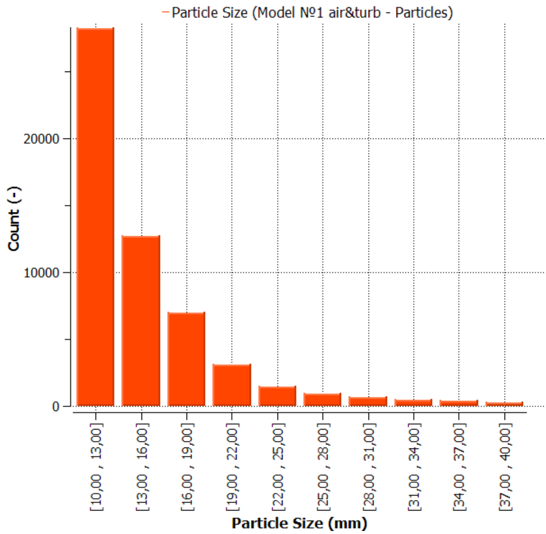

Fig. 2. The dispersed composition of particles in Model № 1

In the simulation, the interaction between particles and air taken into account. The simulation involved more than 55 thousand particles. In the research process, special attention paid to the trajectory of particle motion and the search for places of possible dust emission.

An analysis of the results showed that during the movement on the conveyor belt, the particle velocity is equal to the speed of the conveyor, and there is no mutual movement of particles. When falling from the feed conveyor until the moment of collision with the wall of the receiving hopper, the particles gain speed up to 7.6 m/s. After impact, the particles change the motion vector, and their speed depending on the place of contact with the walls of the reloading hopper is from 2 to 4 m/s (Fig. 3).

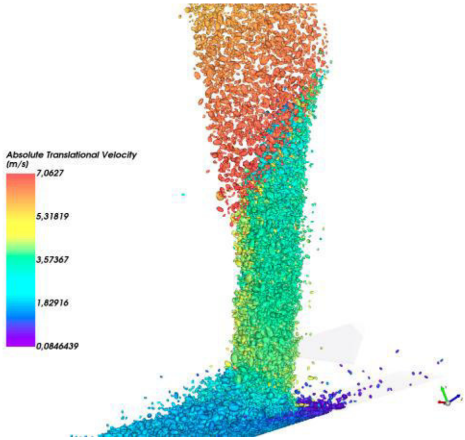

Fig. 3. The speed of movement of particles in the reloading hopper

After hitting the walls of the reloading hopper, the particles receive a rotational momentum. Moreover, as a rule, the smaller the particle size, the higher the rotational speed (Fig. 4).

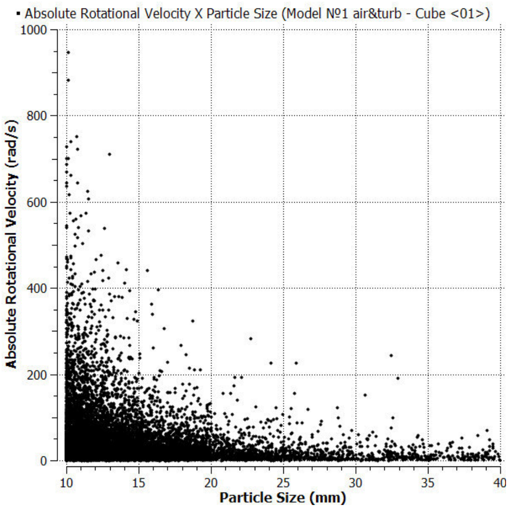

Fig. 4. Rotational speeds of particles in a reloading hopper

However, the number of revolutions of the particles is not significant (Fig. 5). In essence, 82% of the particles manage to make a maximum of 4 revolutions. As a rule, the higher the rotational speed, the smaller the number of revolutions the particle has time to make (Fig. 6).

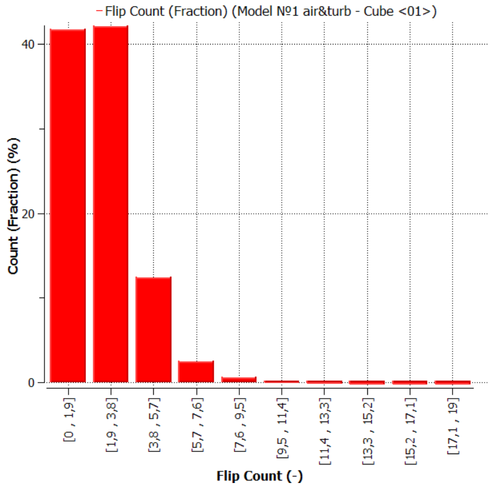

Fig. 5. The distribution of particles by the number of their rotations in the transfer hopper

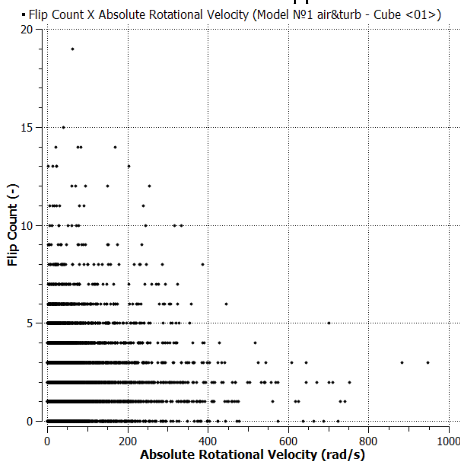

Fig. 6. The dependence of the number of rotations of particles on their angular velocity

The most significant number of rotating particles observed in the zone of their contact with the transfer hopper walls and in the area of particle fall on the conveyor.

Model № 1 allowed us to study the number of interactions between particles over time (Fig. 7). There are two lines in fig. 7. The black line shows the change in the number of particles over time. The number of particles continuously increases until the raw mix reaches the end of the receiving conveyor, and the number of particles entering the model becomes equal to the number of particles dropped from the model.

The green line shows the change in the number of frictional contacts between particles over time. Until time point 1, the particles move along the conveyor belt, and the number of contacts between them increases in proportion to the number of particles in the model. After time point 1 to point 2, there is a free fall of particles from the feed conveyor into the transfer hopper.

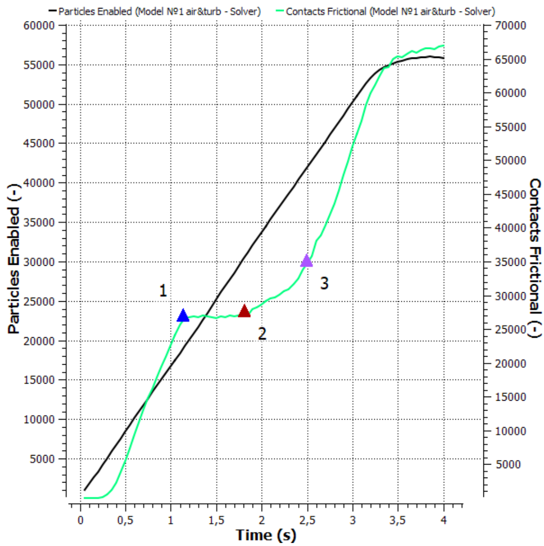

Fig. 7. The number of frictional contacts between the particles of the raw mix

During free fall, the particles practically do not touch each other; that is why line 1-2 is horizontal. In section 2-3, the particles interact with each other and with the walls of the transfer hopper, so the number of contacts increases. After time point 3, particles fall on the receiving conveyor, the speed leveled, and the number of friction contacts again increases in proportion to the increase in the number of particles in the model.

Conclusions on model No 1. Given the data obtained, the following dust emission zones can be distinguished:

a zone of free fall of particles from the feed conveyor into the transfer hopper. Due to the significant difference in the velocities of small and large particles, the mixture flow is stratified, and air turbulence, which formed due to the movement of large particles, carries small particles in space. In more detail, this mechanism will be considered in the second and third models;

the zone of interaction of particles with the walls of the reloading hopper. In this zone, there is a sharp decrease in the speed of movement of particles while giving them a rotational impulse. As a result, air vortices formed and the number of shock interactions between particles increases, which provokes significant dust emission;

zone of falling particles from the reloading hopper to the receiving conveyor. The particle velocity in this zone is 2 to 3 times higher than the speed of the conveyor belt. As a result, shock interactions between particles and their rotation occur, which leads to significant dust emission.

Digital model № 2. This model developed based on the first, but it investigated the features of the motion of spherical particles. The movement of particles was considered only in the zone of their fall in the receiving hopper to reduce model resource consumption. The dispersed composition of particles shown in Fig. 8.

During the simulation, the behavior of more than 3.5 million particles investigated. In the simulation, the interaction between particles and air taken into account. During the analysis, special attention was paid to the vertical velocity of the movement of finely dispersed particles, taking into account air resistance. The turbulent particle motion was not taken into account in this model to reduce the calculation time.

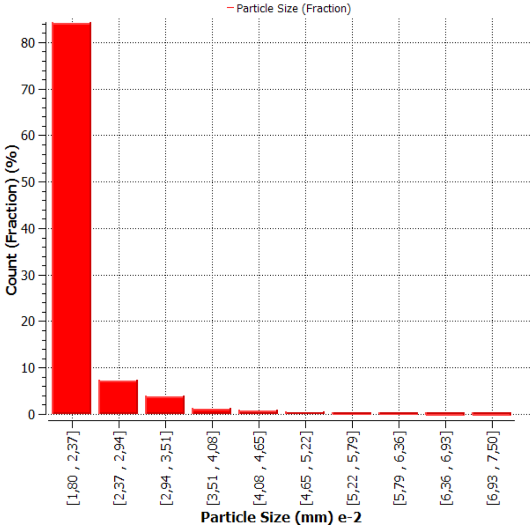

Fig. 8. The dispersed composition of particles in Model № 2

The behavior of the particles studied in the space in the center of the receiver limited by the size of 1×0.6×0.2 m, Fig. 9, to eliminate the influence on the speed of particles of the sidewalls of the receiver of raw flour.

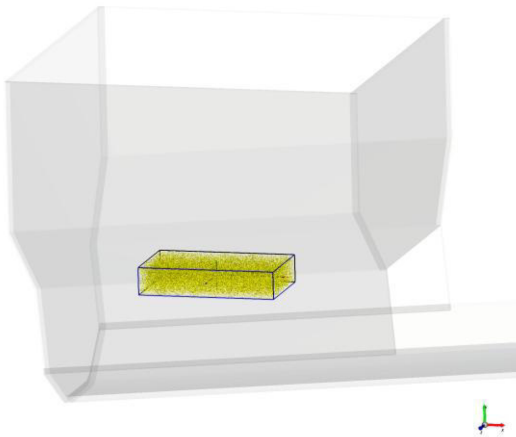

Fig. 9. Area of study of particle behavior

Because small particles reach the investigated space much later than larger ones, an analysis made of the velocity of the particles of the raw mix at different time intervals of the model. The results showed in Figure 10-11.

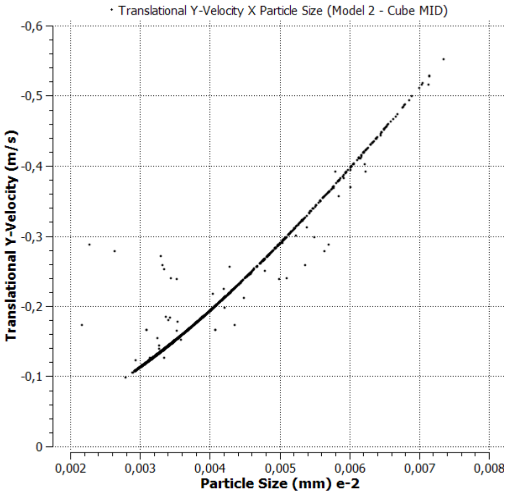

Fig. 10. Dependence of the rate of fall of particles on their size.3rd second of simulation

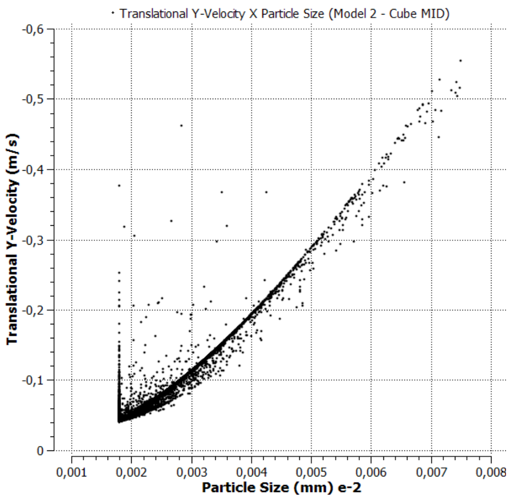

Fig. 11. Dependence of the rate of incidence of particles on their size. 16th second of simulation

An analysis of the results shows that for the main bulk of the particles, the dependence of the rate of fall on their diameter is quadratic, which is consistent with the well-known calculation formula [4]

$$V = d_r^2 g \rho_r / 18\mu$$

where V is the particle fall rate; d_r is the particle diameter, m; μ - dynamic viscosity, Pa·s; ρ_r - particle density, kg/m^3; g - gravity acceleration, m/s^2.

This dependence is especially noticeable in the graph of Fig. 10 formed under conditions when there was a small number of particles in the study zone. However, as the number of small particles increases in the 16th second of the simulation, the dependence becomes more blurred. For small particles, another line is clearly drawn, located slightly below the mainline, as well as a vertical line located in the zone of particles with the smallest diameter. Additional dependencies appear more clearly as the total number of particles of the raw mix increases (Fig. 11). The appearance of other dependences is associated with an increase in the number of frictional contacts between particles.

In some cases, larger particles carry smaller particles, which leads to a deviation of the velocity of small particles to higher values, in other cases, the collision of particles leads to a change in their motion vector, which reduces the vertical velocity component. Since this research model carried out without turbulence, the horizontal component of the velocity vector can arise only because of particle collisions. In Fig. 12 it is seen that the appearance of the horizontal velocity component more often occurs in small particles. Moreover, the lowest the particles, the higher the value of the horizontal component of the velocity vector.

Conclusions on model № 2. In general, the movement of fine particles without taking into account turbulence obeys known laws. However, with an increase in the density of particles in space, the particles begin to interact with each other.

The interaction or collision of particles occurs due to a significant, more than ten times, the difference in speeds of large and small particles.

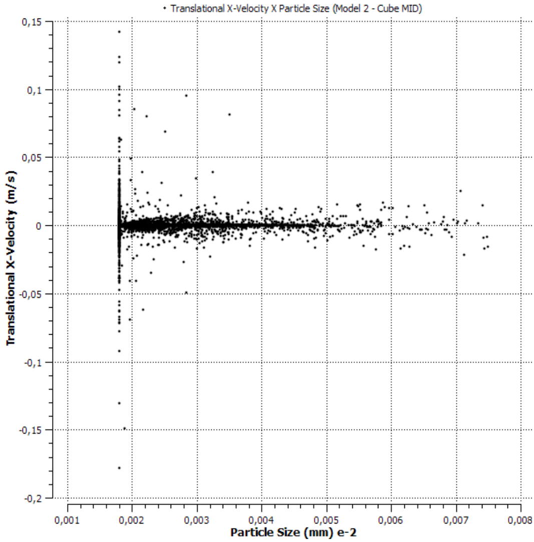

Fig. 12. Dependence of the horizontal component of the velocity vector on particle size

Because of the interaction, a horizontal component of the motion vector arises.

Moreover, the smaller the particles, the higher the frequency of occurrence of horizontal deviations, and the higher the value of the horizontal component of the velocity vector.

Digital Model № 3. This digital model includes a calculation area of 2×2×2 m in the upper part, of which there is a raw mix supply area (Fig. 13). The diameter of the feed area is 0.2 m. The feed rate of the material is 8.7 t/h.

During the simulation, the number of particles in the model initially increased and reached a maximum value of 130 thousand.

After three seconds, the material supply turned off, and in two seconds, the number of particles in the calculation space reduced to zero (Fig. 14).

In this model, the turbulence of particle motion taken into account during the simulation in contrast to the 1st model.

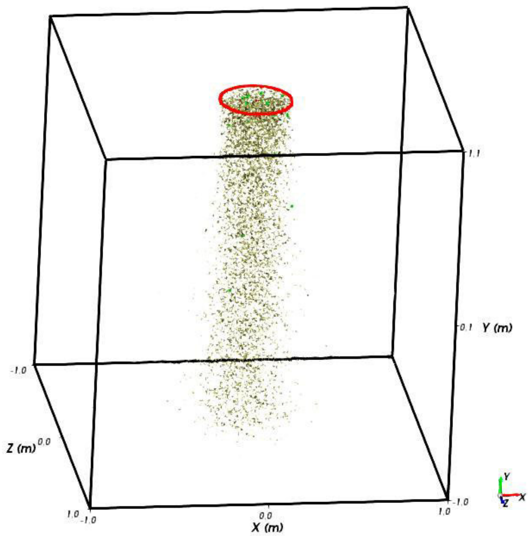

Fig. 13. Model № 3

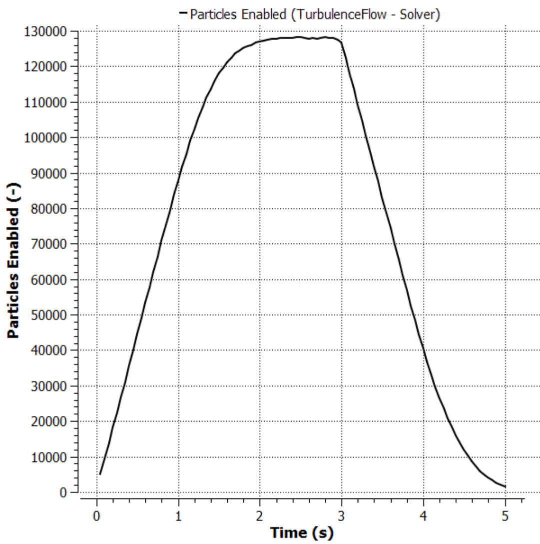

Fig. 14. The number of particles of raw flour that participated in the calculation of Model №3

Special attention paid to the influence of turbulence and collisions on the deviation of the direction of motion of particles from vertical. Simulation results considered at four specific time points of these model:

0.25 seconds - the beginning of the loading of the particles of the raw mix. All fractions represent the composition of the mixture;

1.0 second - the fastest particles have reached the bottom of the computational domain, while the slowest are in the middle of the distance;

2.0 second - all particles reached the bottom of the computational domain. Since some of the larger and faster particles left the computational area, the fractional composition shifted toward smaller particles;

4.5 seconds - the feed mixture stopped 1.5 seconds ago. The upper part of the "dome" of particles is below the middle of the computational domain. Small particles predominantly represent the fractional composition.

The number of particles of various fractions in the computational space of the model at different periods shown in Fig. 15.

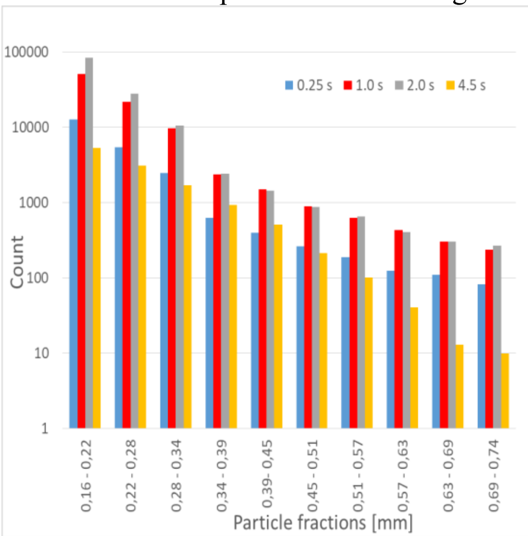

Fig. 15. The number of particles of various fractions in the computational space of Model № 3 at multiple points in time

The changing in the average particle size of the model with time seen in Fig. 16.

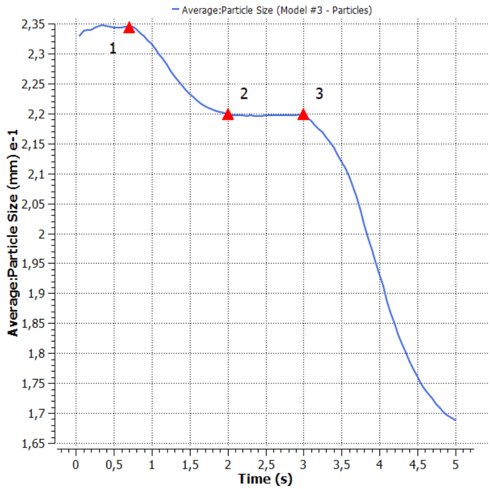

Fig. 16. Changing in the average particle size of the model over time

In fig. 16, point 1 corresponds to the achievement of large particles of the lower boundary of the calculation model, while increasingly smaller particles are still within the calculation zone. Position 2 corresponds to the moment when all particles have reached the lower boundary of the calculation zone, and the number of particles arriving in the model is equal to the number of decreasing ones. Point 3 corresponds to the moment the particle feed turned off. Due to the faster exit from the calculation zone of large particles, the average particle size is continuously decreasing.

An analysis of the results showed that deviations from the vertical direction of particle motion occur at all time stages of the simulation. At the initial points of the simulation, individual particles are characterized by high horizontal, up to 10 m/s, and even reverse vertical ascending velocities that quickly decay (Fig. 17). This phenomenon can be explained by tangential collisions of large particles with small ones and giving the latter a significant horizontal momentum.

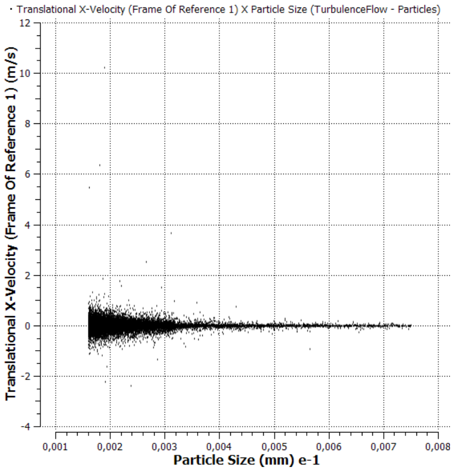

Fig. 17. Horizontal speeds of particles of various sizes at the time point of simulation 0.25 s

Subsequently, with an increase in the number of particles, the quenching of high velocities of small particles occurs due to their mutual collision. At the same time, the overall speed spread increases due to the turbulent motion of particles, Fig. 18.

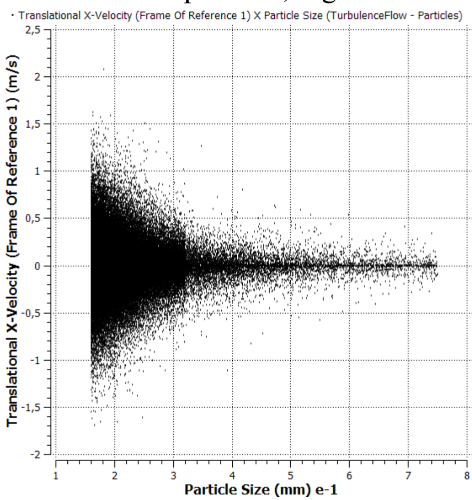

Fig. 18. Horizontal speeds of particles of various sizes at the time point of simulation 1.0 s

The dispersion of horizontal velocities sharply increases for particles less than 0.3 mm in size. This is because, with a decrease in particle size, their number, and, correspondingly, the number of their interactions with each other increases.

Particles smaller than 0.3 mm are also characterized by a significant scatter in vertical velocity, Fig. 19. This confirms that particles smaller than 0.3 mm can have a significant mutual effect due to turbulent phenomena. Given the considerable velocity dispersion for particles 0-0.3 mm in size, substantial dust emissions should be expected for given particle size. An analysis was made of the distribution of particles in space to confirm this assumption.

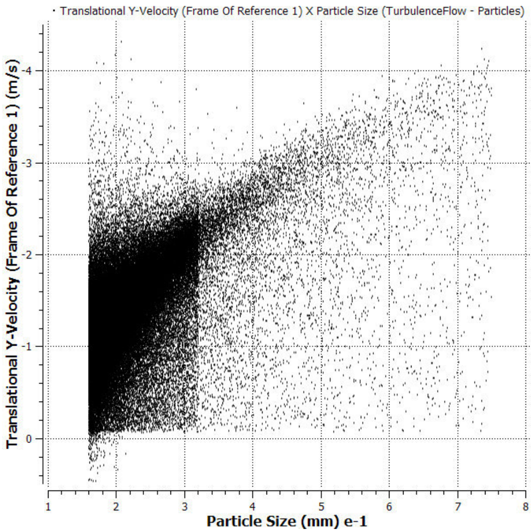

Fig. 19 . The relationship of the vertical component of the particle velocity and particle size

Fig. 20 shows the dependence of the magnitude of the dispersion of particles on their size. As expected, the most significant scatter is characteristic of particles less than 0.3 mm in format.

The mass distribution of the raw mix over the scattering surface shown in Fig. 21.

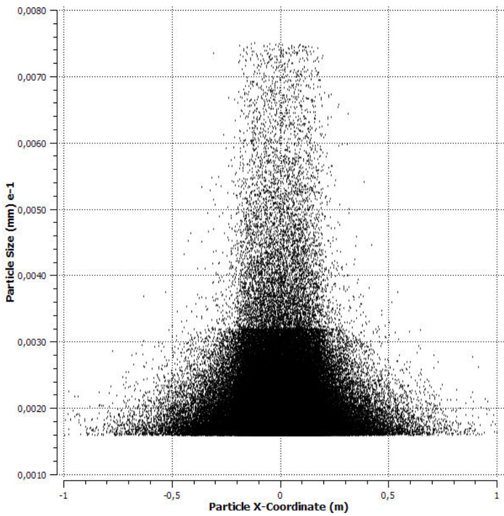

Fig. 20. Dependence of the size of the dispersion of particles on their size (time point - 3 s)

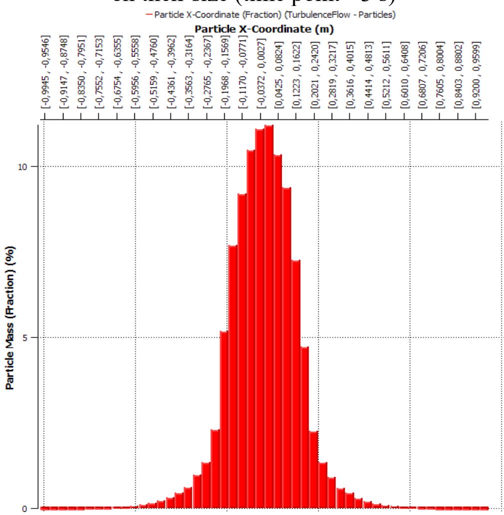

Fig. 21. The distribution of the mass of the raw mix on the surface of the spread

The histogram shows that outside the cylindrical region with a diameter of 0.2 m falls less than 5% of the mass of raw flour, and outside the area with a diameter of 0.3 m less than 2% of the weight of the raw mix. In this case, the original size of the particles extending beyond the region with a diameter of 0.3 m is less than 0.2 m.

The Latis - Boltzmann method has used to check the impact of the particles on the air movement. The calculation results show that when moving, particles have a significant effect on the air. In the center of the particle flow, the air velocity reaches 1.4 m/s with a particle velocity of 6.3 m/s (Fig. 22). At a distance of 0.2 m outside the particle flow, the absolute air velocity is 0.4 m/s.

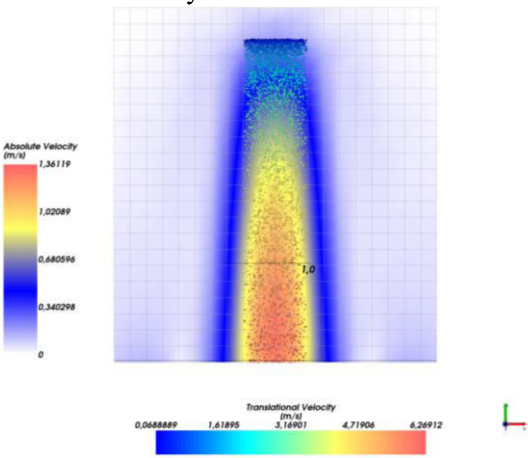

Fig. 22. Absolute speed of air. Vertical section

At the same time, a horizontal movement of air at a speed of about 0.1 m/s, directed inward to the particle stream, is observed near the flow of particles of the raw mix.

The direction of the air velocity vectors at different points of the model shown in Fig. 23.

The analysis results allow us to conclude that the stabilized particle flow is in an air cage. Outside the stream of particles, air tends to its surface.

At the same time, the horizontal component of the air velocity is comparable with the free-fall velocity of the small particles. This "drives" small particles into the stream.

A steady airflow inside the particle stream also entrains small particles and evens their speed with larger ones.

Due to the considered phenomena, dust emission is minimal in the area of the steady flow of particles of the raw mix.

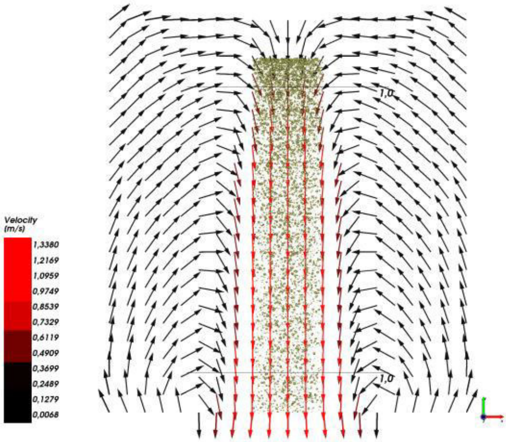

Fig. 23. The direction of the air velocity vectors along with the model

Conclusions on the analysis of Model № 3

When the particles of the raw mix move, they disperse in the horizontal plane. Dispersion occurs both due to the collision of particles with each other and due to turbulence.

Deviation from the vertical and the most significant variation in the velocity of motion is characteristic of particles with a diameter of less than 0.3 mm. The most substantial deviation, and therefore the most significant role in dust formation, is played by particles with a size of less than 0.2 mm.

For most particles less than 0.2 mm in size, the vertical velocity does not exceed 0.5 m/s.

The movement of particles of the raw mix causes a horizontal and vertical flow of air. The air velocity inside the particle stream reaches 1.4 m/s. Dust formation reduced due to the horizontal movement of air directed towards the center of the steady stream of particles.

4. Conclusions

In the process of overloading the raw mix from one conveyor to another, several zones of intense dust formation arise. The first is at the place of departure of the raw mix from the feed conveyor. The second is the place of interaction of the raw mix with the walls of the transfer hopper. The third is in the zone of particles entering the receiving conveyor.

The particles of the raw mix have significant differences in speeds in both vertical and horizontal directions. In the zone of interaction of particles with the reloading hopper walls, the rotational motion added to the translational movement of the particles. In this zone, the most intense dust generation occurs.

Particles of the raw mix during movement form air currents. Peculiarities of air movement help to reduce dust emission in stabilized vertical flows of the raw mix.

The highest dispersion and dust emission observed for particles less than 0.2 mm in size. Therefore, when choosing the parameters of dust cleaning devices, it is necessary to focus on the free fall rate of particles of this size.

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IMPROVEMENT OF IODINE EXTRACTION TECHNOLOGY FROM CONCOMITANT WATERS OF THE OIL-FIELDS

Yatskov M.V.
National University of Water and Environmental Engineering (NUWEE), Professor, Candidate of Technical Sciences, Professor, Department of Chemistry and Physics, Ukraine

Korchyk N. M.
National University of Water and Environmental Engineering (NUWEE), Associate Professor, Candidate of Technical Sciences, Associate Professor, Department of Chemistry and Physics, Ukraine

Budenkova N. M.
National University of Water and Environmental Engineering (NUWEE), Associate Professor, Candidate of Chemical Sciences, Associate Professor, Department of Chemistry and Physics, Ukraine

Mysina O. I.
National University of Water and Environmental Engineering (NUWEE), Senior Lecturer, Department of Chemistry and Physics, Ukraine

Abstract

Systematic analysis of iodine extraction technologies from highly mineralized waters including concomitant waters of oil-fields were done. It is proved that the estimation of the oxidation-reduction equilibrium in iodine extraction technological systems by the value of Eh can be conducted only at identical values of pH. At each stage of the technological process optimal parameters (pH, Eh, rH_2) were determined through a potentiometric method of analysis. Research results are: significant reduction in sulfuric acid consumption; reduction in toxic wastes formation; reduction in sulfate contamination of the final product. The high iodine extraction effect in crystal form is achieved by exclusion of side processes. Results are implemented in research-industrial equipment for iodine extraction from underground highly mineralized water. The mass fraction of iodine in the final product is 99,4-99,6%.

Keywords: systematic analysis, typical processes, concentrates, marketable products, iodine, iodides, oxidation-reduction potential, hydrogen ion concetration pH, chemisorber, desorber, crystallizer.

Introduction

Iodine demand in Ukraine is about 30-40 tons per year (*t*/*y*). Development of iodine extraction methods from wastes including concomitant waters of oil-fields and industrial production waste waters is topical. This water is highly mineralized and can be harmful to the environment. This problem is especially urgent for the Zakarpattya, Ivano-Frankivsk and Vinnytsia regions where oil-fields are located.

Iodine extraction technologies typically include a combination of elementary processes, including those typical for conversion and separation. From a systematic analysis point of view such technologies should be considered as complicated technological systems that include basic elements for the iodine extraction in the form of concentrates from natural highly mineralized waters and auxiliary elements for the raw materials pretreatment for the extraction of iodine from concentrates in the form of marketable product which include typical processes. Systematic approach in technological systems makes it much easier to find and use the necessary information to develop and implement any modification of the technological process.

Research aim consists of modification and improvement of iodine extraction from highly mineralized waters technological system including concomitant waters of oil-fields within systematic analysis by exclusion of side processes.

Highly mineralized underground natural water of the Ukrainian western regions were used in this study, the composition of which is shown in Table 1.

Table 1

Composition of highly mineralized natural waters for the iodine extraction

№ з/п	Parameters	Unists of measarument	Value
1	Density	g/m^3	1,23-1,27
2	pH		2,4-6,85
3	Eh	mV	-100÷100
4	rH_2	V	1,35-17,2
5	Salt content	g/dm^3	239-412
6	Calcium	mg/dm^3	32000-75630
7	Magnesium	mg/dm^3	6906-12920
8	Ferum	mg/dm^3	7100-8600
10	Sodium	mg/dm^3	33870-43510
11	Potassium	mg/dm^3	3000-9420
12	Sulfates	mg/dm^3	71-200
13	Iodides	mg/dm^3	5,92-59,2

Potentiometric analysis method was applied in study on the redox and acid-base parameters of highly mineralized water. It is proved that the estimation of the oxidation-reduction equilibrium in iodine extraction technological systems by the value of Eh can be conducted only at identical values of pH [1,2]. Therefore, in some elements of the iodine extraction technological system the following parameters are considered: pH, Eh, rH_2.

Table 1 data reveals that the dissolved iodine content is sufficient for cost-effectively extraction from the highly mineralized natural waters (concomitant waters of oil-fields).

Results of existing technologies systematic analysis

Since the highly mineralized waters iodine content (including concomitant waters of oil-fields) is negligible (0,001-0,005%), lower than its solubility in water, it is impossible to obtain iodine from such solutions in the form of crystals or soluble salts sediments by the action of oxidizing agents or other reagents. Therefore, the extraction of iodine is preceded by the obtaining of its saturated aqueous technological solutions which are further converting into marketable product. Iodine extraction technological systems include the following stages:

- preliminary impurities purification (naphthenic acids, ferum, salts, etc.);
- removal from highly mineralized natural waters (with preliminary iodide ions oxidation; without preliminary iodide ions oxidation; with simultaneous oxidation and removal of iodide ions);
- concentrates extraction and obtaining of final product [1,3].

The composition of undeground water is a very important factor for the the finished product purity. The most unwanted impurities are naphthenic acids salts and other organic substances contained in the concomitant waters of the oil-fields. Highly mineralized water coming from wells first passes through oil traps and then settles in natural or artificial pools.

At some plants, the water after settling is filtered through sand, mesh or other high-speed filters. The turbidity-free water is fed into a mixer where it is acidified with sulfate H_2SO_4 or hydrochloric HCl acid and then treated with an oxidizer.

During acidification free naphthenic acids are separating and floating to the surface, therefore at some plants, the water is settling after acidification as well.

Accordingly, the following typical processes are applied in pre-treatment technological systems:

- hydromechanical (settling, filtering);
- chemical (acidification, oxidation).

As a result of pre-treatment, the content of organic and inorganic components is reduced in highly mineralized waters (concomitant waters of oil-fields).

Iodine extraction methods from highly mineralized waters with pre-oxidation of iodide ions are:

- iodide ions oxidation by chemical reagents to elemental form followed by adsorption by activated carbon (sodium nitrite $NaNO_2$, chlorine, hypochlorites, iodates, etc. are used as oxidizers);
- electrolysis iodine extraction by iodide ions oxidation on graphite anode (iodine is adsorbed on the electrode);
- iodine extraction with iodide ions pre-oxidation by chemical reagents to elemental form followed by air desorption with concentrates obtaining (better known as air-desorption method).

Thereby, the typical processes that include considered technological systems are:

chemical;
electrochemical;
mass transfer.

As an example, on Fig. 1 technological systems for the iodine extraction from natural brines (highly mineralized waters) by air-desorption method are shown [1].

Iodine extraction methods from highly mineralized waters without preliminary iodide ions oxidation are:

- iodide ions extraction by sorption on ion-exchange resins (anionites);
- iodine extraction in the form of insoluble salts (copper(I) iodide CuI) during electrolysis;
- iodine extraction in the form of insoluble salts by chemical reagents (with formation of copper, argentum and hydrargyrum iodides).

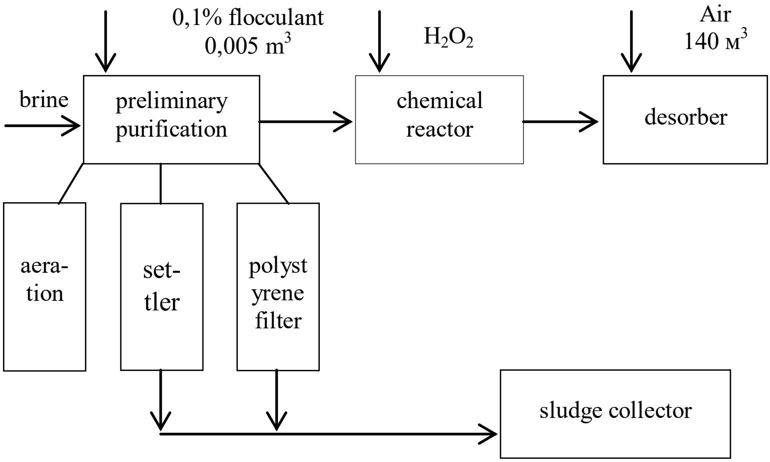

Fig. 1. Technological systems for the iodine extraction from natural brines by air-desorption method

Therefore, typical processes in the considered technological systems are:

mass transfer;

electrochemical;

chemical.

Iodine extraction method from highly mineralized waters with simultaneous oxidation and iodide ions extraction involves iodide ions oxidation by sodium hypochlorite NaClO.

Therefore, the typical processes that include considered technological systems are:

chemical;

mass transfer.

Obtaining concentrates is carried out at the following stages of iodine extraction from highly mineralized waters technological systems:

- Iodine re-extraction from an organic solvent is carried out by the following basic methods: a) three-phase method with metallic copper utilization; b) treatment with sodium sulfite solution, as a result of which elemental iodine is converted into aqueous solution in the form of iodides;

- iodine chemisorption is carried out by: a) from the iodine-air mixture by sulfur(IV) oxide in the presence of water vapor; b) absorption of iodine from the iodine-air mixture by alkali solutions;

- iodine desorption is carried out by: a) desorption from ion exchange resins by transferring iodine to the organic phase which is carried out by special solvents in the presence of ions Mn^{2+}; b) electrochemical desorption from iodine-coal which is based on the transmission of electric current through heated electrolytes (NaCl solutions, etc.) which contain saturated iodine-carbon. In this case, the iodine goes into solution, and then, due to the high temperature, it sublimates, which allows to achieve 100% extraction; c) thermal desorption is carried out by heating saturated iodine coal without air access at 200-400 °C; d) steam-thermal desorption based on the passage of superheated water vapor through saturated iodine carbon heated to 350-400 °C.

- iodine vatting is carried out as follows: first, iodine carbon is washed with hot water to remove salts and acids, and then heated to 90-120 °C with 10-15% sodium hydroxide NaOH or sodium sulfite Na_2SO_3 solution.

Thereby, typical processes that include these technological systems are:

mass transfer;

electrochemical;

thermal.

Iodine extraction from concentrate in the form of marketable product is carried out in the following technological systems:

- obtained alkaline iodide concentrate is treated with methane (formic) HCOOH or oxalate (oxalic) $H_2C_2O_4$ acids with the crystalline iodine extraction;

- obtained concentrate of iodide HI and sulfuric H_2SO_4 acids mixture is oxidized with further crystalline iodine extraction;

- the concentrate in the form of an alkaline solution is acidified and treated with an oxidizer (chlorine Cl_2, potassium chlorate $KClO_3$, hydrogen peroxide H_2O_2) to extract iodine in crystalline form;

- the preparation of iodide acid HI and its salts is carried out from mother solutions after iodine extraction in the form of crystals within treatment by barium chloride $BaCl_2$, resulting in the precipitation of barium sulfate $BaSO_4$ while the solution contains a mixture of hydrochloric and iodide acids (HCl and HI). Barium sulfate is filtered off and the aqueous acid solution is subjected to fractional distillation.

Therefore, the typical processes that include considered technological systems are:

chemical;

mass exchange;

thermal.

In terms of iodine extraction technologies systematic analysis, they should be considered as complex technological systems which includes:

- basic elements for the iodine extraction from raw materials (highly mineralized waters including concomitant waters of oil-fields) without iodide ion pre-oxidation for concentrates obtaining;
- auxiliary elements for the preliminary purification of raw materials (highly mineralized waters including concomitant waters of oil-fields) and iodine extraction from concentrates in the form of marketable product.

With the exception of the auxiliary system, namely "pre-treatment", the resulting product of iodine in the form of paste can be considered as raw iodine. With the exception of the auxiliary system for obtaining a product from concentrates, the latter may be considered as an intermediate for further processing in specialized industries.

Considered technological systems include the following typical processes:

chemical: acidification, alkalizing, reduction, oxidation, complexing;

mass transfer: extraction, absorption, adsorption, desorption, crystallization;

electrochemical: electrolysis;

thermal: heating, distilling.

It should also be noted that systematic approach utilization in the technological diagrams and processes can significantly simplify the search and usage of the necessary information for designing and implementation of any technological process modification.

Results of experimental studies

Existing technologies systematic analysis allowed us to determine the number of technological system elements which allow to provide [1]:

pre-treatment which is carried out by: highly mineralized water (pH=6,1, Eh=-100 mV, iodide ion concentration 20 mg/dm^3) from the well is fed to the aeration reactor and for treatment with flocculant. As a result of aeration processes, Eh increases (Eh becomes higher than +80 mV) and particles of ferum OH^- complexes are formed. This reduces the content of inorganic and organic components in raw materials (highly mineralized water) which reduce production volume and increase its level of contamination;

highly mineralized waters, after pre-treatment, are fed into a chemical reactor for iodide ion oxidation into free iodine in the presence of sulfuric acid H_2SO_4 by hydrogen peroxide H_2O_2 which acts as an oxidant

$$2I^- \rightarrow I_2; \qquad (1)$$

iodine extraction by air (desorption) provides an extraction effect of 80-89% which is 10-15% more in comparison with known methods;

iodine absorption (chemisorption) in a mixture of potassium hydroxide KOH and hydrogen peroxide H_2O_2 which under these conditions acts as a reduction agent

$$I_2 \rightarrow 2I^-; \qquad (2)$$

iodide ion re-oxidation into free iodine from an alkaline solution (concentrate, iodide concentration of 16-48 g/dm^3 depending on the alkali mass fraction) in the sulfuric acid presence and its extraction in the form of crystals (marketable product).

The main concept of the selected technical solutions is to provide high production volumes and its purity by reduction of the chemical reagents consumption which are the main contaminators of iodine and the environment. Therefore, as an oxidation reagent, it is recommended to utilize hydrogen peroxide H_2O_2, which does not form toxic secondary products as a carrier for air desorption. Furthermore, by providing optimal oxidation-reduction and acidic-alkalic parameters in the technological systems presented in Fig. 2, a reduction in the chemical reagents consumption is achieved.

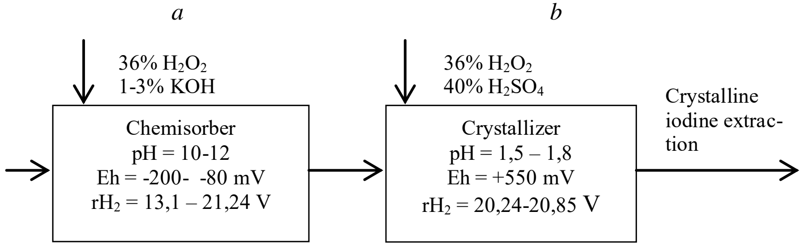

Fig. 2. Technological system includes:
a - element for alkaline concentrate solution extraction; *b* - element for crystalline iodine extraction (marketable product)

Based on the results of potentiometric titration in experimental-industrial conditions, the main parameters were determined at each stage of processing including reduction-oxidation and acidic-alkalic equilibriums (Table 2).

Table 2

State characteristics of the reduction-oxidation equilibrium in aqueous systems at different processing stages

Sampling point	pH	Eh, mV	rH_2, V
Well	4,7	-100	5,95
Clean water hose-pipe	4,5	+100	12,44
Chemical reactor	4,5	+100	12,44
Desorber	4	+400	21,79
Chemisorber	9,5	-200	12,1
Crystallizer	1,8	+550	22,57

The data presented in table 2 shown that highly mineralized water is characterized by a weakly acidic-reductive environment, pre-purified water by a weakly acidic-oxidizing environment, in a chemical reactor and desorber by acidic-oxidizing environment, in a chemisorber by alkalic-reductive environment, in a crystallizer by acidic-oxidizing environment.

Based on the potentiometric titration results and their differentiation (Fig. 3), iodine evaluation forms were determined from the values of pK for acid-base forms. Based on the obtained data it is possible to state the nature of iodine conversion depending on the acidic-alkalic (pH) and reduction-oxidation properties (Eh, rH_2).

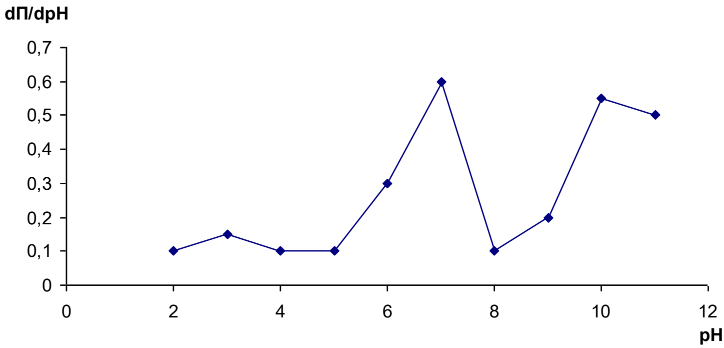

Fig. 3. Differential titration curve of an alkaline solution in the iodine chemisorption ((I^-) =1,69g/dm^3)

In chemisorber alkaline solution (pH =10-12) reductive environment is supported between -200 mV and -80 mV (Eh). To prevent iodate formation depending on conditions

$$3I_2+6KOH_{(hot)}\rightarrow 5KI+KIO_3+3H_2O; \quad (3)$$

$$I_2+2KOH_{(cold)}\rightarrow KIO+KI+H_2O. \quad (4)$$

Iodates have lower solubility and form sediments in the concentrate extraction process. To prevent it and increase the efficiency of absorption by the alkaline solution, it is recommended to carry out the process in the presence of a reducing agent - peroxide hydrogen

$$I_2+H_2O_2+2KOH\rightarrow 2KI+O_2+2H_2O. \quad (5)$$

The data presented in Fig. 4, 5, indicates the need in process regulation in order to shift the equilibrium state (C(I^-)=0,8 g/dm^3; 1,62 g/dm^3…).

The results of studies carried out under industrial conditions indicate that there is no clear dependence of production volume on pH and Eh, which characterize the acidic-alkalic and reduction-oxidation conditions. The reason is that the estimation of the process conditions by the values of Eh is possible only at the same pH values. It is proposed to use independent system state indicators pH and rH_2 to control the technological process $rH_2=(Eh/0{,}029)+2pH$. The pH is regulated by the dosing of KOH and the rH_2 is regulated by the addition of hydrogen peroxide, which allows the implementation of an automatic process control system within special computer programs. The range of potassium iodide alkaline solution extraction optimal values according to the experimental data is pH=11,1-11,4, rH_2=15,2-

15,9 V which provide stable process with the highest efficiency. Thereby, the high iodine extraction effect is achieved by exclusion of side processes.

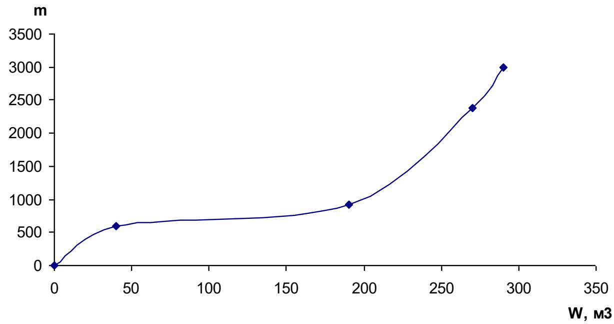

Fig.4. Absorbed iodine amount (m, g) depending on the treated highly mineralized water volume (w, m^3)

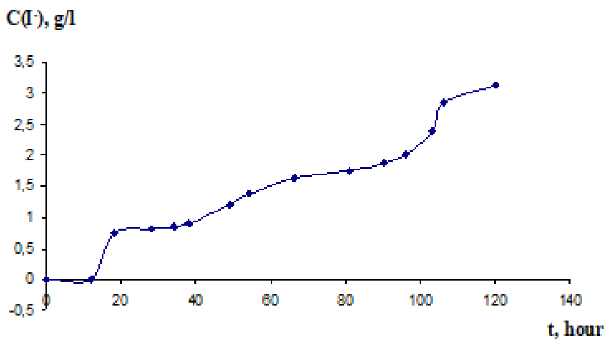

Fig.5. Iodide ions concentration change in an alkaline solution over the time

Thermodynamic calculations were done to obtain the starting materials optimum ratio for reaction (5) by the formulas

$$\Delta G = -RT \cdot \ln K_p; \tag{6}$$

$$\Delta G = \Delta H - T\Delta S, \tag{7}$$

description: ΔG - the Gibbs energy change, kJ/mol; *R* - universal gas constant; *T* - temperature; K_p - the equilibrium constant; *H* - enthalpy change, kJ/mol; *S* - entropy change, J/mol.

At a temperature of 298K, the change in the standard isobar-isothermal potential (Gibbs energy) is, kJ/mol

$$\Delta G_p^0 = -255{,}6, \tag{8}$$

The reaction occurs in the direction of the reaction products formation at a ratio of reagents I_2: H_2O_2: KI = 1: 0,13: 0,53 g/dm^3.

The equilibrium constant is

$$K_p = \frac{[KI]^2}{[H_2O_2]\cdot[KOH]} = 0{,}1857 \tag{9}$$

and Gibbs energy change, kJ/mol

$$\Delta G = -RT \ln K_p = 4170{,}5, \tag{10}$$

this means that the reaction is impossible.

The reaction is possible if ΔG<0 which is corresponding to K_p=0,66, then the mass ratio between KOH and H_2O_2 should be equal to 5,6:1.

For the iodine extraction in the crystalline form, the interaction between potassium iodide KI and acidified solution H_2O_2 is used

$$\mathrm{H_2O_2 + 2KI + H_2SO_4 \rightarrow I_2 + K_2SO_4 + 2H_2O}; \tag{11}$$

an interesting example is the interaction between iodine at pH=1 with hydrogen peroxide

$$\mathrm{I_2 + 5H_2O_2 \rightarrow 2HIO_3 + 4H_2O}, \tag{12}$$

as well as the further interaction with hydrogen peroxide at pH=2 with iodine extraction

$$\mathrm{2HIO_3 + 5H_2O_2 \rightarrow I_2 + 5O_2 + 6H_2O}. \tag{13}$$

The advantage of this method is that hydrogen peroxide H_2O_2 is used as the oxidant and the reducing agent which depending on the environment (pH and rH_2) can be both an oxidant and a reducing agent which greatly simplifies the technological process [1].

It is established that at system redox potential lower than 550 mV, the process of excess iodine oxidation to iodate IO_3^- is possible, and therefore it is necessary to strictly regulate the water systems active reaction, in particular pH and rH_2 parameters.

For reaction (11) by the Nernst equation

$$\varphi = \varphi^0 + 0{,}059 \cdot \lg[H_2O_2] - 0{,}059\,\mathrm{pH} \tag{14}$$

at pH values: 0,5; 1,0; 1,5; the accordingly calculated potential values are: φ=1,72; 1,69; 1,67; 1,63.

Taking into account reduction-oxidation systems (3-5 and 11-13), the theoretically calculated surplus of H_2SO_4 is 32.5% whereas in many technologies 100-200% surplus is used. The results of the studies indicate that smaller than calculated acid addition quantities is unable to provide optimum pH value throughout the process.

Studies in industrial conditions showed that at pH<1,8 production volume is reduced due to its excess oxidation and the iodic acid formation according to equation (12). Furthermore, hydrogen peroxide H_2O_2 is an unstable reagent: oxidative decay is possible in acidic environment $H_2O_2 \rightarrow H_2O + O$, in alkalic - reductive decay $H_2O_2 \rightarrow O_2 + 2H$. Therefore, it is promising to utilize lactic acid which is stable and, depending on the conditions, performs either reductive or oxidizing properties. Stage (5) is described by the following reactions

$$\begin{gathered} I_2 + 2KOH \rightarrow KI + KIO + H_2O \\ 2KIO + CH_3CH(OH)COOK \rightarrow CH_3COOK + 2KI + CO_2 + H_2O \end{gathered} \quad .(15)$$

At a molar ratio of I_2: KOH: lactic acid 1: 1,1: 1,1, the pH drops from 10,4 to 6,2, with the Eh change from - 234 mV to +318 mV. For stage (11), the chemistry of the process can be represented

$$CH_3CH(OH)COOH + 2KI + H^+ \rightarrow CH_3CH_2COOK + I_2 + H_2O + K^+ .(16)$$

Change of the system parameters: decrease in pH from 6,2 to 1,5 and increase in Eh from +318 mV to +468 mV. As an intermediate product, a Lugol's solution is formed which disappears at low pH values.

Conclusions

In terms of iodine extraction technologies systematic analysis, they should be considered as complex technological systems which includes:

- basic elements for the iodine extraction from raw materials (highly mineralized waters including concomitant waters of oil-fields) without iodide ion pre-oxidation for concentrates obtaining;
- auxiliary elements for the preliminary purification of raw materials and iodine extraction from concentrates in the form of marketable product.

With the exception of the auxiliary system, namely "pre-treatment", the resulting product of iodine in the form of paste can be considered as raw iodine.

With the exception of the auxiliary system for obtaining a product from concentrates, the latter may be considered as an intermediate for further processing in specialized industries.

The chemisorber technological system parameters regulation by the values of Eh and pH is complicated, due to the Eh value dependence on the pH values. It is proposed to use independent system state indicators pH and rH_2 to regulate the process, while the pH is regulated by KOH dosage and rH_2 is regulated by the H_2O_2 dosage.

The range of potassium iodide alkaline solution extraction optimal values according to the experimental data is pH = 11,1-11,4, rH_2 = 15,2-15,9 V.

The optimal parameters for the oxidation and formation of crystalline iodine are: pH=1,8, Eh=+550 mV, rH_2=22,57 V. In our opinion, it is promising in the iodine extraction technological systems instead of hydrogen peroxide which is unstable, to utilize lactic acid which is stable and, depending on the conditions, performs either reductive or oxidizing properties and can be utilized in chemisorption processes (concentrate extraction) and in the crystallization processes (obtaining a marketable product).

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FEATURES OF METHANE RELEASE IN THE WING OF A MINE FIELD DURING MINING OF GAS-BEARING COAL SEAMS

E. Filatieva
Postgraduate student, Volodymyr Dahl East
Ukrainian National University, Ukraine

V. Sokolenko
Candidate of technical science, Associate Professor,
Volodymyr Dahl East
Ukrainian National University, Ukraine

A. Oleynichenko
Postgraduate student, Volodymyr Dahl East
Ukrainian National University, Ukraine

M. Filatiev
Doctor of technical science, Associate Professor,
Volodymyr Dahl East
Ukrainian National University, Ukraine

Abstract

Nowadays, gas emission has been studied, in most cases, within separate extraction sites and preparatory mine workings. The process of gas emission outside the exploited extraction sites under the influence of displacement activation of the undermined coal and rock stratum have not been studied thoroughly. This is not reflected in the regulatory documents on the issues of predicting gas emission.

Methane release from the undermined sources within and outside the extraction sites is determined by the degree of the stope works development both at the exploited site and in the mine field wing under the influence of the mined-out space of the stopped longwall faces.

Methane release under the influence of rock displacement activation in some cases leads to unpredictable situations at present. They are conditioned by additional influx of an unpredictable gas amount and the lack of information about mine workings, where this gas emission is possible.

Monitoring of changes in gas emission in all mine workings and degassing wells has been conducted in the course of mining the extraction sites, from the beginning of their exploitation to the end of the stope work operations, as well as in the mine field wing. In full such observations have been conducted at “Gazeta Izvestia” Mine in the mine field wing when mining a gas-bearing anthracite seam. The total observation duration was 94 months. During this period, eleven extraction sites have been mined by panels to the rise.

The experimental data obtained made it possible to determine the patterns of changes in gas emission from the undermined coal and rock stratum within the exploited extraction sites and beyond their boundaries in the mine field wing.

Gas emission from the coal and rock stratum within the extraction site is directly proportional to the area of mined-out space, the total coal output from the extraction panel, the average daily coal output, and the average velocity of the stope face advance. The most convenient and intuitive parameter for determining the total amount of gas released from the undermined stratum within the extraction site and the average level of methane release is the specific gas emission per unit area of the mined-out space.

Gas emission outside the exploited extraction site from the mined-out space of the stopped longwall faces does not occur in case of incomplete earth's surface undermining. When the complete earth's surface undermining is achieved, the ratio of gas emission within the extraction site to methane release from the mined-out space of the stopped longwall face directly proportionally depends on its dimensions.

Keywords: methane release, coal seams, mine workings, extraction site, mine field, stope workings, rock development, activation of displacement, mined-out space, area, specific gas emission.

Introduction. Until now, the change in methane release during the gas-bearing coal seams mining has been studied, in most cases, when exploiting the separate extraction sites. For this reason, absolute (m^3/min) or relative gas emission (m^3/ton) for separate extraction sites and preparatory mine workings are considered as the main predicted values [1]. According to this regulatory document, methane release within the mine field (wing) is determined as the simple sum of gas emission into the mine workings of separate extraction sites and preparatory faces.

The main gas emission source in the overall gas balance of the mine is the stope workings. As a rule, their share is 90 percent or more. In turn, the predominant gas emission at the extraction site occurs from undermined coal seams and host rocks. Methane release from these sources to a considerable extent, other conditions being equal, depends on the degree of stope works development, both within the extraction site and the entire mine field [2]. Gas emission from undermined sources in the studied case depends on the rocks displacement processes under the influence of stope works and can occur both within the extraction site and beyond its boundaries into all-mine workings. The main influencing factor of gas emission into all-mine workings is the displacement activation of the undermined coal and rock stratum. Methane release under the influence of rock displacement activation at a certain stage of the stope works development in the mine field can lead to unpredictable situations at present. They are conditioned by additional influx of an unpredictable gas amount and the lack of information about mine workings, where the methane release is possible under the influence of undermined rocks displacement activation. These circumstances are not taken into account by the current regulatory document [1] due to insufficient knowledge of gas emission from the undermined coal and rock stratum depending on the degree of the stope works development within the mine field boundaries. Research in this direction is relevant, since the effectiveness of measures for the safe development of gas-bearing coal seams, which help to reduce the accident rate in coal mines associated with outbreaks and explosions of methane-air mixture, largely depends on their results.

The **purpose** of this work is to determine the dependences of gas emission on influencing factors during the stope works development within the boundaries of the extraction site and the mine field on the basis of the experimental data of the extraction sites mining.

To achieve the purpose, the following tasks should be solved:

- in the course of mining the extraction sites in the mine field, the following is set for each of them: the length of the longwall face and the extraction panel, the area of the mined-out space, the total coal output during the period of site exploitation, the average daily coal output for the entire period, the average velocity of the stope face advance, the total amount of gas released, the average absolute and relative gas emission, the amount of gas released within the extraction site and beyond its boundaries;
- a change in the absolute and relative gas emission from influencing factors is determined in the course of mining the extraction sites in the mine field;
- a change in the nature of gas emission is analysed depending on the influencing factors within each extraction site and compared with similar dependences for the mine field wing.

The methodology provides for monitoring a change in gas emission into all mine workings of the mine field wing and degassing wells in the course of mining the extraction sites, from the beginning of their exploitation to the end of the stope work operations. Periodic measurements of methane flow rate in mine workings and degassing wells were conducted using portable devices every 1-5 days. Measurements in certain points of mine workings were duplicated by the readings of the automatic gas protection (AGP) service. In full such observations have been conducted at "Gazeta Izvestia" Mine of Donbasantratsyt, DP in the mine field wing when mining a gas-bearing anthracite seam $l_2^{в}$ with a thickness of 0.9 m in the mine field heaved area. The total observation duration was 94 months. During this period, eleven extraction sites have been mined by panels to the rise. The order of their mining and the mine field cutting was significantly affected by the presence of discontinuous geological faults (Karlovsky and Sofievsky faults). This influence consists in an increase in the natural gas content of the mined seam as it recedes from the discontinuous faults and the necessity to prepare for exploitation of the sites with different dimensions of longwall faces and mined-out extraction panels.

Main Body. The gas content of the mined seam at each extraction site significantly decreases with the stope faces removal from the face entries. This is caused by a decrease in the depth of mining and the approach of the stope faces to the flash gas zone of the anthracite seam when it is mined to the rise. For this reason, the analysis (Table 1) for each extraction site is based on the accepted average values of anthracite gas content between their indicators for face entries and stopped stope faces.

Table 1

Exploitation indicators of longwall faces during mining the seam $l_2^{в}$ at "Gazeta Izvestia" Mine

Longwall face	Mining-and-geological and mining-engineering conditions of longwall faces mining					Exploitation indicators of extraction sites			
	Distance of the face entry from Sofievsky fault, m	Average natural gas content of the mined seam, m^3/t.g.m.	Mined panel length, m	Longwall face length, m	Mined-out space area, m^2	Total coal output, thousand tons	Average daily coal output, tons	Average velocity of the stope face advance, m/day	Time period of mining the extraction site, days
	1	2	3	4	5	6	7	8	9
1st western	90	11.7	1422	180÷100	185811	280.5	236.9	1.2	1184
1st bis western	between faults	9.2	775	100	69924	153.6	180.3	0.8	913
2nd western	270	13.6	1186	200	237200	389.6	914.6	2.8	426

Continuation of table 1

2nd bis western	470	24.0	279	200	51598	78.4	197.5	0.7	397
3rd western	670	18.2	1559	215	335185	535.3	1099.1	3.2	487
4th western	885	21.9	1491	210	313110	493.8	1080.6	3.3	457
5th western	1095	23.4	1421	216	307922	492.8	1014.0	2.9	486
6th western	1311	25.4	1169	230	268870	472.6	819.0	2.0	577
7th western	1541	24.6	1309	230	301070	493.8	901.1	2.4	548
8th western	1741	31.0	787	215	169205	265.7	874.0	2.6	304
9th western	1956	34.8	329	250	82250	116.8	320.9	0.9	364

Continuation of table 1

Exploitation indicators of longwall faces during mining the seam $\lambda_2^{в}$ at “Gazeta Izvestia” Mine

Information on gas emission						Amount of gas released during extraction site exploitation, thous. m3		mine field wing outside the zone of geological faults	Average gas emission from sources outside the extraction site, m^3/min	Average gas emission within the extraction site per $1m^2$ of the mined-out space	Average absolute gas emission within the extraction site, m^3/min	Note
Total amount of gas released, thousand m^3	Average absolute gas emission, m^3/min	Average relative gas emission, m^3/ton	Average gas emission per $1m^2$ of mined-out space	Maximum gas emission, m^3/min	Coefficient of variation	Within the extraction site	Outside the extraction site					
10	11	12	13	14	15	16	17	18	19	20	21	
4698.7	2.8	16.8	25.3	10.4	3.7	4698.7	0.0	-	0.0	25.3	2.8	
2546.4	1.9	16.6	36.4	6.3	3.3	2546.4	0.0	-	0.0	36.4	1.9	
13560.7	22.2	34.8	57.2	55.2	2.5	13560.7	0.0	200	0.0	57.2	22.2	
4665.7	8.2	59.5	90.4	20.0	2.4	4665.7	0.0	200	0.0	90.4	8.2	Start of exploitation in the 3rd western longwall face has not influenced on gas emission
16670.7	23.8	31.1	49.7	48.9	2.1	16670.7	0.0	415	0.0	49.7	23.8	
15513.8	23.6	31.4	49.5	50.0	2.1	15513.8	616.9*	625	0.9	49.5	23.6	Start of exploitation of the 4th western longwall face has not influenced on a change in gas emission
22042.2	31.5	44.7	71.6	57.0	2.3	17093.4	4948.8	841	7.1	55.5	24.4	
23041.2	27.7	48.8	85.7	66.0	2.4	16407.2	6634.0	1071	8.0	61.0	19.7	
16407.2	20.8	33.2	54.5	56.7	2.7	16407.2	-	1301	-	54.5	20.8	
12764.6	29.2	48.0	75.4	80.0	2.7	6918.4	5846.2	1516	13.4	40.9	15.8	
8911.6	17.0	76.3	108.3	29.0	1.7	3843.3	5068.3	1766	9.7	46.7	7.3	

* - gas emission into wells at the site of the 3rd western longwall face

Analysis of the experimental and calculated parameters

Proceeding from the initial mining-and-geological conditions of the seam $1_2^{в}$, the 2nd western longwall face was the first prepared by mining operations. In the course of its placement, it was assumed that the subsequent prepared extraction sites (2nd bis, 3rd, 4th, 5th, 6th, 7th, 8th and 9th western longwall faces) with the stope works development in the mine field wing, will be located outside the zone of geological faults influence.

The coal reserves mining in close proximity to geological faults was initially carried out by the 1st western longwall face. Then, due to another geological fault, located approximately in the middle of the extraction site, it was divided into two parts, which have formed the 1st and 1st bis western longwall faces, 100 m long each, respectively. The rest data on the extraction sites exploiting conditions are given in Table 1, and the order of mining the longwall faces, including their joint exploitation – in Table 2. The schedule for commissioning the longwall faces and their decommissioning over time is shown in Fig. 1.

According to the plan for stope works development, the main (supporting) longwall faces are the 2nd, 3rd, 4th, 5th, 6th, 7th, 8th and 9th western ones, which were planned to mine sequentially with a load about 1000 tons/day. Such a plan has been actually implemented at these extraction sites (Table 1). The exception is the 9th western longwall face, during mining of which the indicated load was provided only in the first months of its operation. Then, due to the worsening of mining-and-geological conditions (roof rocks fall and blockages of the longwall face work space), coal mining has decreased significantly. The average daily coal output for the entire period of the extraction site exploitation amounted to 320.9 tons.

The rest mining sites (1st, 1st bis and 2nd bis western longwall faces) were planned to mine with an average daily load of about 200 tons/day. Their main goal was to perform a more complete extraction of coal reserves, including near geological faults.

Table 2

Information about the sequence and duration of extraction sites joint exploitation during mining the anthracite seam $1_2^{в}$ at "Gazeta Izvestia" Mine in the mine field heaved area

Longwall face, mine field wing	Exploitation period, month, year	Sequence of longwall faces commissioning	Maximum number of longwall faces in operation in certain periods of time	Number of exploited longwall faces and periods of their joint operation, month				Period of extraction sites exploitation, month
				1	2	3	4	
1st western	I.1981-III.1984	5	4	-	8	15	16	39
1st bis western	XII.1981-IX.1984	6	4	-	1	16	17	34
2nd western	IX.1978-X.1979	1	2	12	2	-	-	14
2nd bis western	III.1980-IV.1981	3	3	-	8	6	-	14
3rd western	VIII.1979-XI.1980	2	3	4	9	2	-	15
4th western	X.1980-XII.1981	4	4	-	8	6	1	15
5th western	XII.1981-III.1983	6	4	-	-	11	5	16
6th western	XII.1982-VI.1984	7	4	-	1	2	16	19
7th western	IV.1983-IX.1984	8	4	-	-	5	12	17
8th western	V.1984-II.1985	9	4	5	-	3	1	9
9th western	V.1985-VI.1986	10	1	13	-	-	-	13
Mine field wing	IX.1978-VI.1986	-	1-4	34	37	66	68	94

Lawa	Years of extraction sites exploitation								
	1978	1979	1980	1981	1982	1983	1984	1985	1986
1st western					39 months				
1st bis western						34 months			
2nd western		14 months							
2nd bis western			14 months						
3rd western			16 months						
4th western				15 months					
5th western					16 months				
6th western						17 months			
7th western						18 months			
8th western							10 months		
9th western								14 months	

Fig. 1. Schedule for extraction sites exploitation at “Gazeta Izvestia” Mine during mining the seam $1_2^{в}$ in the mine field heaved area

The rock displacement activation is related both to the stope works development within the extraction site boundaries, and within the mine field.

The obtained experimental data make it possible to assess the tendency of a change in gas emission outside the extraction site from the undermined coal and rock stratum, when the rock displacement is activated with the stope works development in the mine field wing.

To determine the general trend of gas emission under the influence of the degree of stope works development in the mine field, the changes have been studied in the total amount of gas ΣI, released in the course of mining the separate extraction sites (Fig. 2).

The analysis involved the results of mining all the extraction sites, except for the 1^{st} and 1^{st} bis western longwall faces.

This is due to the fact that they were located in the immediate zone of geological faults influence and according to exploiting conditions, differed significantly from other longwall faces (host rocks disturbance, gas content of coal, volume of coal output and velocity of stope faces advance, etc.) (Table1).

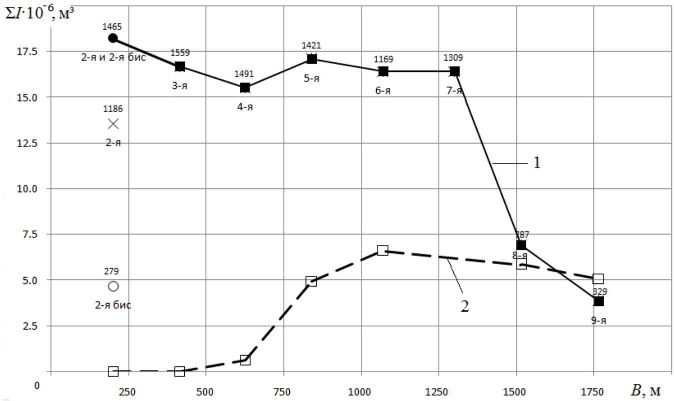

Fig. 2. Dependence of the amount of gas released (ΣI) on the width of the mined-out space (*B*) outside the zone of geological faults influence in the mine field heaved area of "Gazeta Izvestia" Mine

1,2 - curves of changes in the amount of gas released from the undermined coal and rock stratum, within the extraction sites and beyond their boundaries, respectively; $\overset{1559}{\underset{3-я}{+}}$, □ - experimental gas emission data within and outside the extraction sites under the influence of activation of the undermined coal and rock stratum displacement, respectively (a digit above - the panel length (m), below - the number of longwall face); ○,×,● - experimental data on gas emission within the 2nd bis and 2nd western longwall faces with the extraction panels length of 279 and 1186 m., respectively, and their total length of 1465 m.

The extraction panel of the 2nd western longwall face with a length of 1186 m was the first to be partially mined in the mine field wing, and then it was completely finalized by the 2nd bis western longwall face with the extraction panel length of 279 m (Table 1, Fig.2). The total length (1465 m) of the panel mined by these longwall faces was accepted to analysis.

Based on the mined panels length, the extraction sites were divided into two groups that differ from each other. The first group includes longwall faces, the length of extraction panels of which is in the range of 1169-1559 meters, and the second group involves the extraction panels of the 8th and 9th western longwall faces. The length of their extraction panels is 787 and 329 meters, respectively. Besides the extraction panels length, the longwall faces of the first group have a higher load on the stope faces. For the 2nd, 3rd, 4th, 5th, 6th and 7th western longwall faces, the average daily coal output (for the entire exploitation period) was 819.0-1099.1 tons, and the total amount of coal mined for each extraction panel was in the range of 468.0÷535.3 thousand tons. These indicators are significantly different for the longwall faces of the second group. The total coal output from the 8th western longwall face amounted to 265.7 thousand tons with an average daily output - 874.0 tons, and for the 9th western longwall face these indicators were 116.8 thousand tons at 320.9 tons/day, respectively.

To set a general trend of a change in the gas emission amount within the extraction sites and beyond their boundaries, the influence has been studied on these parameters of a change in the dimensions of the mined-out space (B) in the course of mining the extraction panels in the mine field wing (Fig. 2).

The shorter lengths in the longwall faces extraction panels of the second group (the 8th and 9th) significantly affected the decrease in the amount of methane released within the extraction sites. For the first group longwall faces, with panels lengths of 1169-1559 m, 15.5-18.2 million m^3 of methane evolved. When mining the longwall faces of the second group, with panels lengths of 329-787 m, only 3.8-6.9 million m^3 of gas evolved (Table 1, Fig.2).

Experimental data evidence that in addition to coal output, the extraction panels dimensions influence the amount of gas released from the undermined coal and rock stratum within the extraction sites. They are fully determined by the length of the panel and the length of the longwall face. The most integral parameter for characterizing the extraction panel in this case is the area of its mined-out space. This is confirmed by a close correlational directly proportional dependence of the amount of gas released within the extraction sites on the mined-out spaces area (r=0.97) for the first group of longwall faces (Fig.3). The length of the mined-out panels ($L_{ст}$) is in the range of 279-1559 m, and the length of the longwall faces - in the range of 200-250 m. A high correlation coefficient of the dependence between the parameters ΣI and S evidences an approximate constancy of gas emission per unit area of the mined-out space ($\Sigma I_y/S$) under identical mining-and-geological conditions. This indicates that the parameter $\Sigma I_y/S$, with an additional substantiation, can be an initial value for predicting the total amount of gas released from the undermined coal and rock stratum within each extraction panel. For the longwall faces group under study, the average value is $\Sigma I_y/S$=56.2 m^3/m^2 (Fig. 3).

The areas of mined-out spaces at the mined-out extraction sites directly or indirectly depend on the parameters ($\overline{A}, \Sigma A, L_л, \overline{\upsilon}_{оч}$).

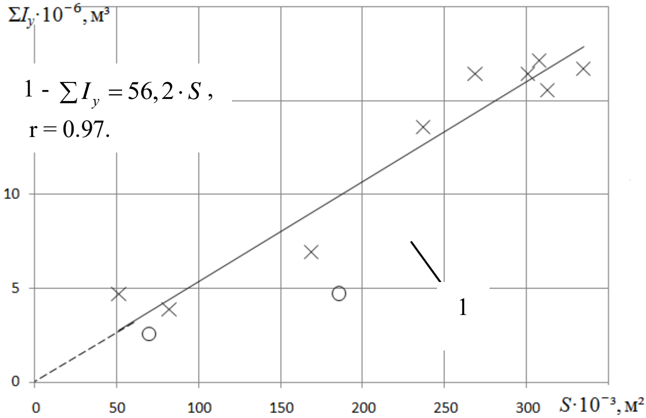

Fig.3. Dependence of the amount of gas (ΣI_y) released within the extraction sites of “Gazeta Izvestia” Mine on the area of their mined-out spaces (S)

1 - averaging line for longwall faces located outside the zone of geological faults; r - correlation coefficient; × - experimental data for longwall faces (2nd, 2nd bis, 3rd, 4th, 5th, 6th, 7th, 8th, 9th) outside the zone of influencing discontinuous geological faults; ○ - experimental data for longwall faces (1st and 1st bis) near geological faults.

The product of the longwall face length $L_л$ by the extraction panel length $L_{ст}$ functionally corresponds to the mined-out space area S. The average velocity of the stope face advance ($\overline{\upsilon}_{оч}$) determines the period of mining the extraction panel of a length ($L_{ст}$), with the longwall face length $L_л$, and the total coal output ΣA depends on the area of mined-out space S. The listed parameters are interdependent, and they determine both the total amount of gas released from the undermined coal and rock stratum within the extraction site ΣI_y, and the average gas emission $\bar{I}$ for the period of mining the extraction panel. Given the foregoing, the link of parameters has been studied (Fig.4). The total coal output from the extraction panel ΣA is almost functionally (r=0.99) related to the area of mined-out space (S). The empirical coefficient 1.622 of equation 1 by its nature characterizes the seam efficiency with a thickness of 0.9 m (Fig.4*a*). Similarly, with a high direct proportional correlational dependence (r=0.98), the average daily coal output $\bar{A}$ is determined by the average daily velocity $\bar{\upsilon}_{оч}$ of the stope face advance (Fig.4*б*).

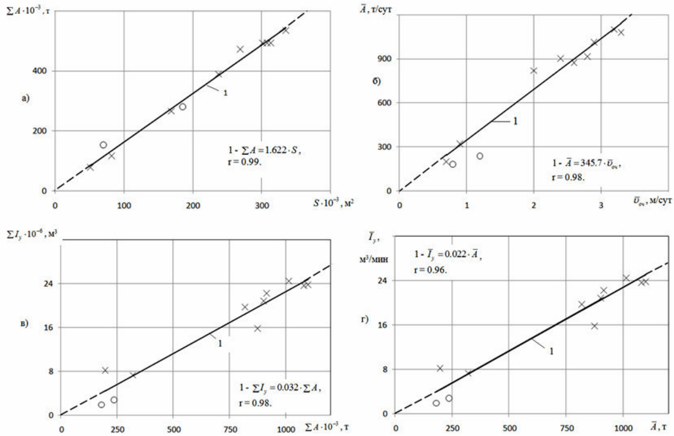

Fig. 4. Dependences between the parameters determining the total amount of gas (ΣI) and its average value ($\bar{I}$) within the extraction sites of "Gazeta Izvestia" Mine

a) dependence of coal ΣA, mined at the extraction sites, on the area of mined-out space S; b) the link between the average daily coal output $\bar{A}$ with the average velocity ($\bar{\upsilon}_{оч}$) of the stope faces advance; c) dependence of the amount of gas released within the extraction sites ΣI_y on the mined coal amount ΣA; d) the influence of the average daily coal output $\bar{A}$ on the level of average gas emission from the undermined coal and rock stratum $\bar{I}_y$ within the extraction sites;

1 - averaging lines; r - correlation coefficients; ×, ○ - experimental data obtained outside and within the zone of geological faults influence.

With high reliability (r=0.98), the total coal output (ΣA) determines the total amount of gas (ΣI_y) released within the extraction site from the undermined coal and rock stratum (Fig.4*в*).

The average daily absolute gas emission $\bar{I}$ directly proportionally depends r=0.96 on the average daily load $\bar{A}$ on the stope faces (Fig. 4*г*), as well as on the average velocity $\bar{\upsilon}_{оч}$ of their stope faces advance r=0.93. This confirms that a change in $\bar{A}$ and $\bar{\upsilon}_{оч}$ in the same proportion influences on the change of $\bar{I}$.

The results of experimental data processing

To obtain more reliable results, when determining the calculated (predicted) values ΣI_y and $\bar{I}_y$, it is necessary to proceed, in our opinion, from the projected extraction panel area (S). In this case, the errors are excluded, caused by the determination of ΣA with the use of the mined seam efficiency and $\bar{A}$, the average velocity ($\bar{\upsilon}_{оч}$) of the stope face advance and the exploited longwall face length ($L_{л}$).

Such a proposal is supported by the following arguments:

- an assumed area S of mining the extraction panel is sufficiently reliably determined by the simple lengths product of the longwall faces and extraction panels;
- it is directly visible the physical significance of the ongoing processes associated with gas emission from the undermined coal and rock stratum, as well as with the processes of its possible displacement activation;
- under otherwise equal mining-and-geological conditions, according to the experience of mining one extraction site, the specific gas emission ($\Sigma I_y/S$) is set per unit area of the mined-out space. This can be used to predict gas emissions when mining the subsequent extraction sites, which are developed with other parameters $L_{ст}$, $L_{л}$, $\bar{\upsilon}_{оч}$, ΣA and $\bar{A}$.

An integral part of the gas emission from the undermined coal and rock stratum is methane release outside the extraction sites (panels) under the influence of the stope works development in the mine field and the rock displacement activation. Depending on the ventilation schemes used, gas emission in this case can occur both within the boundaries of the extraction sites and beyond them into all-mine workings (Table 1, Fig.2).

Unlike gas emission within the extraction sites ($\Sigma I_y, \bar{I}_y$), methane release beyond their boundaries from the mined-out space of the stopped longwall faces ($\Sigma I_в, \bar{I}_в$), at first glance, does not depend on the area of the mined-out space of extraction panels S, total ΣA and average daily coal output $\bar{A}$, as well as the velocity $\bar{\upsilon}_{оч}$ of the stope face advance (Fig. 5). The correlation coefficients r between the specified parameters are in the range of 0,10-0,17, which indirectly indicates the absence of any link between the studied parameters. This fact indicates a different mechanism of the gas emission process from the undermined coal and rock stratum within the exploited site and beyond its boundaries from the mined-out space of stopped longwall faces. This also indicates the influence of other factors that are more significant in this case. Such factors are obviously associated with the stope works development within the entire mine field wing and the manifestation of the rock displacement activation processes.

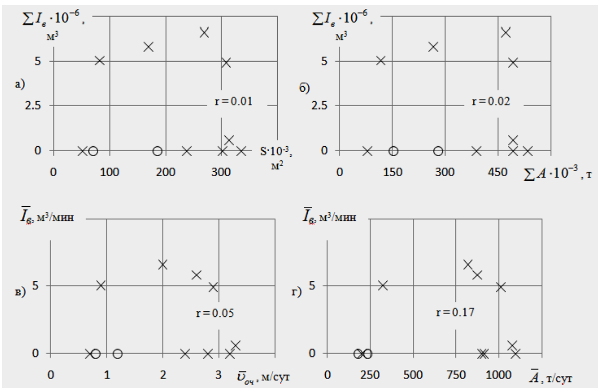

Fig. 5. Influence of some parameters on the total amount of gas released ($\Sigma I_в$) and its average level ($\bar{I}_в$) outside the extraction sites (from the mined-out spaces of the stopped longwall faces) at "Gazeta Izvestia" Mine

а) and б) the influence of the area (S) of the extraction sites mined-out spaces and coal output (ΣA) on the amount of gas released

$\Sigma I_в$, respectively; в) and г) the influence of the average velocity $\bar{\upsilon}_{оч}$ of the stope faces advance and average daily coal output $\bar{A}$ on the average gas emission outside the extraction sites $\bar{I}_в$, respectively; r - correlation coefficient; ×, ○ - experimental data obtained outside and within the zone of geological faults influence, respectively.

The parameters of gas emission ($\Sigma I_y, \bar{I}_y$) within the exploited extraction site depend on the stope works development within its boundaries and factors characterizing the conditions and regime of coal mining (S, ΣA, $\bar{A}$, $\bar{\upsilon}_{оч}$).

An analysis of the gas emission process from the mined-out space of the stopped longwall faces indicates (Table 1, Fig. 2) that it begins to occur with some stope works development in the mine field wing. Significant level and volume of gas emission ($\bar{I}_в$=0.9 m^3/min and $\Sigma I_в$=0.6 million m^3) were revealed during the 4th western longwall face exploitation, when the dimension (width) of the mined-out space of the stopped longwall faces was 415 m (Table 3). When exploiting the subsequent longwall faces (5th, 6th and 7th conjugated, 8th, 9th), the amount of gas released from the mined-out space of the stopped longwall faces $\Sigma I_в$ has increased and was in the range of 4.9-6.6 million m^3 with an average value of 5.6 million m^3. The general nature of a change in $\Sigma I_в$ depending on the degree of stope works development in the mine field depended on the ratio of the parameter B-$L_л$to the depth H of mining operations. A high correlation ratio (r=0.97) of such a dependence (Fig. 6*a*) evidences a non-random link between the studied parameters.

Table 3

Information on the total amount of gas ($\Sigma I_в$) released outside the exploited sites (from the mined-out spaces of completed longwall faces) and its average level ($I_в$)

Longwall face	Longwall face length, $L_л$, m	Total width of mined-out space, B, m	Width of mined-out space of completed longwall faces, B-$L_л$, m	$\frac{B\text{-}L_л}{H}$	Gas emission within the exploited site		Gas emission outside the exploited site		$\frac{\Sigma I_в}{\Sigma I_y}$, fractions	$\frac{\bar{I}_в}{\bar{I}_y}$, fractions
					$\Sigma I_y \cdot 10^{-6}$, m^3	$\bar{I}_y$, m^3/min	$\Sigma I_в \cdot 10^{-6}$, m^3	$\bar{I}_в$, m^3/min		
4th western	210	625	415	1.38	15.5	23.6	0.6	0.9	0.04	0.04
5th western	216	841	625	2.08	17.1	24.4	4.9	7.1	0.29	0.29

6th western	230	1071	841	2.80	16.4	19.7	6.6	8.0	0.40	0.41
7th western	230	1301	1071	3.57	16.4	20.8	-	-	-	-
8th western	215	1516	1301	4.34	6.9	15.8	5.8	13.4	0.84	0.85
9th western	250	1766	1516	5.05	3.8	7.3	5.1	9.7	1.34	1.33

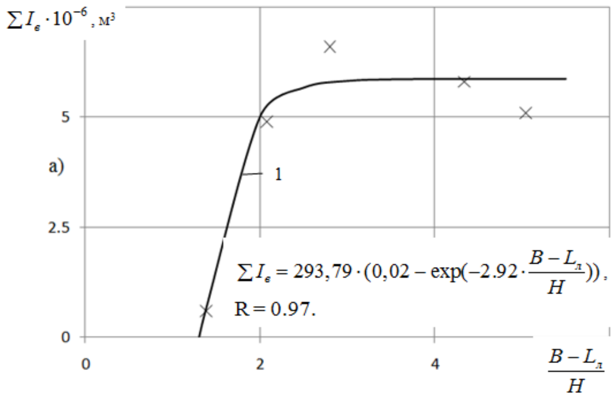

Fig. 6. Dependence of the total amount of gas ($\Sigma I_в$), released outside the extraction sites (a) and its ratio with the amount of gas (ΣI_y), released within the extraction sites (б) on the degree of the stope works development ($(B\text{-}L_л)/H$) in the mine field

1 - averaging curve and line; R and r - correlation ratio and correlation coefficient, respectively; × - experimental data; B and $L_л$ - the total mined-out space dimension and the exploited longwall face length, respectively; H=300 m - average depth of conducted stope works.

The experimental data (Table 3) also made it possible to set a ratio of gas emission from the mined-out space of the stopped longwall faces $\Sigma I_в$ to methane release within the exploited extraction site ΣI_y, depending on the degree of stope works development in the mine field $(B\text{-}L_л)/H)$. This ratio in the studied mining-and-geological conditions varied from zero (when $(B\text{-}L_л)/H)\leq 1.4$) and reached the value of 1.34 (when $(B\text{-}L_л)/H)=5.05$). The obtained dependence (Fig. 6*б*) evidences that gas emission from the mined-out space of the stopped longwall faces begins to occur when the degree of undermining of the coal and rock stratum is more than 1.4, which corresponds to the complete undermining of the earth's surface according to the regulatory document [3].

Conclusions. The studies performed made it possible to draw the conclusions that have important scientific-practical importance for ensuring safe conditions in coal mines relative to the gas factor:

- gas emission from the coal and rock stratum within the exploited extraction site directly proportionally depends on the area of the mined-out seam S, the total coal output from the extraction panel (ΣA), the average daily coal output $\bar{A}$ and the average velocity of the stope face advance $\bar{\upsilon}_{оч}$. The correlation coefficients for these dependences were in the range of 0.95-0.99;

- the most convenient and intuitive parameter for determining the total amount of gas ΣI, released from the undermined stratum within the extraction site and the average level of methane release $\bar{I}$ is the specific gas emission per unit area S of the mined-out space ($\Sigma I/S$). This does not exclude the possibility of using other parameters (ΣA, $\bar{A}$, $\bar{\upsilon}_{оч}$);

- with incomplete undermining of the earth's surface and coal and rock stratum, gas emission from the mined-out space of the stopped longwall faces outside the exploited extraction site is almost equal to zero;

- gas emission from the mined-out space of the stopped longwall faces is possible with the degree of the earth's surface undermining equal to $(B-L_{л})/H)\geq 1.4)$;

- the ratio of gas emission within the extraction site (ΣI_y) to methane release from the mined-out space of the stopped longwall faces ($\Sigma I_в$) directly proportionally depends on the parameter $B-L_{л})/H$. With a sufficient degree of stope works development in the mine field $(B-L_{л})/H>2$ $\Sigma I_в$ can significantly exceed ΣI_y. Such an excess is most probable at low rates of the exploited site development.

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SYSTEMATIC APPROACH TO THE ANALYSIS OF OCCUPATIONAL INJURIES IN POTASH SALT PRODUCING ENTERPRISES

Shwager N.
Kryvyi Rih National University, doctor of technical sciences, Professor, Ukraine

Komisarenko T.
Kryvyi Rih National University, PhD, Associate Professor, Ukraine

Nesterenko O.,
Kryvyi Rih National University, PhD, Associate Professor, Ukraine

Bliznukova O.
Kryvyi Rih National University, PhD, Associate Professor, Ukraine

Abstract

The aim of this work is to analyze the industrial injuries at potassium salt mining enterprises on the basis of a systematic approach and to develop of recommendations to prevent accidents, reduce the probability of industrial injuries and occupational diseases.

Methods. The system approach was taken as a basis for the research. The data on the main causes of accidents are analyzed, and indicate the urgent need to take into account the peculiarities of injury and death of workers, the development and implementation of measures to improve the safety of the industrial environment. The functioning of the occupational safety management system has been assessed. It is carried out on the basis of health and safety criteria - analysis and evaluation of the results of inspections, according to which measures aimed at improving health and safety are developed.

Results. The presented method of analysis of risks and industrial injuries makes it possible to offer a set of solutions in the field of safe development of potash deposits, taking into account that employees of all levels should be actively involved in health and safety management. The standards particularly emphasize the importance of occupational safety training, professional development and competence of personnel (their knowledge and skills). The enterprise should control the level of knowledge of the employee regarding his or her health and safety responsibilities, the results of his or her actions in terms of safety, understanding of his or her responsibility, including actions in emergency situations, and possible consequences of non-compliance with technological instructions.

Statement of the problem and its connection with scientific and practical tasks.

The level of industrial injuries is the main indicator of the state of labor protection in any area of economic activity and in the state as a whole. According to experts, losses in the economy from one fatal or severe accident are tentatively estimated at $ 75,000. Accidents without a severe outcome (with loss of ability to work), absenteeism due to illness, employee turnover result in the loss of about 100 thousand person-days per year [1]. Insurance payments for compulsory insurance against industrial accidents and occupational diseases amount to about $ 25 million annually, and about $ 130 million to compensate for adverse working conditions [2; 3].

According to statistics from the International Labor Organization, one person dies every three minutes as a result of an accident or occupational disease in the world, and every second four workers are injured. This number of injured exceeds the number of injured in traffic accidents, [1]. The total number of people injured by accident in the workplace is 270 million people a year, about 160 million more people suffer from diseases associated with labor activity. When extracting potassium salts according to official data, more than 5 thousand workers are injured annually due to violations of labor protection requirements at the workplace, of which about 250 people die, more than 800 people are seriously injured [2].

Data on the main causes of industrial accidents indicate the urgent need to take into account the characteristics of injuries and deaths of workers when developing and implementing measures to improve the safety of the working environment [4].

Problem statement. The aim of this work is to analyze occupational injuries in enterprises producing potash salts based on a systematic approach. Development on this basis of recommendations for preventing the occurrence of accidents, reducing the likelihood of work-related injuries and occupational diseases.

Analysis of research and publications. Of practical interest is a risk management in accordance with labor protection policies and goals. It is carried out in the context of a general growth of interest in labor protection issues [5].

Many enterprises conduct analyzes or audits in the field of labor protection in order to evaluate the effectiveness of the OSH management system. However, these measures alone are not sufficient to ensure that OSH management system meets the requirements stipulated by legislation and labor protection policy. Analyzes and audits at the enterprise should be carried out within the framework of a structured management system integrated into the management of the organization. OSH standards are designed to provide enterprises with elements of an effective labor protection management system that can be implemented along with other management elements to achieve goals in the field of labor protection [5].

The main one is the OHSAS 18001 standards system, which provides for the implementation of management and administration of occupational health and safety at work. The main attention of the standards of the OHSAS 18001 series is aimed at ensuring the safety of workers [6]. These standards are used by enterprises to minimize the risks and dangers associated with their production activities.

Statement of the main research material. Belarus is one of the largest producers and exporters of potash fertilizers in the world. According to the International Fertilizer Association, it accounts for one fifth of the global potash fertilizer production. Having a sufficient natural raw material base, high production potential of workers and specialists, enterprises cope with these tasks by introducing new equipment and improving technological processes.

The labor protection and industrial safety management system (OSHMS PB) is a fundamental document on labor protection and industrial safety management and is developed in accordance with the requirements of STB 18001, OHSAS 18001 [6,7].

SUOT PB establishes the main tasks of conducting preventive work on labor protection and the implementation of production control at the production facilities of enterprises and is mandatory for all employees.

The main tasks of the OSH management system are:

- the fulfillment by workers of the requirements for labor protection and industrial safety in the decision-making process during the organization and execution of work;
- the constant formation of safe employee behavior in the workplace, the implementation of labor discipline, taking into account the psychology of the individual;
- minimization or elimination of risks for employees;

- Continuous improvement in the field of labor protection and industrial safety;

- increasing the responsibility of each employee for the fulfillment of their duties to create safe working conditions and timely elimination of the reasons that could lead to work injury, accident, incident;

- implementation of production control by managers and specialists.

An assessment of the functioning of the OSH management system is carried out on the basis of labor protection criteria - analysis and evaluation of the results of inspections, according to which measures aimed at improving the state of labor protection are developed.

The control procedures and evaluation of the effectiveness of the OSH management system and its elements are the basis for the development of appropriate measures to improve working conditions.

Control procedures are carried out with the involvement of personnel who have been trained and instructed in the field of labor protection in the following areas:

- monitoring the implementation of planned labor protection measures;

- continuous monitoring of the state of the production environment;

- multistage control of the state of working conditions at the workplace;

- check the readiness of the organization to work in the autumn-winter period;

- internal audit (audit) of the management system;

- external verification (audit) by the certification body.

Evaluation of the effectiveness of the system is carried out according to the following criteria: statistical reporting on the state of working conditions of workers - information on the state of working conditions and compensation for work in harmful and (or) dangerous working conditions and on the status of injuries; the results of the investigation of accidents, occupational diseases and industrial incidents and their impact on safety and health; analysis of occupational injuries; assessment of the effectiveness of the OSH management system and its elements.

For this, taking into account the complexity, danger and technical equipment of units, depending on the number of employees, the possibility of using a set of indicators and their achievement, basic integrated indicators are established annually in the enterprise units. The basic integrated indicator and the calculated integrated indicator for units and management as a whole is determined as the arithmetic average of the integrated indicators for units.

The use of a safety assessment indicator in practice makes it possible to identify and describe all sources of hazards, evaluate risks, provide management measures, substantiate and develop measures to reduce risks to acceptable levels.

When identifying possible dangers (dangerous situations) associated with the fulfillment of labor duties by employees, the OSH determines that the sources of dangers may be:

- equipment used in the work or affecting the employee;
- environment and production environment;
- the person (working or student enterprise, the visitor, the representatives of other organizations).

In the process of identifying potential hazards (dangerous situations), characteristic for the activities of units and employees (enrolled in the enterprise), head of structural unit may determine the hazards (dangerous situation), not previously entered in the register of hazards. In this case, the head of Department updates the register of hazards and risks and bring the updated document to stakeholders.

To date, developed a variety of documents on implementation of hazard analysis and risk assessment in industrial plants [8,9,10].

Are the fundamental documents OHSAS 18001:2007 "management System occupational safety and health"; OHSAS 18002:2008 "management System safety and health. Guidance for use". In these documents the necessity, principles and methods of hazard identification and estimation of occupational risks for workers. Also known standard ISO/IEC 31010:2009 "risk Management. Risk assessment techniques" (ISO/IEC 31010:2009 "Risk management - Risk assessment techniques"). It contains recommendations for selection, depending on initial conditions, methods to assess production risks. The standard lists the 31 method of analysis (for example: brainstorming; Delphi technique; evaluation of Toxicological risk; analysis of the fault tree analysis; causal analysis; the study of hazard and operability (HAZOR); Markov analysis; simulation Monte-Carlo, etc.).

The International Social Security Association (ISSA) has developed the Zero Injury Concept, or Vision Zero. In September 2017, the 21st World Congress on Occupational Safety and Health in Singapore launched a global international campaign to promote and implement this concept.

Zero Injury, or Vision Zero.

This is a qualitatively new approach to the organization of prevention, combining three areas - safety, occupational health and well-being of workers at all levels of production.

The Vision Zero concept developed by ISSA is flexible and can be adapted to any place of work, at any enterprise in any industry in all regions of the world [11].

When conducting hazard identification and risk assessment, the following are considered:

- normal working conditions (planned activities of employees in accordance with the requirements of regulatory documents);
- non-standard (emergency) working conditions (in case of deviations from normal activities, accidents, disasters, incidents, etc.).

There is a large number of documents on identification, analysis and risk assessment at industrial enterprises

The ballast method is used to assess the risks at potash production facilities.

The risk assessment is carried out by the formula below

$$R = S \times Pr,$$

where R is the estimated risk, in points; S - severity of damage from exposure to a specific hazard, in points; Pr - probability of a specific hazard, in points

$$Pr = St \times P \times D \times C,$$

where St - hazard statistics; P - coefficient of exposure of workers to the effects of danger, in points; D - coefficient characterizing the frequency (duration) of exposure to a specific hazard, in points; C - coefficient determined by the probability of failure to take measures to control the impact of a specific hazard (taking into account the human factor), in points.

The quantification of the severity of the harm caused by the specific hazard is determined in Table 1.

Table 1

Severity of harm assessment

The severity of harm from exposure to a specific hazard, St	Consequences of exposure to hazards	The severity of the consequences
1	minimal	Slight impact, painful sensations, fatigue
1.5	moderate	Disability accident, micro-injuries
3.0	significant	Disability accident, no threat to life (non severe injury)
4.0	considerable	Injury related to serious, occupational disease
5.0	catastrophic	Fatal accident, group accident with severe consequences, fatal accident

The statistics of hazard occurrence is given in Table 2.

Table 2

Hazard statistics

Hazard statistics, St	Frequency of danger
1	No injuries
1.5	1 case of injury during 10 years of work
2.0	2 cases of injury during 10 years of work
3.0	3-4 injuries during 10 years of work
4.0	5 or more injuries during 10 years of work

Depending on the level of safety of technological processes, the state of equipment, the degree of safety of workers, the coefficient of exposure of workers to danger is determined, the values of which are given in Table 3.

Table 3

Ratio of exposure of workers to the hazard

Operating Hazard Exposure Coefficient, P	Exposure to danger	Safety characteristics of the used equipment, processes and the degree of employee protection
1	minimum	
2	moderate	

Continuation of table 3

3	significant	Used equipment is practically not worn out, undergoes periodic maintenance (inspection), is equipped with protective devices to prevent the impact of this danger at the workplace. Qualified personnel provided with individual and collective protection means are allowed to work.
4	High	Used equipment is practically not worn out, undergoes periodic maintenance (inspection), is equipped with protective devices. The technological documentation for individual operations is not developed. Qualified personnel, partially provided with individual and collective protection means, are allowed to work.
5	very high	Used equipment of own manufacture or practically worn out, the fact of passing maintenance (examination, testing) is not established, equipped with separate protective fences. Work performance is not regulated by safety measures set out in the technological documentation.

The coefficient D characterizing the frequency (duration) of exposure to a particular hazard is determined by Table 4.

Table 4

Coefficient characterizing the frequency (duration) of exposure to a specific hazard

Value D, score	Description of the duration of exposure to a specific hazard
0.5	Isolated (once a year, once every few years)
1	Minimum (several times a year)
1.2	Random (once a month)
1.5	Temporary (once a week)
2.5	Frequent (every day)
3.0	Continuous exposure

The manifestation of the human factor (coefficient D), is determined by Table 5.

When determining the coefficient, cases of injuries of employees of a given profession, which occurred at a particular workplace, are taken into account.

Table 5

Coefficient of human factor manifestation

Value C, score	Description of the probability of failure to implement measures to manage the impact of the hazard
1	The probability of failure to implement hazard management measures is low (less than 10%)
1.1	The probability of failure to implement the hazard management measures is low. (10 % to 25 %)
1.2	The probability of failure to implement hazard management measures is at an average level. Such violations are not systematic (25% to 50%)
1.3	The probability of failure to comply with the hazard management measures is high. Violations occur quite regularly and/or within a certain time interval (50% to 75%).
1.4	The probability of failure to implement the hazard management measures is very high. Disturbances always occur for a sufficiently long period of time (usually under normal operating conditions) (75 % or more).

Conclusions. The presented method of risk analysis and industrial injuries allows us to propose a set of solutions in the field of safe development of potash salts, taking into account the fact that workers of all levels should be actively involved in the management of labor protection.

The standards especially emphasize the importance of training on labor protection issues, staff development and competence (their knowledge and skills).

The enterprise must control the level of knowledge of the employee regarding his labor protection duties, the results of his actions in terms of safety, understanding his responsibility, including actions in emergency situations, as well as the possible consequences of non-compliance with technological instructions.

Once every five years, certification of jobs should be carried out and a map of working conditions should be drawn up.

Measures to reduce risk may be associated with:

- introduction of additional requirements, revision, amendment, development of local regulatory acts on labor protection and industrial safety (including instructions on labor protection);

- additional training, briefing on safe work methods, periodic and production control;

- repair, modernization, reconstruction or replacement of faulty and obsolete equipment;

- the use of a safer way to perform work (alarm, blocking, etc.);

- preventing access to hazardous and harmful production factors;

- the organization of work in a way that reduces the impact of a harmful or dangerous production factor;

- the issuance of additional personal protective equipment, their replacement, etc.

Consequently, hazard identification, risk assessment and control should be the basis of the entire OSH management system.

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UTILIZATION PROCESSING OF WASTE OF ALUMINA PRODUCTION INTO WATER TREATMENT REAGENTS

Astrelin I.

National Technical University of Ukraine "Igor Sikorsky Kyiv Polytechnic Institute", DSc.(Engineering), Professor, Dean of Chemical Technology Faculty, Ukraine

Kosogina I.

National Technical University of Ukraine "Igor Sikorsky Kyiv Polytechnic Institute", Ph.D., Associate Professor of the Department of Inorganic Substances Water Treatment and General Chemical Technology, Chemical Technology Faculty, Ukraine

Kyrii S.

National Technical University of Ukraine "Igor Sikorsky Kyiv Polytechnic Institute", Ph.D. Department of Inorganic Substances Water Treatment and General Chemical Technology, Chemical Technology Faculty, Ukraine

Abstract

Continuous accumulation of waste from alumina production "red mud" leads to contamination of large land areas, reservoirs, and soils and is a potential threat of environmental catastrophe, that leads to the urgent need to find and implement rational ways of its recycling.

The most studied areas for alumina wastes recycling include the production of building materials and metals extraction. Red mud acts as a binder filler in the field of production of building materials. Products based on such filler have higher heat resistance, strength and density compared to the control samples. The obtained concrete samples are more resistant to negative temperatures due to the high alkalinity of the filler. However, analyzing the results of existing studies about precious metals extraction from the red mud, its usage in the construction industry, as soil improvers etc., we can conclude that today there are no economically and environmentally justified technologies that would allow red mud utilization in industrial-scale. This is prevented by its high alkalinity, small particle size, complex chemical and mineralogical composition.

Red mud is mainly composed of iron and aluminium oxides, which characterizes it as an attractive secondary raw material. Based on a critical analysis of the results of the work on the disposal of red mud by foreign and domestic researchers, it is possible to outline a promising direction of red mud recycling. Namely, it is the creation of water treatment reagents that can be effective for the removal of organic pollutants of natural and human-made origin. Creating water treatment reagents with the full

use of red mud as a raw material can help to solve the problem of accumulation of the latter and provide inexpensive and effective reagents in the relevant field.

Introduction

Aluminium production consists of bauxites extraction with their further purification to the aluminium oxide by the Bayer method and smelting of aluminium oxide to commercial aluminium [1]. Red mud (RM), which is formed as a by-product in the bauxite leaching process by the Bayer method, is one of the major problems of the aluminium industry in terms of resource conservation and environmental protection. According to various sources, it is established that 1 to 2 tonnes of RM is produced per 1 ton of received aluminium product [2–4]. Around 750 million tonnes of aluminium had been produced in the world over the past 20 years, and about 56 million tonnes - in 2017 alone [5]. Therefore, annually in the world dimension, respectively, from 50 to 100 million tons of RM is produced (Fig.1).

There are two alumina enterprises in Ukraine: JSC "Zaporizhzhya Aluminum Mill" and LLC "Mykolaiv Alumina Plant" (MAP). According to the State Statistics Service of Ukraine in 2017, 1.77 million tonnes of alumina waste was generated in Ukraine [6]. The sludge dump of these enterprises, where such waste is collected and accumulated, poses an extreme danger and have not been modernized for many years, which can lead to environmental accidents [7].

An example of such accidents could be the accident in Hungary, that took place on 4 October 2010 at the Ajkai Timfoldgyar Zrt aluminium plant in the Ajka city (160 kilometres from Budapest). The blast at the plant destroyed the dam, holding the waste tank. Thus, about 1.1 million m^3 of red mud leaked and flooded the area in three regions of Hungary. Hungarian government declared a state of emergency and reported nearly 140 casualties [8].

In the Mykolaiv area, there was a leak at MAP (2011). The repository broke, resulting in the ejection of red mud. Because of the frost, it turned into dust, and because of the strong wind (so-called wind deflation), it swept to villages of the region. Besides, sludge dumps are potential sources of surface and groundwater contamination.

All this leads to an urgent need to find ways of alumina waste disposal. After all, the utilization of such waste is not only a means of reducing the anthropogenic load on the environment but is economically attractive because of the red mud chemical composition and its low cost as raw material.

The purpose of the work is to analyze the existing ways of solving the problem of red mud accumulation and determination perspective direction of its recycling. Identification of critical aspects of the complex technologies application for the alumina waste utilization, aimed at both direct consumption red mud, and its transformation into a secondary resource. Recycling can be implemented mainly in three directions: first, the recovery of useful components (metals) from red mud, second, the reuse of red mud as raw materials, especially in the production of cement; third, the use of red mud in environmental processes, in particular as sorbents and coagulants in water treatment. Therefore, when solving practical problems and creating a theoretical substantiation of the treatment conditioning technologies, obtaining and using red mud as an effective sorbent of water treatment one should take into account the difficulties associated with the fineness of the particles of native sludge, with its diverse chemical and mineralogical composition.

Materials

Bauxites. This type of raw material accounts for more than 90% of world alumina production. The main aluminium-containing minerals are gibbsite $Al(OH)_3$, boehmite AlOOH and diaspore AlOOH depending on the content of which, bauxite is divided into gibbsite, boehmite or diaspore. Silicon-containing minerals: kaolinite, quartz, opal, chamosite; iron-containing minerals – hematite, goethite, magnetite, siderite, pyrite. Bauxites contain impurities of calcium, gypsum and others.

Alumina worldwide production is mainly made from high quality (low silica) bauxite of the gibbsite or gibbsite-boehmite type, processed by the Bayer method. These include bauxite with a silicon modulus (mSi) above 8-10 (mSi is the ratio of Al_2O_3 to SiO_2 mass).

Nefelin raw materials. Most significant interest to the aluminium, industry is nepheline concentrate (29.1% Al_2O_3), nepheline urtite (27.3%) and nepheline syenite (18-24%).

Red mud (RM) - granulometric and mineral composition of red mud can vary greatly depending on the quality of bauxite and the methods of their treatment.

Depending on the quality of bauxite and the characteristics of its processing, red mud contains (wt.%): 40-55 Fe_2O_3, 14-18 Al_2O_3, 5-10 CaO, 5-10 SiO_2, 4-6 TiO_2, 2-4 Na_2O [9, 10]. The impurity element content is as follows (g/t): 5 Cu, 10 Be, 50 B, 4 S, 0,2 Co, 30 Ga, 30 Sc, 20 La, 30 Ce, 20 Mo, 80 Y, 20 Ni [11-17].

High humidity (up to 80%) is a significant drawback of RM, which complicates its use. Existing sludge dewatering technologies are energy-intensive and inefficient. Also, when developing standards and technical specifications for sludge preparation, transportation and sludge usage, it should be borne in mind that with a residual humidity of 8-12% dry sludge is subject to wind deflation.

Analysis of the main methods of processing ore raw materials

Depending on the composition and physicochemical properties of the processed ore, alumina is obtained in several ways. There are three groups of processes: alkaline, acid and acid-alkaline. Nowadays, almost all alumina is obtained by alkaline methods, which in turn are subdivided into hydrochemical (Bayer method), thermal (sintering) and combined [18,19].

Alkaline is the most widespread (Bayer) method and is used only for processing high-quality bauxite with a small amount (up to 5 - 6%) of silica and impurities.

Sintering methods for the processing of low-quality aluminium raw materials (low-quality bauxites, nepheline, alunite, raw clay material, kaolinite, etc.) have become widespread. The basis of sintering methods is as follows: sodium aluminates formation during heating with baking soda and the binding of silicon dioxide in its interaction with limestone in two-calcium silicate.

The Bayer method

The essence of the Bayer method is that aluminium solutions (NaOH solvent) decompose rapidly with the formation of $Al(OH)_3$ with the introduction of aluminium hydroxide (seed). The remaining decomposition solution can again dissolve the alumina contained in the bauxite after evaporation under vigorous stirring at 169-170 °C.

Sodium aluminate and sodium silicate form insoluble sodium aluminosilicate in solution. Titanium and iron oxides, giving the residue a red colour, also pass in the insoluble residue that called red mud. After dissolution, the resulting sodium aluminate is diluted with an aqueous alkali solution while reducing the temperature by 100°C.

By Bayer method, the technological cycle of the alkaline reagent is closed. Spent on leaching reagent is released at decomposition and returns on the beginning of the process for processing new portions of ore. The original bauxite is crushed in a medium of concentrated circulating alkali solution. But simultaneously with natural aluminium hydroxides, free silica and various aluminosilicates interact with alkali, forming of sodium hydroalumosilicate insoluble in alkali. It leads to the cost of expensive alkali and the reduction of the conversion of aluminium into a solution. Therefore, it is impractical to process bauxites by Bayer method with high SiO_2 content.

Aluminate solution and sludge separation are usually doing by concentration with the subsequent filtration of the solution from a thin suspension. The sludge released from the thickeners is subjected to repeated washing by the principle of counterflow. It allows for red mud being more fully washed away from the aluminate solution residues and obtain flushing water with higher concentration. The washed red mud is pumped into sludge dump.

The pure solution is subject to decomposition. For this, the solution is diluted and cooled: seed (small crystals of aluminium hydroxide) is introduced, and the slurry is stirred for 50-90 hours to obtain large enough $Al(OH)_3$ crystals. Small crystals are centres of crystallization. The crystal growth interval is 52-56 °C at the beginning and 44-46 °C at the end.

For separating the hydroxide crystals from the mother liquor and classifying them by size, hydro separator, hydro cyclones and thickeners are used. The concentrated slurry is filtered and washed on drum or disk vacuum filters, with large crystals settling.

Two fractions (aluminium hydroxide and the mother liquor) receive after separation of the slurry. The large fraction (40-100 microns) is productive hydroxide and is directed to calcination. Small particles (up to 40 microns) go as a seed when decomposing aluminate solutions. The yield of aluminium hydroxide is about 65-70%.

The mother liquor is combined with the wash water and sent for evaporation to a special apparatus. Excessive moisture is released there, and the concentration of caustic soda is thus increased to an optimum value for cleaning the solution of soda and some other impurities. Sodium carbonate in the Bayer process is formed at the leaching stage as a result of the interaction of alkali with the carbonates of the original charge. For returning the soda to the process, it is turned into a caustic meadow by the method of limestone caustification. In this case, white sludge is obtained as a waste, which is returned for leaching or subjected to special processing.

Alumina production by sintering method

The essence of the method is the formation of sodium aluminate at high temperature as a result of the interaction mixture of aluminium ore, soda and limestone. The resulting sinter is leached with water. The solution of sodium aluminate after leaching is decomposed with carbon dioxide with aluminum hydroxide precipitating, which is calcined to obtain anhydrous alumina.

This method can process almost all types of aluminium raw materials. Nowadays it is used for processing of high silica bauxites, nepheline ores and concentrates.

The raw materials (bauxite and limestone) after grinding are fed to the mills, where they are crushed in a circulating soda solution. Some fresh soda is added there to compensate for its loss in the processing process and reversible sludge.

After leaving the refrigerator, the sinter is directed to a leach, grounding to a size of 6-8 mm. Leaching is carried out with water and reversible weak solutions of soda.

The advantages of the method are to obtaining strong solutions containing Al_2O_3 up to 300 kg/m^3. But compared to the Bayer method, sintering methods require considerable energy costs. On the other hand, they are more versatile because they have lower raw material quality requirements. Such methods make it possible to process virtually any raw material (including waste) containing aluminium to alumina with a high extraction of Al_2O_3. The processing of raw materials by the method of sintering on alumina is quite advisable. In addition to alumina, in this way, soda, potash, and cement are formed as by-products. As a result, this reduces the environmental impact and burden.

Features of bauxite processing at the Mykolaiv alumina plant (MAP)

Bauxite, which is processed by MAP, consists mainly of aluminium hydroxides (boehmite, amorphous aluminium hydroxides, gibbsite) and iron hydroxides (goethite, aluminometite, hematite, dispersed hematite), as well as of minor minerals (rutile, anatase, quartz, kaolinite, zircon, etc.). The technological scheme of processing bauxite involves grinding it in ball mills in the presence of concentrated alkaline solution with the addition of calcium hydroxide in the form of lime milk. The resulting slurry feed is pumped for further processing. The leaching of aluminium from bauxite is carried out in autoclaves with mechanical stirring. The slurry feed after separation is directed to the thickening and washing apparatus, where the aluminate solution is separated from the sludge. The last one, after six times washing, enters the sludge storage [11].

Global practice processing of waste from alumina production

Recovery of precious metals from waste from alumina production

Scientists from different countries have conducted experimental studies on the iron production from red mud [20-23]. In particular, in Central-South University (Shanghai, China) received samples of steel directly from an iron removed from RM [24]. Scientists from the Metallurgical Research Institute (China) have identified the main patterns of magnetic separation of RM for increasing the iron extraction degree from RM to 86% [25].

One way to use red mud is to restore rare earth elements. D.I. Smirnov et al. [26] developed a new method for the recovery and extract of scandium, uranium and thorium from a clay slurry of red mud by adsorption of rare earth metals by ion exchange resins with resin up to 50% by scandium. A. Xue et al. [27] proposed a recovery method of scandium from red mud more than 80% by acid dissolution. J.J. Zhang et al. obtained a solution with metal ions Ti, Sc, Fe and Al by double leaching from RM by low-concentrated hydrochloric acid [28]. A Wang-led research team also investigated Sc extraction from RM and obtained the final product with a 95% purity using hydrochloric acid as a leaching agent [29].

Usage of waste from aluminium production in the construction industry

In the construction industry, red mud can be used in the manufacture of cement, building ceramics (bricks, foam blocks, ceramic tiles, etc.), wall materials, for road construction, and as a binder in cement-clay compositions. Red mud usage in the production of cement (up to

30% of the charge) not only reduces energy consumption but also stabilizes the cement, improves its initial strength and resistance to sulfate attack [30 - 35].

Currently, the MAP annually ships 50-60 thousand tonnes of red mud to cement plants. The potential volume of deliveries is 400-450 thousand tons per year. However, it should be taken into account the narrowing of the scale of RM utilization and its percentage in the cement mixture caused by the limitation of the total alkali content, water-soluble compounds and sludge moisture. Besides, the Fe_2O_3 content in RM is preferably greater than 50%. However, specific strict standards for limiting the chemical composition of red mud for the cement industry are not yet scientifically substantiated or are absent.

Bricks production from the RM has recently have been implemented in Germany and Hungary. The firing temperature is 950 - 1250 °C. In the charge mixture, except for RM (51-90%), quartz sand, volcanic rocks, silicate sludge, as well as plastic clays (7.5-15%) use. As an alternative to traditional raw materials used in the bricks production, the RM usage can not only reduce the cost of raw materials but also has excellent environmental value. World practice shows the possibility of producing fire-resistant brick, decorative brick with black granules and ceramic tile based on RM [36 – 39]. The Shandong Aluminum Company Institute and the Great Wall Aluminum Company Institute have proposed technology of fire-resistant brick production from RM and fly ash. Since from 70 to 80% of the active components (SiO_2 and CaO) are recovered in RM and fly ash, this waste is a promising raw material for the fire-resistant bricks production for reasons of cost and productivity [40].

Based on RM, quartz sand, fluorite, manganese and chromium-containing wastes, decorative and other black glass materials have been successfully produced, with acceptable mechanical strength, chemical resistance and optical properties [41-43].

Foam-concrete is a proven innovative trend in the development of building materials production, due to such advantages as heat insulation, fire resistance, seismic resistance. It is usually made using lime and siliceous materials and some additives. Foam concrete based on cement (15%), lime (12 - 15%), RM (35 - 40%) and quartz sand (33 - 35%) were proposed to produce in China. However, it has been found to meet the lowest MU7.5 level of Chinese standards for the strength of concrete blocks [43]. But the technology of its production does not differ from that of other foam concrete production, so this method with the RM usage reduces the cost of foam concrete.

The prospect of using prepared RM as a part of asphalt in road construction has been determined [44]. Tests on the road stability and road strength 4 km long and 15 m wide covered with asphalt using RM as a basis, showed that this section of the road meets the requirements of the road and the standards of China [45].

The introduction of bauxite sludge as a raw material allows not only to improve the quality of the obtained products but also to increase the production profitability. High iron content in RM (up to 55% wt.) makes it promising to use it in ferrous metallurgy. Red mud is used here as an iron-containing additive in the charge for the agglomerates and pellets manufacture from iron ore concentrate for the blast furnace process. The red mud usage in, for example, blast furnace charge is prevented by the presence of up to 6% of alkali metals in the form of sodium, potassium and zirconium oxides. Studies on the red mud addition in the sintering board show that the introduction of them up to 2-5% is possible. However, it requires the development of blast furnace technology, and most importantly, the problem of RM complete disposal is not solved [46-48].

Wastes from alumina production as soil ameliorant

According to [49-50], red mud can be used as a soil ameliorant and fertilizer containing a wide range of trace elements. Thus, scientists [51] claim that the introduction of certain doses in saline soils has a positive effect on the growth and productivity of some crops (buckwheat, corn, etc.). It is claimed that RM is a more effective ameliorant in saline soils of the southern regions of Ukraine compared to phosphogypsum. Besides, red mud helps to fix calcium in the metabolic-soil complex, which prevents its displacement during salinization of soils [52]. However, despite the positive effect of red mud on soil quality and yield, according to the authors, its use is not always appropriate, especially on human-made contaminated soils. Since red mud contains an increased number of relatively mobile forms of heavy metals with migratory capacity, which can negatively affect the soil microflora, contribute to the pollution of agricultural products.

In general, the red mud assessment by agrochemical science is not unambiguous and sometimes even contradictory. Thus, it is emphasized that the RM has a positive effect on the soil of the environment, contaminated with heavy metals [53]. This is explained by the fact that RM can absorb heavy metals ions and metals with variable valence, such as Cu^{2+}, Ni^{2+}, Zn^{2+}, Pb^{2+}, Cd^{2+}, Cr^{6+}, Mn^{4+}, Co^{3+} i Hg^{2+}. Another mechanism of heavy metals binding is based on the reaction of the red mud carbonates interaction with heavy metal ions with their deposition. In turn, the activity and reactivity of heavy metal ions in the soil decreases, the microbial activity of soils and plant development increases. W.G. Gao et al. [54] conducted some studies showing that RM can significantly reduce the content of Cd and Zn by linking mobile forms of these contaminants in soils.

R. Ciccu and co-workers [53] have also used RM to improve the quality of soils contaminated with heavy metals and declare that RM can reduce the content of heavy metals in severely contaminated soils. ther researchers have found [55] that the addition of up to 2% of RM per 1 kg of subsurface soil layer inhibits the uptake of Cu^{2+}, Ni^{2+}, Zn^{2+}, Cd^{2+} by crops.

Reagents from waste using in water treatment

Since red mud of MAP contains a large number of iron, aluminium, titanium, and other compounds, it is advisable to consider and investigate the possibility of its usage in the native, packed state or composites as raw materials for coagulation and sorption reagents for water treatment [56-61]. It would contribute to the greater economization of alumina production, reduce the human-made load on the environment, provide water treatment technology with efficient and inexpensive reagents.

To date, research on the possibility of red mud using for the synthesis of coagulants and sorbents in wastewater treatment is becoming increasingly important. However, the development of red mud usage ways as a sorbent to remove contaminants from wastewater is accompanied by many problems. The main of which are high alkalinity (pH 10) and small particle size of red RM, which significantly complicates the separation of sorbents after using from treated water. Besides, red mud has a complex chemical and mineralogical composition, which significantly increases the number of factors that affect both the synthesis of reagents from RM and wastewater treatment processes using coagulants and sorbents based on it.

The usage of reagents from secondary raw materials in coagulation processes

Coagulation wastewater treatment processes are promising and widely used today. Coagulation wastewater treatment processes are promising and widely used today. Metal salts (most often iron and aluminium (iron sulfate, aluminium sulfate, iron chloride, polyaluminumchloride, oxyhydroxides of iron and aluminium)) are most often used as coagulants. Such coagulants, along with technological advantages, have certain disadvantages (high cost, corrosive effects on equipment, the dependence of the coagulation process on the pH medium and temperature).

To improve the efficiency of the method of destabilization of stable aqueous suspensions of wastewater, expanding the temperature range of hydrolysis processes are increasingly using complex coagulants, which contain both iron and aluminium [62, 63].

For reducing coagulants cost, it is advisable to use secondary raw materials for their synthesis.

In [64], the RM/$MgCl_2$ system was used to remove dyes from model solutions. The possibility of using a composite coagulant based on RM was investigated in paper [65]. Thus, scientists from China and Singapore have created a coagulant consisting of RM and polyaluminum chloride in different mass ratios, which is effective for phosphates removing.

The usage of reagents from secondary raw materials in sorption processes

The red mud using as a sorbent for the removal of heavy metal compounds, inorganic anions (nitrates, phosphates, fluorides), organic dyes, phenol and phenol derivatives, as well as organic compounds is quite promising [66-68].

Arsenic compounds found in natural waters pose a severe threat to human life and health. Low-dose arsenic poisoning can lead to cardiovascular disorders, skin damage, hearing loss and cancer. In paper [69], the possibility of using RM-based sorbents to remove arsenic As (III) and As (V) compounds from aqueous solutions was studied. In paper [70,71], to increase the red mud adsorption capacity, its heat and acid treatment were performed with a positive result relative to the adsorption capacity of activated red mud. The maximum removal degree of As (V) compounds reached 96.52%, and As (III) compounds - 87.54%.

To date, a severe environmental problem, which still has no effective solution, is the wastewater pollution by hexavalent chromium compounds. In case of insufficient treatment, chromium-containing wastewater is discharged into natural reservoirs and soils, where together with vegetables and fruits the absorbed chromium compounds enter the human body, which causes malignant neoplasms, blocks enzyme systems, disrupts biological oxidation processes. In article [72], the possibility of using red mud-based sorbents to remove chromium (VI) compounds from aqueous solutions was studied.

Copper is a necessary element for the normal functioning of the human body. It is believed that the optimal copper dose in the body is 2 - 3 mg/day. Copper deficiency in the body can develop with insufficient intake of this element (1 mg/day or less), and the toxicity threshold for humans is 200 mg/day. At excess receipt of copper in a human body, there are functional frustrations of a nervous system (deterioration of memory, depression, insomnia). The possibility of RM-based sorbents using to remove copper compounds from aqueous solutions was studied in [73]. The mechanism of the process was presented as a surface reaction of complexation under the influence of electrostatic forces of interaction between copper ions and the adsorbent surface.

When red mud was treated by hydrogen peroxide at room temperature for 24 h, followed by washing and drying at 100 °C, in [74] a sample of sorbent with better adsorption properties in comparison with native red mud was obtained. A high removal degree (almost 100%) of Cd and Zn compounds was achieved in the low concentrations range of the latter. In comparison, at their high concentrations, the adsorption efficiency decreased to 60-65% at pH values of 4.0 and 5.0, respectively.

In [75], the removal efficiency of Rhodamine B and Methylene Blue with the help of sorbents based on red mud in 92.5% and Methylene Blue - 75.0%, respectively, were achieved. In [76, 77], hydrochloric acid-activated RM was used to remove Congo red from aqueous solutions. The most effective adsorption process proceeded at pH 7.0 with reaching equilibrium in 90 minutes.

RM and ash after pre-treatment and acid activation were used [78] as an adsorbent to remove Methylene Blue from aqueous solutions. Activation by nitric acid helped to increase the adsorption capacity of ash and reduce the red mud adsorption capacity. This was explained by the fact decomposition of some organic and hydroxide groups, which, according to the authors, were effective adsorption centres, was not excluded during heat treatment. In our opinion, acid treatment leads to the selective dissolution of some ash components and the creation or increasing of pores and specific surface area of ash, which has the effect of increasing its adsorption capacity. According to the author, during red mud activation by nitric acid, the sorbent surface is re-charged. Due to this, there is a change in its surface properties, which increases the dye adsorption degree.

Several studies [79-81] have determined the ability of effectively RM using to remove phenol, 2-chlorophenol, 4-chlorophenol and 2,4 dichlorophenol from wastewater. It is indicated that the neutralized red mud usage to remove phenol from aqueous solutions with a maximum degree in a wide pH range and contact time of 10 hours must first be "neutralized" by distilled water. In paper [80], the removal of phenol from an aqueous solution using activated red mud by hydrochloric acid was considered. It was found that the maximum removal degree was obtained at pH below 8 and contact time 10 hours. It is established that chemisorption is the determining process in phenol removal.

In article [82] it was found the efficiency of phosphorus ions removal from contaminated water with peat is only in the range from 17 to 21%, and RM in a mixture with peat - 95%. Phosphate uptake by thermal and acid-activated red mud was investigated in. It was found that their removal by activated hydrochloric acid red mud is quite effective. It was found that the removal of phosphates is most effective at pH 7. In paper [83], the red mud was activated with concentrated hydrochloric acid with the suspension washing by distilled water and drying the residue, which was then used to remove phosphates. The study of the dose effect of treated red mud and the pH medium established phosphates removal degree 85%.

Fluorides removal from aqueous solutions using native and by hydrochloric acid-activated red mud was studied in [84]. The maximum degree of their removal (82%) was reached at pH 5.5. In paper [85], the nitrates removal from aqueous solutions by native and hydrochloric acid-activated red mud was investigated. The increase in the red mud sorption capacity after acid treatment is explained by the leaching of sodalite compounds, which are known to prevent adsorption by blocking free adsorption centres. The mechanism of nitrate removal, according to the authors, is explained by the chemical nature of red mud and the interaction between metal oxides and nitrate ions on the surface.

Conclusions

Since red mud contains a large number of iron, aluminium, titanium, and other compounds, it is advisable to consider and investigate the possibility of its usage in the native, packed state or composites as raw materials for coagulation and sorption reagents for water treatment. It would contribute to the greater economization of alumina production, reduce the human-made load on the environment, provide water treatment technology with efficient and inexpensive reagents.

According to the results of experimental data activated red mud can be used as a sorbent for wastewater treatment from heavy metal ions, inorganic anions (nitrates, phosphates, fluorides), organic dyes, phenol and phenol derivatives, as well as from organic compounds. The efficiency of pollutant removal depends not only on the method of red mud activation but also on several additional factors, such as adsorption duration, sorbent dose, pH and contaminants concentration in wastewater.

Despite the noted efficiency of sorbents based on red mud, the technology of wastewater treatment from contaminants with such sorbents is under development today. There are several reasons for this. This fact significantly complicates the possibility of theoretical substantiation of the processes that occur during the sorbents synthesis and the removal of the contaminants from wastewater.

Besides, some of the red mud components can pass into the wastewater directly during the treatment process, thereby introducing additional contamination into the water.

Red mud from different alumina enterprises can differ significantly in their chemical and phase-mineralogical composition. This fact is due to different plants use bauxite (nepheline, syenite) ore of different quality and several technologies for producing alumina. Due to this fact, the process of obtaining a sorbent based on red mud with a set of specific characteristics is much more complicated. Also, the red mud phase-mineralogical composition depends on the conditions and duration of its storage in the sludge dump. Therefore this causes heterogeneity in the composition of the red mud, even at the same enterprise.

RM has high alkalinity, which requires additional costs to neutralize it. Also, RM particles fineness complicates the separation process of sorbents based on red mud from wastewater solutions after their treatment.

Therefore, when solving practical problems and creating a theoretical substantiation of the treatment conditioning technologies, obtaining and using red mud as an effective sorbent for water treatment, one should take into account the difficulties, associated with the fineness of the particles of native sludge, its diverse chemical and mineralogical composition and high alkalinity.

One of the possible directions of solving the RM involving in recycling problem may be the extraction of active components (metal oxides) of the future sorbent from the red mud and their application on a porous granular carrier. It can help to easily separate the resulting sorbent from the treated wastewater, prevent additional contamination (because the sorbent will contain only the target components, mainly applied to the porous carrier) and facilitate the process of adapting the technology of sorbent obtaining from red mud of different enterprises.

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ECOLOGICAL ASSESSMENT OF THE STATE OF ROCKS IN THE OF RECLAMATION PROCESS IN THE NIKOPOL MANGANESE ORE BASIN

Kharytonov M.M.
Dnipro State Agrarian and Economy University (DSAEU), Professor, Doctor of Agricultural Sciences, Professor, Department of Soil Sciences and Farming, Ukraine

Gonchar N.V.
Dnipro State Agrarian and Economy University (DSAEU), Associate Professor, Candidate of Agricultural Sciences, Professor, Department of Soil Sciences and Farming, Ukraine

Gavryushenko O.O.
Dnipro State Agrarian and Economy University (DSAEU), Associate Professor, Candidate of Agricultural Sciences, Professor, Department of Soil Sciences and Farming, Ukraine

Mytsyk O.O.
Dnipro State Agrarian and Economy University (DSAEU), Associate Professor, Candidate of Agricultural Sciences, Professor, Department of Soil Sciences and Farming, Ukraine

Abstract

Improving the technology of reclamation of disturbed lands, leading to the formation of technozems is justified. Parameters of changes in edaphic characteristics of technozem models of different quality in lithogenic composition depending on time as a factor of soil formation in the conditions of the Pokrov experimental station of land reclamation were determined. The long-term (more than 50 years) impact of intensive phytomelioration contributed to an increase in nutrients in technosols, optimization of physical and biological properties was determined.The loose, crumbly rocks of tertiary and quaternary deposits deposited at the dumps form edaphotopes that have no analogues in nature. Disturbed rocks do not retain their former physical, chemical and biochemical properties and are characterized by a large heterogeneity, complex soil and environmental conditions: a low content of nutrients and lack of moisture for vegetation and microorganisms - producers of soil enzymes. Features of growth of enzymatic activity in the formation of "young" soils were revealed. Gradation of the degrees of biogenicity of edaphotopes by enzyme activity has been improved. The content of trace elements in plants grown on rocks was studied.

Keywords: rocks, biological reclamation, technozems, trace elements, plants.

Introduction

Mining activities can cause drastic disturbances in soil properties, which adversely affect the nutrient cycling and soil environment. The suitability of rocks taken up to the day surface as a result of open mining for biological reclamation is determined by their physical, chemical and biological properties [1]. Destruction and grinding of the soil structure at the initial technological stage is due to the use of a rotary complex [2]. This process leads to an increase in the content of small particles of different quality rocks and black soil. As a result, the water characteristics of the substrates also change. The study of physical properties of reclaimed land depends on the specific indexes of substrates, the method of their formation and biological development. The technology of creating model two and multilayers lyzymeter and field experiments determines the differentiation of the density, porosity, aeration borehole, humidity state along the artificial reclamation profile. In the process of open-pit mining, the hydrological regime of not only waste dumps, but also the environment is violated [3]. The availability of moisture to plants is largely due to its movement in specific ground conditions. Water characteristics depend on the soil texture. Water in the soil is in continuous motion: it accumulates and is consumed, turns from one form to another, and is used by plants. As a result of the functioning of the root systems of plants and various physiological groups of microorganisms, enzymes accumulate, which are accumulated by the organo-mineral part of edaphotops. These functional manifestations are the ecological basis that contributes to the restoration of ecosystems and are an integral part of the ecological rehabilitation of used land [4]. The correlation analysis shows that soil organic carbon contents in aggregate fraction of 0.25-0.5 mm were correlated with aggregate distribution and enzyme activities [5]. The non - top - soiled areas, even after 6, 7 and 8 years, appeared to have lower enzyme activities than the younger top soiled areas or the undisturbed soil. The effects of top-soiling and reclamation age on dehydrogenase, nitrogenase, phosphatase, arylsulphatase, amylase, cellulase, invertase and urease activities were evaluated on three reclaimed non-top-soiled and five reclaimed topsoiled areas and compared with an undisturbed reference soil [6]. Optimization of the properties of model structures of technosols in the process of long-term biological reclamation and use is slow. After all, enzymes significantly affect the biochemical processes, and subsequently the level of culture of the microorganisms. The main objective is to study water-physical, nutrition regimes and oxidizing-reducing conditions the phytomeliorated rocks of Nikopol manganese ore mining basin.

Materials and Methods.

The experiments were conducted at the Pokrov Reclamation Research Station of DSAEU, located at 47º39' N, 34º08' E, with an elevation of 60 m. The mining rocks are taken up to the surface during process of manganese ore mining [7]. The stratigraphy of the Nikopol manganese ore basin is shown in table 1.

Table 1

Rock deposits stratigraphy

Age	Depth, m	Name of substrate
Q	0-7	Soils, loess-like loam
N_1SQ	7-12	Red-brown loam and clay
N_1Srm^2+1	12-47	Grey-green clay
N_1Srm_1	47-63	Sand-clay deposits
Pg_1ch_1	63-71	Green montmorillonite clay
Pg_1ch_1	>71	Manganese ore

Q – quaternary; N_2 – Pliocene, upper (late) Neogene; N_1 – Miocene, lower (early) Neogene; Srm1 – lower Sarmat; Srm^2 – middle Sarmat; Srm^3 – upper Sarmat; Pg3 – Oligocene, upper Palaeogene.

Accepted conventional meanings for the substrata as following: SBS - southern black soil; LLL - loess-like loam; RBL - red-brown loam; RBC - red-brown clay; GGC - green-grey clay; GMC - green montmorillonite clay; AAS - ancient-alluvial sand; DGSC - dark-grey schist clay. The soil mass was taken off, piled up and heaped onto the land after the rock has been replaced (Fig.1). Substrates formed in this way can be attributed to the category of technosol [8]. Technosol are soils dominated or strongly influenced by human-made materials and correspond to soils whose properties and pedogenesis are dominated by technical origin. Their parental material is made of all kind of materials made or exposed by human activity that otherwise would not occur at the Earth's surface. Rock and soil substrata were tested in different artificial profiles to estimate their physical - chemical properties in lyzimeter experiments (Fig 2).

Fig. 1. Quarry board (left) and field experiments (right)

Fig. 2. Model experiments in lyzimeters at Pokrov land reclamation station

Several models of rock substrates were tested: 1) loess-like loam (LLL), taken from the board of the quarry (0-150 cm); 2) red-brown clay (RBC) taken from the board of the quarry (0-150 cm); 3) green-grey clay (GGC) taken from the board of the quarry (0-150 cm); 4) black soil (0-50 cm) + loess-like loam (50-150 cm); 5)) black soil (0-50 cm) + Sand (50-70 cm) + LLL(120-150 cm) 6) black soil (0-50 cm) + loess-like loam (50-100 cm) + sand (100-120 cm) + green-grey clay(120-150 cm)

The mechanical, physical, chemical and biological properties of some rock substrata were studied with use of standard methods [9]. The main minerals of rocks silty fraction consist of feldspar, calcite, illite, montmorillonite, chlorite and kaolinite. Previous studies have revealed differences in the content of montmorillonite and illite [10].

Their bigger content causes a higher plasticity of grey-green and red-brown clays.

In order to determine mineral elements contents samples weighing 2g were combusted in a muffle furnace at 450°C by means of drying method and then dissolved in 5 ml 6N spectral purity hydrochloric acid. Determination of trace elements contents in obtained mineralizates were made by means of flame using a spectrophotometer Saturn 3. The obtained data on chemical properties of the plant studied represent the arithmetic means of three replicates of each sampling, their ranges and standard deviations values.

Water-physical properties and nutrition regimes of reclaimed minelands

The water regime of soils of the dumps is changed under the action of gravity, meteorological conditions, hydrogeological situation and the type of vegetation cover [11]. Heterogeneity of factors determines the spatial differentiation of the hydrological regime of the soil cover. The indicators of maximum hygroscopicity in the upper layers of technozems are higher than in the humus horizon of the black soil after 15 years of formation of technozems (Table 1). Maximum hygroscopicity is constantly increasing due to the long-term reclamation effect of plants on the soil. The texture and compaction of the soil affect the formation of an artificial reclamation profile. The models with red-brown and gray-green clays were distinguished by the highest indicators of the productive moisture reserve. The moisture content and water regime in technosol models changes in comparison with the initial data and depends on meteorological conditions, soil texture and organic matter content. The nutritional regimes of artificial reclamation profiles were determined (Table 2). Resources of humus and macroelements status determine the structures of technozems. The use of potentially fertile rocks and their mixtures as the main substrates is possible for artificial reclamation profiles creating. The estimation of easily accessible (AAB pH 4.8) and hard-to-reach (1N HCl) reserves of phytomeliorated substrates in relation to heavy metals is given in table 3.

Table 1

Water-physical properties of edaphic modul structures of the Nikopol manganese deposit (based on 0-100 cm layer)

Substrata	Maximum water absorption, %	Humidity persistent fading, %	The smallest moisture capacity, %	Range active moisture,%	Moisture reserves, mm in layer 0 - 100 cm
BS	6.3 (5.5-7.4)	8.4 (7.2-9.3)	28.6 (27,4-30,6)	20.2 (18.6-21.4)	337 (295 - 348)
LLL	6.8 (5.8-7.3)	9.1 (8.6-10.3)	25.5 (24.8-30.8)	16.4 (15.3-17.3)	352 (344-486)
RBC	11.4 (9.5-12.6)	15.3 (14.1-16.7)	35.8 (34.6-37.2)	20.5 (18.9-22.1)	457 (411-524)
GGC	15.7 (13.8-17.1)	21.0 (19.2-23.4)	42.1 (40.7-43.6)	21.1 (20.2-22.8)	552 (543-566)

Table 2

Nutritional regime of some artificial relamation profiles

Depth, cm	BS+LLL				BS+LLL+Sand+GGC			
	*A**	*B**	*C**	*D**	*A*	*B*	*C*	*D*
0-10	2.2	0.1	1.47	34.7	2.41	0.22	1.46	35.5
10-20	2.13	0.18	1.54	33.8	2.47	0.21	1.49	34.8
20-30	2.08	0.16	1.51	29.5	2.38	0.19	1.50	32.6
30-40	2.11	0.16	1.44	25.3	2.31	0.14	1.46	31.1
40-50	1.97	0.14	1.44	24.2	2.29	0.13	1.44	27.4
50-60	0.66	0.06	1.21	13.3	1.18	0.06	1.21	17.6
60-70	0.58	0.06	1.21	12.7	1.12	0.05	1.19	16.4
70-80	0.58	0.05	1.17	12.4	1.03	0.05	1.16	15.7
80-90	0.51	0.04	1.15	12.3	0.72	0.03	1.11	14.2
90-100	0.46	0.04	1.14	12.2	0.68	0.03	0.09	13.1
100-110	0.45	0.03	1.12	12.2	0.09	0.008	0.09	4.7
110-120	0.45	0.03	1.11	11.9	0.09	0.008	0.08	4.8
120-130	0.42	0.02	1.10	11.8	0.19	0.02	0.35	57.6
130-140	0.42	0.01	1.09	11.7	0.19	0.01	0.31	57.2
140-150	0.41	0.01	1.09	11.7	0.17	0.01	0.29	56.1

A - humus, %; B – total nitrogen, %; C – mobile phosphorus, mg/100 g;
D – exangable potassium, mg/100 g

Table 3

The content of trace elements in black soil and rocks, mg / kg (numerator-AAB pH 4.8, denominator-1N HCl)

Soil	Fe	Mn	Zn	Cu	Ni	Cr
BS	5.0±0.5	140±3	16.0±1.3	1.96±0.4	1.8±0.3	1.0±0.04
	1747±7	535±17	62.0±8	8.3±0.4	8.4±0.4	3.0±0.2
LLL	5.0±0.5	70±3	10.0±1.1	2.5±0.04	3.6±0.6	2.0±0.1
	1283±7	343±20	81.0±8.1	6.6±0.2	7.9±0.5	3.1±0.2
RBL	4.3±0.2	62±4	11.3±2.2	2.8±0.2	3.8±0.2	2.2±0,1
	1243±3	348±7	28.3±2,4	6.8±0,2	8.43±0.1	3.0±0.2
RBC	4.0±0.2	68±3	9.5±1.8	2.3±0.2	3.2±0.2	1.9±0.1
	1848±108	467±7	35.0±3.6	8.57±0.3	11.3±0.3	3.8±0.1
GGC	4.4±0.1	51±4	5.9±1.7	2.2±0.3	2.7±0.15	1.4±0.21
	1082±43	155±15	16.3±3	7.3±0.7	4.0±0.32	3.5±0.3
DGC	11.2±2	206±19	8.7±1.1	2.2±0.2	5.8±0.8	1.3±0.1
	2005±111	2053±80	35.8±4.4	8.22±0.6	9.2±1	3.0±0.3
GLC	4.7±0.5	50±2	11.7±0.6	2.1±0.3	1.5±0.2	1.3±0.02
	1330±147	291±14	37.2±1.9	7.9±0.2	3.5±0.5	3.3±0.5

The increased concentration of manganese in dark gray clay limits the prospect of using this substrate during creating of reclaimed land.

Meanwhile, the content of trace elements in the remaining weathered rocks did not exceed the indicators of the black soil. The highest degree of microelements balance was fixed loess like and red-brown loam, red-brown and gray-green clay.

Elongated plant melioration leads to an increase in the share of available reserve of trace elements in these substrates during the process of biological weathering of rocks.

The level of trace elements in weathered rocks did not exceed the indicators of black soil even after 14 years of their phytomelioration.

Trace elements spectrum was studied to assess the species differences in their concentration in the vegetative mass of plants, when growing barley, peas and alfalfa in the pot experiments with black soil and three phytomeliorated rocks.

The results of these studies are shown in figures 3 and 4.

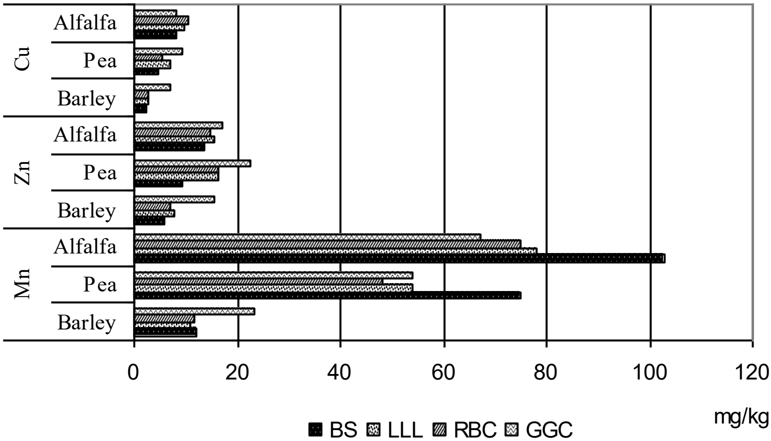

Fig. 3. Mn, Zn, Cu content in aboveground biomass of crops

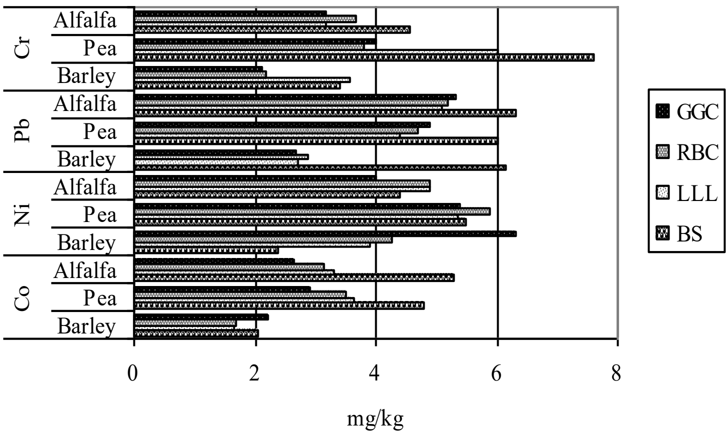

Fig. 4. Co, Ni, Pb, Cr content in aboveground biomass of crops

It was found that the content of trace elements in the aboveground mass of barley, peas and alfalfa grown on three phytomeliorated rocks (loess-like loam, red-brown and gray-green clay) it is comparable to the data obtained when growing these plants on black soil.

Evaluation of enzymatic activity in phytomeliorated rocks

Research of enzymatic activity in rocks after their 33-year phytomelioration. The results of research indicate that the change in humus content is slow, while the activity of enzymes increases significantly. For example, the humus content in the loess like loam taken directly from the quarry was 0.5 %. However, invertase activity was not observed here. The content of humus in the upper 20-cm layer increases almost 2 times after 33-year phytomelioration of loess like loam (Fig. 5).

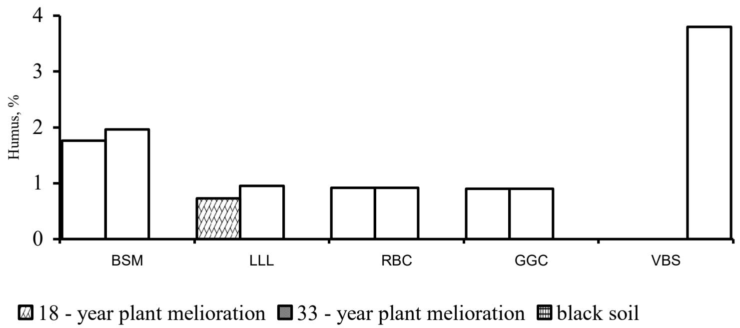

Fig. 5. A change in the content of humus in edaphotope under phytocenozes in the layer 0-20 cm

The upper 40-centimeter thickness of the studied edaphotopes under vegetation cover reaches an average level of enrichment with the enzyme invertase. The greatest growth (up to 2.5 times) of this parameter is observed in the layer of 0-20 cm loess-like loam (Fig. 6). Red-brown clay had the poorest level of invertase activity after 18 years of plant melioration process. The increase in the level of invertase activity in the upper 20-cm horizon was 1.6 times in the next 15 years of phytomelioration. The growth of invertase activity in the upper layer of black soil and gray-green clay is less intense. The active interaction of roots and soil microorganisms leads to the fact that the rocks are cultivated faster. The increase in the level of invertase activity occurs more intensively under the influence of vegetation process over the past 15 years in the 20-40 cm layer compared to the 0-20 cm layer.

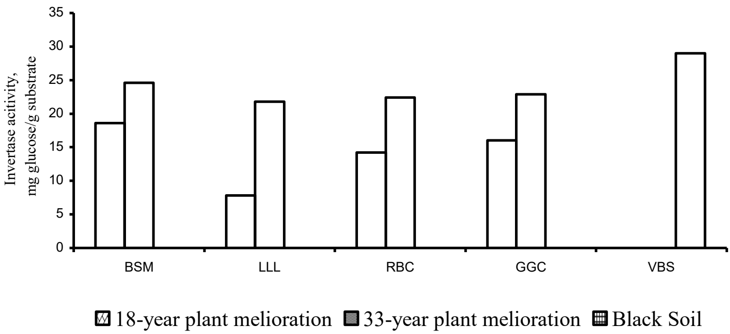

Fig. 6. Changes in the level of invertase activity of edaphotopes under the vegetation cover in the 0-20 cm layer

The parameters of this indicator in the upper layer of black soil and red-brown clay grow by 1.8 times, in gray-green clay - by 2.3 times, and in loess like loam by almost 4 times (Fig. 7).

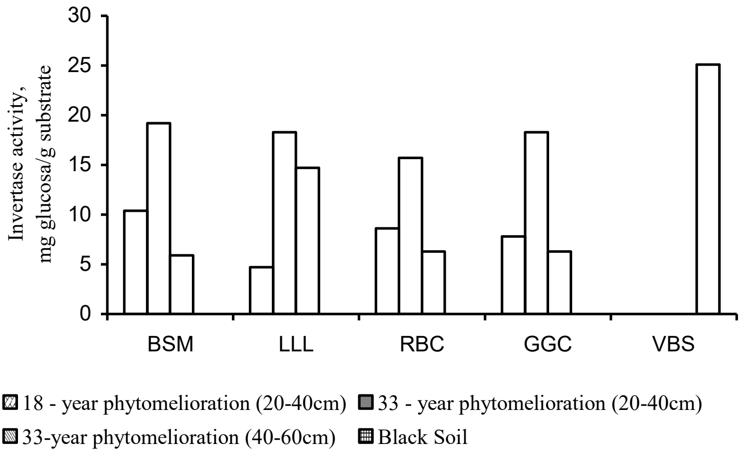

Fig. 7. Changes in the level of invertase activity of edaphotopes under phytocenoses in the 20-60 cm layer

Intensive hydrolysis of glycosides in a layer of 40-60 cm was recorded only for loess like loam. Invertase activity of loess like loam in the horizon of 40-60 cm is 14.7 mg of glucose/g of suspension after 33 years of this edaphotope under the influence of vegetation process. This exceeds the 18-year level of this indicator for a layer of 20-40 cm by almost 3 times. Thus, over the past 15 years, the loess-like loam layer has been cultivated most intensively under the influence of phytomelioration. In other words, loess-like loam is the most suitable rock for the formation of zonal soil in natural conditions. Urease catalyzes the hydrolysis of urea to ammonia and plays an important role in optimizing the nitrogen balance in the edaphotopes of man-made landscapes. Urease activity in the 0-20 cm layer increased little after 18-year rocks phytomelioration (Fig. 8).

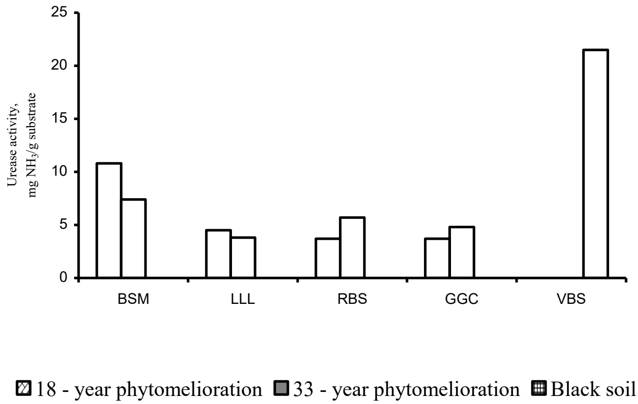

Fig. 8. Changes in the level of urease activity in the 0-20 cm edaphotops

The change in the level of urease activity in the 0-20 cm edaphotops due to the action of grass groups of meliorated rocks increased little. The low level of enrichment of rock substrates with urease is explained by the fact that legumes are able to provide themselves with nitrogen. All the studied edaphotopes are characterized by a poor degree of enrichment of urease activity in the upper 20 cm and after 33 years of phytomelioration. It is known that the phosphatase enzyme is directly related to the processes of biological mobilization of phosphorus from rocks. The upper layers of black soil and loam increased phosphatase enrichment to an average degree in the meliorated rocks after their long-term staying.

Red-brown and gray-green clays were characterized by a low degree of phosphatase enrichment. The indicators of these two edaphotopes are close to the indicators indicating the average degree of enrichment of rocks with phosphatase.

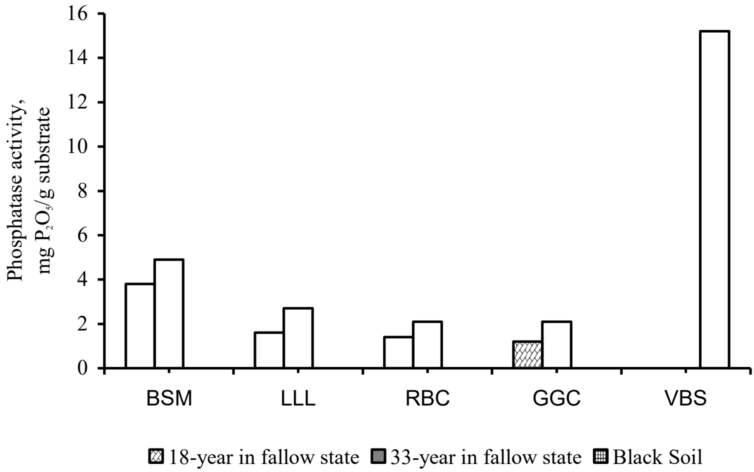

Fig. 9. Changes in the level of phosphatase activity of edaphotopes in the layer 0-20 cm

The growth of phosphatase activity in the upper layer of 0-20 cm occurs under the impact of vegetation growth (Fig.10).

Indicators of activity of this enzyme under phytocenoses are higher in comparison with areas without vegetation cover.

The process of phosphatase accumulation is quite intense in the 0-20 cm layer.

The level of phosphatase activity after a 33-year stay of edaphotopes on the day surface in the 40-60 cm layer is almost equal to the 18-year indicators of this parameter for the 20-40 cm layer (Fig. 11 and 12).

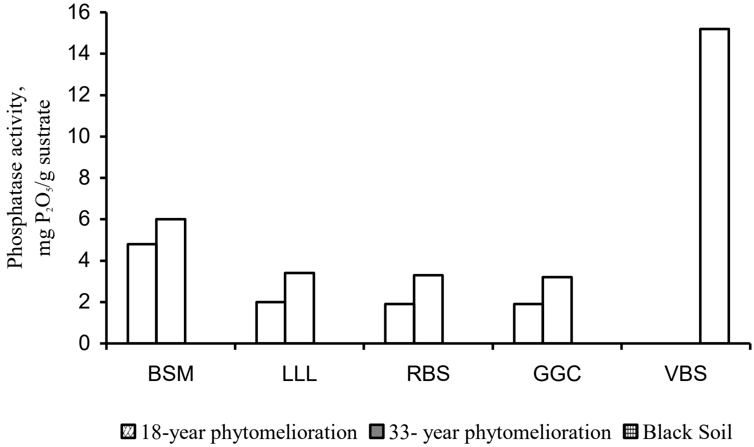

Fig. 10. Changes in the level of phosphatase activity of edaphotopes under phytocenoses in the 0-20 cm layer

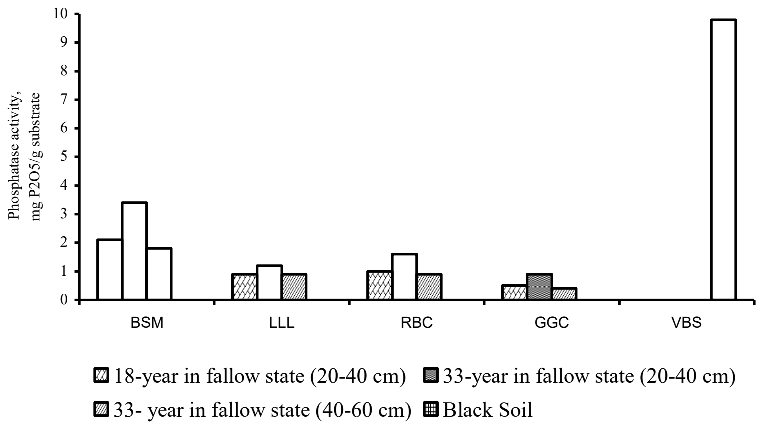

Fig. 11. Changes in the level of phosphatase activity in the edaphotop layer 20-60 cm (without vegetation)

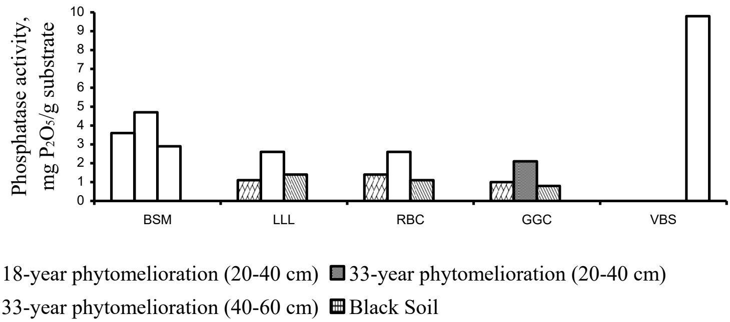

Fig. 12. Changes in the level of phosphatase activity of edaphotopes under phytocenoses in the 20-60 cm layer

It is known that the processes of soil formation in phytomeliorated rocks occur with the participation of dehydrogenase. The effect of grass vegetation is outstanding for the formation of dehydrogenase potential. The positive effect of biocenoses is shown in the upper layer of black soil. Meantime the level of dehydrogenase activity under phytocenoses exceeds the activity of this enzyme in samples from the virgin field of black soil (Fig. 13 and 14).

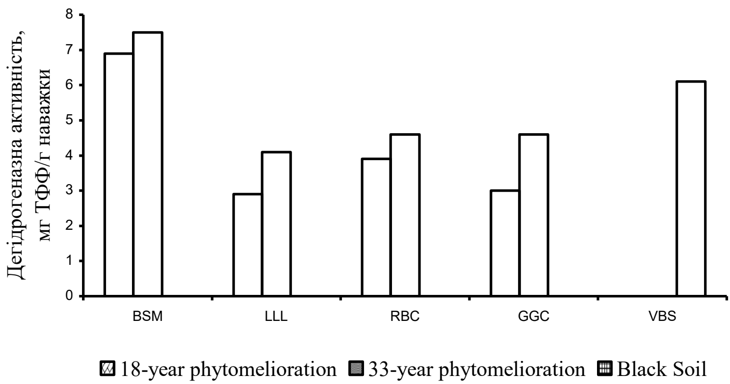

Fig. 13. Changes in the level of dehydrogenase activity in the 0-20 cm layer under the influence of grass groups

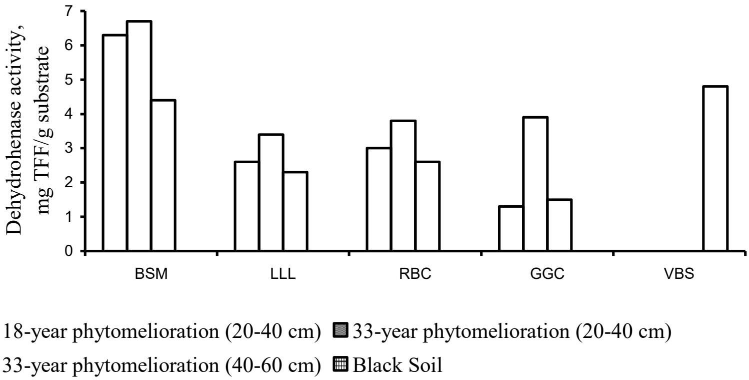

Fig. 14. Changes in the level of dehydrogenase activity in the 20-60 cm layer due to the action of phytocenoses

The greatest accumulation of the enzyme dehydrogenase occurred over the past 15 years in a layer of gray-green clay due to the phytomeliorative action of herbal groups. The greatest mass of roots and microorganisms accumulated in the gray-green clay stratum. So, the 33-year stay of tertiary clay deposits on the daytime surface was accompanied by an intensive accumulation of dehydrogenase. Usually, catalase activity is evaluated as an indicator of the functional activity of the soil microflora (Fig. 15).

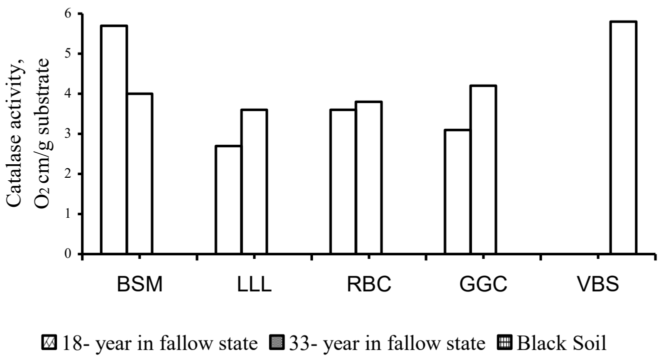

Fig. 15. Changes in the level of catalase activity in the 0-20 cm layer of food plants that are in the vapor (without vegetation) state

Catalase activity in the 0-20 cm layer reached an average degree of enrichment after 18-year stay on the daytime surface. A significant increase in the activity of catalase in the layer 0-20 cm edafotop – up to 70% relative to the control was recorded after 33 years weathering. The level of catalase activity in the edaphotop stratum after 33-year staying under phytocenoses did not exceed the level of activity of this enzyme in zonal black soil and amounted to 88% relative to the control (Fig. 16).

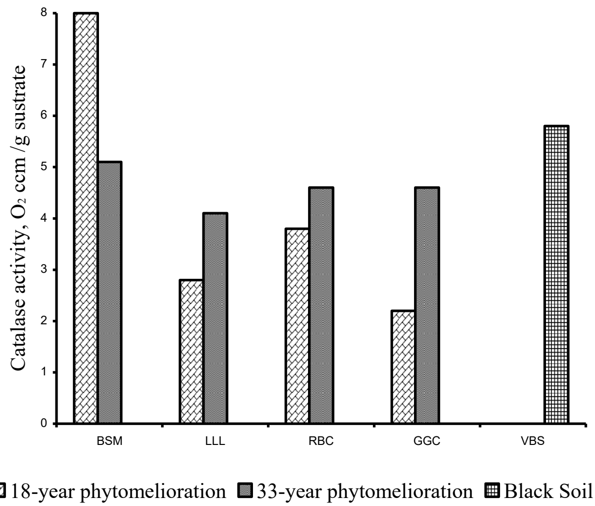

Fig. 16. Changes in the level of catalase activity in the 0-20 cm layer in edaphotopes due to the action of herb groups

Consequently, the most intense increase in the activity of redox enzymes occurs in the layer of gray-green clay. Thus, the long-term stay of technogenic landscapes on the day surface (as fallow state, or in the state of arable land) contributes to a significant increase in their enzyme potential. At the same time, there is a more intensive accumulation of activity of hydrolytic enzymes in comparison with redox enzymes. In other words, hydrolysis reactions of complex organic compounds predominate in long term phytomeliorated rocks, and they enrichment of edaphotopes with nutrients available to plants and microorganisms. The process of synthesis of humic substances is less intensive. A long stay of rocks on the surface of the day without vegetation contributes to the fact that the level of enzymatic activity they are slowly approaching zonal soils. However, these edaphotopes are characterized as "weakly biogenic". At the same time, it is interesting that in the top layer of the black soil and in the gray-green clay. The recovery of the biochemical potential is faster. That is why, according to the level of activity of enzymes, gray-green clay can be attributed to the mid-biogenic edaphotopes. Phytocenoses contribute to a significant increase in the accumulation of enzyme activity in rocks. Therefore, even in the layer of 20-40 cm, almost all the studied edafotopi, except for red-brown clay, are characterized as biogenic. Thus, the study of the formation of the enzyme potential in the top and subtop stratums of edafotops indicates their gradual approach to the level of zonal black soil due to the process of phytomelioration. On this basis, the following can be argued: a) positive ecological rehabilitation of disturbed lands by open mining; b) restoration of the biochemical potential contributes to increasing the environmental stability of the created phytocenoses, which can withstand harsh steppe conditions and disturbed ecological balance in the region. These functional manifestations are the ecological basis that contributes to both the conservation of ecosystems and their restoration. The study of the growth of enzyme activity in the technozems allows us to know the initial stages of soil formation, the speed and direction of soil-forming processes and ways to accelerate them, the laws of soil formation on a zonal scale.

Numerous studies have shown that enzymes are resistant even to environmental conditions that are unfavorable for the life of microorganisms and the development of plant root systems. The changes to the soil organic carbon, activities of soil enzymes and glomalin-related soil protein were measured and the effects of arbuscular mycorrhizal fungi (AMF) on activities of soil enzymes and carbon sequestration capacity in reclaimed mine soil were analyzed [12].

It was established that enzyme activity differed in different seasons [13]. All samples indicated different enzyme activity in different seasons.

High enzyme activity was caused by high humidity [14]. Effect of seasonality on biological activity of litter (loss of biomass, enzyme activity, decomposition rate and C : N ratio) in litter layer was approved as well [15]. So, at the first stages of the formation of technogenic ecosystems, the leading role is played by enzyme activities, which are the starting biotic components.

After inoculation of edaphotop spores of microorganisms brought from nearby natural ecosystems, conditions are gradually created favorable for the functioning of newly formed biocenoses.

Over time, the level of enzyme activity stabilizes, becomes independent of seasonal fluctuations in the number of microorganisms and is a pool that reflects the quality indicators of edaphotops.

Thus, the enzyme pool edaphotops takes part in all stages of transformation of organic compounds.

It is an important regulator of the biochemical homeostasis of waste lands and performs biogeocenotic functions, ensuring the continuity of metabolic processes even in extreme conditions typical of man-made ecosystems.

Environment management of reclaimed minelands

Improving the management of the reclamation works is proposed to increase the fertility of man-made lands.

This mode means studying the sequence of removal of black soil mass on virgin lands and applying it to the surface of heaps.

Edaphic characteristics of technosol act as indicators of reclaimed lands. Such parameters of edaphic structures of technosol are able to provide stable and high crop yields.

The presence of shrinkage deformations requires additional measures to optimize the terrain.

The costs at the biological stage of reclamation will be determined by a set of technical and phytomeliorative measures that will improve the structural and aggregate state, water-air and nutrient regimes, as well as the biological activity of the restored lands.

It was established that the quality management of technozems provides for the implementation of phytomeliorative measures that will restore their maximum fertility to the level of zonal soils (Table 4).

Table 4

System management of individual properties of technozems

Factors	Parameters level	Restoration measures	Limiting factor
Total	Low content humus and nitrogen	Ability to change multiple measures	Humus accumulation
			Desalinization
			The increase of biological activity
	Insufficient availability of available phosphorus	Ability to change one-time events for a short period of time	Providing nutritional elements
			Moisture accumulation and water conservation
			Optimization of air mode
Specific	Low aggregation and formation of soil crust on the surface of technozems	Ability to change it for a long time	Increasing the water resistance of structural units
	High level of toxic salts and heavy metals		Desalination and phytoremediation
	Heavy soil texture	Cannot be changed	

Creating a common information base for managing edaphic parameters of various technozem structures is the main prerequisite for monitoring the state of nutrient regimes of "young" soils (Fig. 15).

The costs at the biological stage of reclamation are determined by a set of agrotechnical and phytomeliorative measures that will improve the structural and aggregate state, water-air and nutrient regimes, as well as the biological activity of the restored land.

Creating a common information base for managing edaphic parameters of various technozem structures is the main prerequisite for monitoring the state of nutrient regimes of "young" soils (Fig. 15).

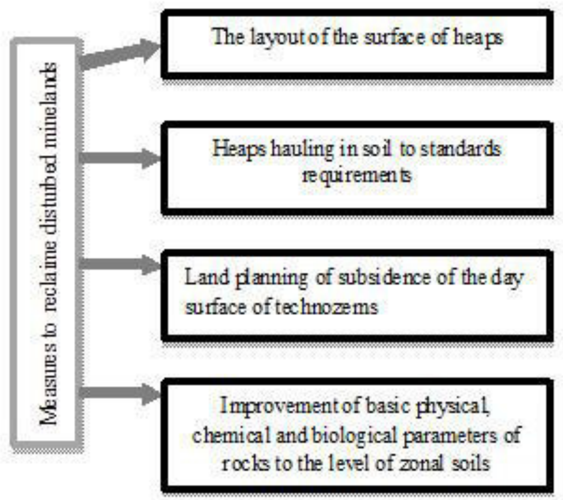

Fig. 17. List of the main types of work aimed at restoring reclaimed dumps

Improving the management of the regime of reclamation works is proposed to increase the fertility of man-made soils.

Conclusions.

1. The use of efficient technologies reclamation aimed at forming technozems as a bulk layer of fertile soil, and without coverage was estimated. An edaphic parameters of technozems different designs were estimated to develop reasonable and advanced mining technology and biological stages of biological reclamation.

The parameters change edaphic characteristics for various technozems models were analyzed for a long - term phytomelioration and forthcoming using of reclaimed land patterns depending on the time factor as soil formation the "zero moment". The research results complement information database edaphic indicators and their dynamic changes in the different quality models technozems, enabling more effective management of technological processes during the stages of mine technical and biological reclamation.

2. Change of a level of enzymes activity in meter thickness of artificial land reclamation profiles for certain historical time is demonstrated. The features of accumulation of enzymatic activity at formation of young soils in conditions of technogenic landscapes are revealed. The orientation of biochemical processes in thickness of artificial edaphotopes descends the same as in zonal chernozem: reactions of hydrolysis of complex organic compounds dominate, and process of synthesis of humus substances is carried out slowly. For 33 years of stay on the daily surface of phosphatase, the activity of artificial edaphotopes in terms of the degree of enrichment was the highest in comparison with the activity of other hydrolytic enzymes studied. The most intensive growth of invertase activity occurred in these rocks over the past 15 years. Invertase activity can serve as an indicator of the cultivation of man-made landscapes.

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HE EFFECT OF THE DECREASE IN POWER INTENSITY OF SELF-OSCILLATING GRINDING IN A TUMBLING MILL WITH A REDUCTION IN AN INTRACHAMBER FILL

Deineka K. Yu.
Technical College of the National University of Water and Environmental Engineering (TK of NUWEE),
Candidate of Technical Sciences (Ph.D),
Chairman of the Cycle Commission "Industry Engineering", Ukraine

Naumenko Yu. V.
National University of Water and Environmental Engineering (NUWEE), Doctor of Technical Sciences (Dr.Sc), Associate Professor, Professor of the Department of Construction, Road, Reclamation, Agricultural Machines and Equipment, Ukraine

Abstract

Effect of the degree of chamber filling with the charge on efficiency of the self-oscillating grinding process in a tumbling mill has been assessed.

By using the approximate analytical and experimental method, dynamic effect of increasing the self-oscillating impact action of grinding fill on the crushed material was compared with the conventional steady-state motion mode. A significant increase in average sums of vertical components of the self-oscillating collision momenta and the average sums of power of such components with a decrease in the chamber filling degree was found. Manifestation of this effect is due to the increase in the self-oscillations swing with decreasing filling.

Technological effect of significant decrease in the specific power intensity and productivity growth of the innovative self-oscillating grinding process as compared to the characteristics of the conventional steady-state process with a reduction in the chamber filling degree have been experimentally established.

The effects established in operation have allowed us to predict rational parameters of the self-oscillating grinding process carried out in a tumbling mill with variation in the chamber filling degree

Keywords: tumbling mill, chamber filling degree, fill self-oscillation, specific power intensity in grinding

Introduction

Due to a series of their operational advantages, the tumbling type mills remain to be the main equipment in many industries for small-and large-tonnage fine grinding of solid materials.

Replacement of the conventional steady-state grinding process with a novel self-oscillating process improves existing equipment of relatively low power efficiency [1,2]. Use of the phenomenon of excitation of self-oscillations makes it possible to apply conventional solutions to designing the tumbling mills with a smooth working chamber surface without additional activating elevators in a form of protruding elements which undergo rapid abrasive wear.

On the other hand, significant variability of the self-excited pulsation behavior of the rotating chamber fill depending on structural, kinematic, and technological parameters of equipment operation [3-7] makes it difficult to establish rational conditions for effective realization of the self-oscillating grinding process.

The data obtained from numerical simulation and experiments have shown a significant influence of the chamber filling degree κ on parameters of the grinding process occurring in the tumbling mills. This effect consists in a growth of the impact action of the grinding fill with a decrease in κ. Instead, with an increase in κ, impact action is reduced and the abrasion effect increases, dissipation of kinetic energy of the grinding bodies increases because of collision and the grinding power efficiency is reduced. However, these results relate just to a conventional grinding process at simple conditions of steady-state intrachamber fill motion.

No models have been constructed to date to determine effect of the chamber filling degree κ on grinding parameters in self-excitation of a complex mode of transient motion of the pulsating fill. Absence of such models is particularly negative in the case of realization of the innovative self-oscillating grinding process in tumbling mills.

In view of the above, the problem of predicting the effect of the degree of filling the chamber with the fill on dynamic force of the grinding bodies applied to the particles of the crushed material as well as on the technological and power efficiency of the process of self-oscillating grinding in the tumbling mill seems rather relevant.

1. The aim and objectives of the study

The study objective consists in establishing the effect of chamber filling degree on characteristics of dynamic action of the fill and technological and power parameters of conventional steady-state and innovative self-oscillating processes of grinding in a tumbling mill. It would make it possible to predict efficiency of realization of self-oscillating grinding processes at a varying degree of filling the chamber with fill.

To achieve this objective, the following tasks were set: to perform analytical and experimental modeling of dynamic action of the fill; to perform experimental modeling of technological and power parameters of the grinding processes in a tumbling mill.

2. The process of grinding in a tumbling mill

It is convenient to use dimensionless criterial parameters for modelling the grinding process in a tumbling mill. The drum rotation is defined by relative velocity $\psi_\omega = \omega\sqrt{R/g}$ where ω is the angular velocity of rotation; R is the radius of the drum chamber; g is gravitational acceleration.

Useful filling of the drum is defined by the degree of filling the chamber with the fill $\kappa = w/(\pi R^2 L)$ where w is the volume of the charge at rest; L is the length of the drum chamber.

Conventional steady-state grinding process in a tumbling mill is realized at a moderate relative rotational velocity ψ_ω=0.7-0.9. At the chamber filling degree κ=0.25-0.45, the fill circulates in a three-phase mode with formation of three flow zones in the chamber cross-section (Fig. 1).

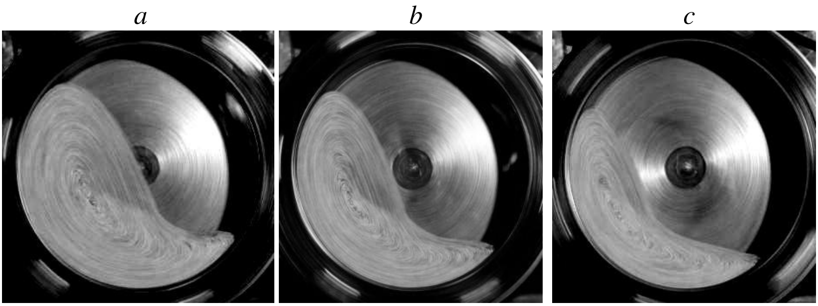

Fig. 1. A pattern of steady-state fill motion with absolute d and relative $d/(2R)$=0.01-0.03 particle size, ψ_ω=0.75: a – κ=0.45; b – κ=0.35; c – κ=0.25

Conventionally, grinding (Fig. 2) is performed mainly by an impact action on the boundary BC of transition from the zone of restricted fall 2 to the zone of the creeping layer 3 and the abrasion action in the creeping layer 3. Technological results of grinding resulting from such actions are commensurate.

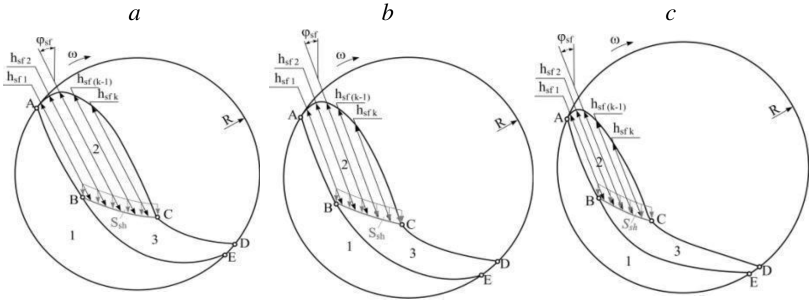

Fig. 2. Design diagram for determining the fill impact in a steady motion at *d*/(2*R*)=0.01-0.03 and ψ_{ω}=0.75 (according to Fig. 1) (1 – solid state zone; 2 – restricted fall zone; 3 – creeping zone): a – κ=0.45; b – κ=0.35; c – κ=0.25

The innovative self-oscillating grinding process in a tumbling mill is realized in the case of increased rotational velocity $\psi_{\omega}\approx$1-1.2. At κ=0.25-0.45, a pulsating zone of oscillating fill flow appears due to self-excitation. The creeping layer and the restricted fall zone partially (Fig. 3 and 4) or completely (Fig. 5) transform into this zone.

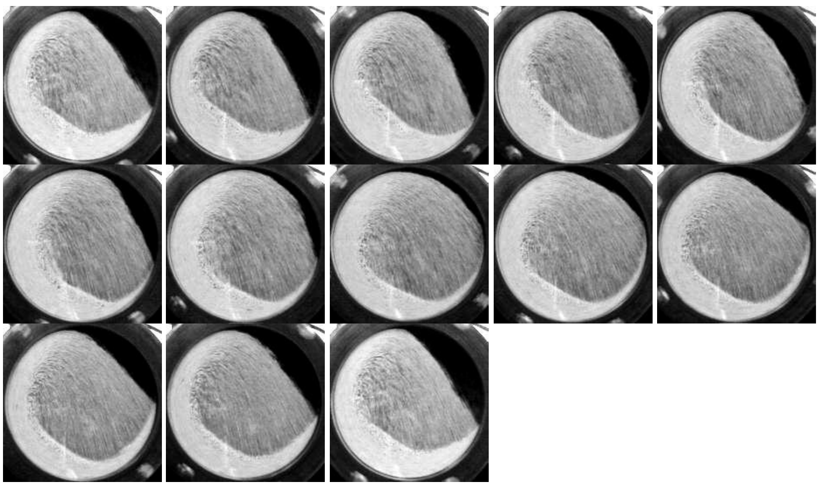

Fig. 3. Consecutive patterns of the fill motion over time for one period of self-oscillations with a maximum swing at *d*/(2*R*)=0.01-0.03, κ=0.45 and ψ_{ω}=1.1

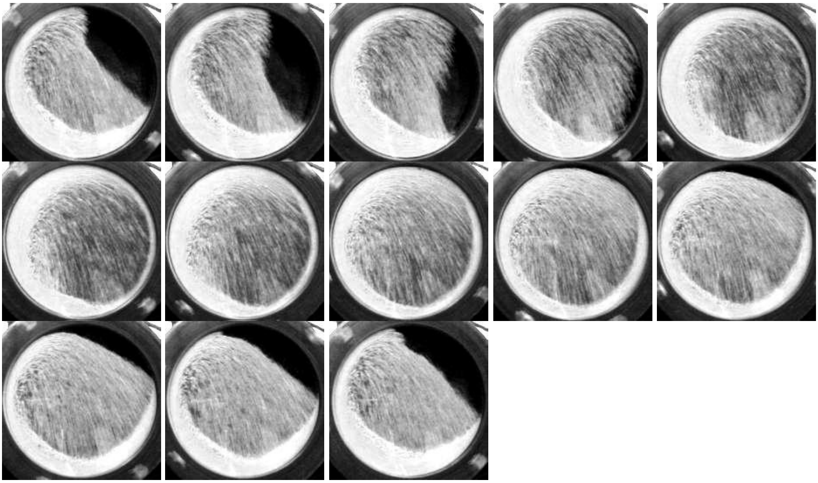

Fig. 4. Consecutive patterns of the fill motion over time for one period of self-oscillations with a maximum swing at $d/(2R)$=0.01-0.03, κ=0.35 and ψ_ω=1.075

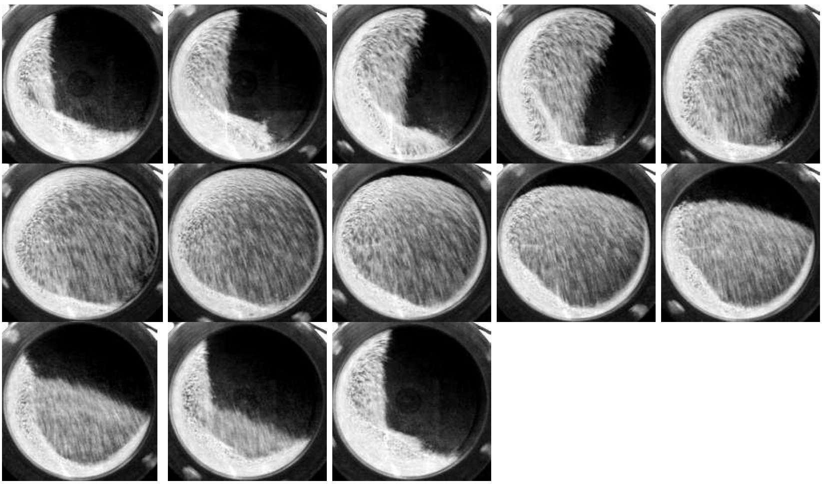

Fig. 5. Consecutive patterns of the fill motion over time for one period of self-oscillations with a maximum swing at $d/(2R)$=0.01-0.03, κ=0.25 and ψ_ω=1.05

The self-oscillating grinding (Fig. 6) is realized by an impact action on the BCE boundary including sections BC and CE. The BC boundary corresponds to the transition of the restricted fall zone 2 (Fig. 6a and 6b) or the pulsation zone 4 (Fig. 6c) to the solid state zone 1. The CE section corresponds to the contact of the pulsation zone 4 with the chamber surface.

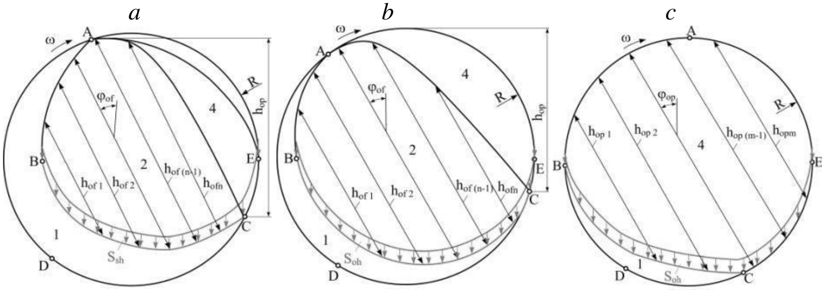

Fig. 6. Calculation diagram for determining impact of the fill in a self-oscillating motion at $d/(2R)$=0.01-0.03 and ψ_{ω}=1.075 (according to Fig. 3-5) (1 – solid state zone; 2 – restricted fall zone; 4 – pulsation zone): a – κ=0.45; b – κ=0.35; c – κ=0.25;

The adopted qualitative model of fill behavior (Fig. 1–6) makes it possible to perform quantitative analysis of dynamic action for various chamber filling degrees κ.

3. Analytical modeling of impact action of the fill

It is convenient to estimate technological results of the grinding process at conventional and self-oscillatory modes of a mill by performing a comparative analysis of parameters for the filling's impact action only.

When a grinding particle M_v hits the respective surface of the transition between zones of the filling's flow, there occurs a jump-like ultimate change in its speed over a small time τ. In this case, the contact's surface is exposed to impact force $\overline{F}_v$. A measure for the impact interaction is the impact pulse $\overline{S}_v = \int_1^{\tau} \overline{F}_v dt$.

Duration of the impact is very low ($\tau \approx 10^{-2} \ldots 10^{-4}$ s). Since the impact pulse S_v has a finite value, the impact force module of the milling body could be quite large ($F_v \to \infty$ at $\tau \to 0$), which ensures the implementation of grinding process via the impact action. In this case, effect of the non-impact forces can be disregarded.

It is expedient to estimate the technological effect of impact action not based on a value for the impact force, but rather based on its pulse, work, and power.

An impact pulse corresponds to a change in the amount of motion $\Delta\overline{Q}_\nu$ by a particle of mass m_ν over the time of impact

$$\overline{S}_\nu = \Delta\overline{Q}_\nu = m_\nu(\overline{u}_\nu - \overline{v}_\nu), \tag{1}$$

where $\overline{v}_\nu$ is the speed of a particle before the impact, $\overline{u}_\nu$ is the speed of a particle after the impact.

Work of the impact force corresponds to a change in the kinetic energy ΔT_ν of a particle during impact

$$A_\nu = \Delta T_\nu = m_\nu(\overline{u}_\nu - \overline{v}_\nu)^2/2. \tag{2}$$

The largest, in terms of magnitude, and defining, in terms of effectiveness of the implementation of an operating process, component of the impact action is vertical. The impact under consideration is a complex non-perfect interaction between particles. Then the expressions for the vertical component of the impact pulse and work of the vertical component of the impact force, according to (1) and (2), can be approximately represented in the form

$$S_{\nu h} = m_\nu K_{\nu h\nu} v_{h\nu}, \tag{3}$$

$$A_{\nu h} = (m_\nu K_{\nu h\nu}^2 v_{h\nu}^2)/2, \tag{4}$$

where $v_{h\nu}$ is the vertical component of particle's velocity $\overline{v}_\nu$ before the impact, $K_{\nu h\nu}$=0-1 is the coefficient of loss of the vertical component $v_{h\nu}$ of velocity $\overline{v}_\nu$ during impact.

Assuming

$$v_{h\nu} = \sqrt{2gh_\nu},$$

expressions (3) and (4) take the form

$$S_{\nu h} = \sqrt{2g}\, m_\nu K_{\nu h\nu} \sqrt{h_\nu}, \tag{5}$$

$$A_{\nu h} = g m_\nu K_{\nu h\nu}^2 h_\nu, \tag{6}$$

where h_ν is the height of particle's fall before the impact, g is the gravitational acceleration.

We shall further consider the vertical component of the impact action of particles at the surface of transition between zones of the filling's motion for N (ν=1.2,…,N) grinding particles over an arbitrary time span Δt. Then, according to (5) and (6), expressions for the sum of vertical components of the impact pulses and the sum of work of the vertical components of impact forces over Δt take the following form

$$S_h^{\Delta t} = \sqrt{2g}\sum_{\nu=1}^{N} m_\nu K_{vh\,\nu}\sqrt{h_\nu}\ , \tag{7}$$

$$A_h^{\Delta t} = g\sum_{\nu=1}^{N} m_\nu K_{vh\,\nu}^2 h_\nu\ , \tag{8}$$

The mean, over time Δt, value for the total power of the vertical components of impact forces corresponds to the average value for the sum of work of such forces A_h^{ut} per unit time (1 s) and is equal to

$$P_h = A_h^{ut} = A_h^{\Delta t} / \Delta t\ . \tag{9}$$

For the case of the conventional steady mode of operation (Fig. 3), the sum of the vertical components of impact pulses over Δt (7) can be calculated from formula

$$S_{sh}^{\Delta t} = \sqrt{2g}\, m_s^{\Delta t} K_{vhs}\sqrt{h_{sf}}\ , \tag{10}$$

where $m_s^{\Delta t} = \sum_{\nu=1}^{N} m_{\nu s}$ is the sum of masses of N particles, K_{vhs} is the average coefficient of loss of the vertical component of particles' velocity in a non-free fall zone, which can be determined using a method of the filling's flow visualization, $h_{sf} = cos\,\varphi_{sf}\sum_{i=1}^{k}\frac{h_{sfi}}{k}$ is the average height of fall of N particles in a non-free fall zone, h_{sfi} is the elementary linearized trajectory of fall of a separate particle in a non-free fall zone, φ_{sf} is the angle of inclination to vertical of the averaged linearized trajectories of falling particles in a non-free fall zone.

The sum of the vertical components of impact pulses under a steady mode (10) for a single rotation of the drum is

$$S_{sh}^{tn} = \sqrt{2g}\, m_s^{tn} K_{vhs}\sqrt{h_{sf}}\ , \tag{11}$$

where $m_s^{tn} = mK_{ts}$ is the sum of masses of grinding particles that execute an impact within a single turn of the drum; $m=\pi R^2 L\kappa\rho$ is the mass of the milling fill; R is the radius of the drum's chamber; L is the length of the drum's chambers; κ is the degree of filling the chamber with a fill; ρ is the density of the milling fill at rest; K_{ts} is the loading's turnover under a steady mode.

Turnover describes the number of periods of circulation of the fill over a single drum rotation

$$K_t = (2\pi)/(t_c\omega),$$

where t_c is the duration of circulation period of a fill in the rotating drum's chamber, ω is the angular speed of the drum. Turnover can be determined based on an experimental analysis of the fill flow patterns.

The mean, over a single rotation of the drum, value for the sum of the vertical components of impact pulses under a steady mode per unit time is

$$S_{sh}^{ut} = S_{sh}^{tn} / T_s, \quad (12)$$

where $T_s = 2\pi / \psi_{\omega s}\sqrt{R/g}$ is the period of drum rotation under a steady mode, $\psi_{\omega s} = \omega_s\sqrt{R/g}$ is the relative rotation speed of the drum under a steady mode, ω_s is the angular velocity of the drum under a steady mode. Then, taking into consideration (11) and (12),

$$S_{sh}^{ut} = \left(gmK_{ts}\psi_{\omega s}K_{vhs}\sqrt{h_{sf}/R}\right)/\left(\sqrt{2}\pi\right). \quad (13)$$

For the case of the self-oscillatory mode of operation (Fig. 4), the sum of the vertical components of impact pulses over Δt (7) can be approximately calculated from expression

$$S_{oh}^{\Delta t} = S_{ofh}^{\Delta t} + S_{oph}^{\Delta t} = \sqrt{2g}\left(m_{of}^{\Delta t}K_{vhof}\sqrt{h_{of}} + m_{op}^{\Delta t}K_{vhop}\sqrt{h_{op}}\right), \quad (14)$$

where $S_{ofh}^{\Delta t}$ is the sum of the vertical components of impact pulses of particles in a non-free fall zone over Δt; $S_{oph}^{\Delta t}$ is the sum of the vertical components of impact pulses of particles in a pulsating zone over Δt; $m_{of}^{\Delta t} = \sum_{\nu=1}^{N_f} m_{\nu o}$ is the sum of masses of N_f particles in a non-free fall zone, which execute an impact over Δt; $m_{op}^{\Delta t} = \sum_{\nu=1}^{N_p} m_{\nu o}$ is the sum of masses of N_p particles in a pulsating zone, which execute an impact over Δt; K_{vhof} is the average coefficient of loss of the vertical component of velocity by particles in a non-free fall zone; K_{vhop} is the average coefficient of loss of the vertical component of velocity by particles in a pulsating zone, $h_{of} = cos\,\varphi_{of}\sum_{i=1}^{n}\frac{h_{ofi}}{n}$ is the average height of fall of particles in a non-free fall zone, h_{ofi} is the elementary linearized trajectory of fall by a separate particle in a non-free fall zone, φ_{of} is the angle of inclination to vertical of the averaged linearized trajectories of falling particles in a non-free fall zone, h_{op} is the average height of fall by particles in a pulsating zone.

The sum of the vertical components of impact pulses under a self-oscillatory mode (14) for a single drum rotation is

$$S_{oh}^{tn} = \sqrt{2g}\left(m_{of}^{tn} K_{vhof}\sqrt{h_{of}} + m_{op}^{tn} K_{vhop}\sqrt{h_{op}}\right), \qquad (15)$$

where $m_{of}^{tn} = m\left(1-\kappa_{op}\right)K_{to}$ is the sum of masses of particles in a non-free fall zone, which execute an impact within a single drum rotation; $m_{op}^{tn} = \left(m\kappa_{op}\psi_{\omega op}\right)/\psi_{\omega o}$ is the sum of masses of particles in a pulsating zone, which execute an impact within a single drum rotation; K_{to} is the turnover of fill under a self-oscillatory mode (Fig. 5); κ_{op} is the mass fraction of the pulsating zone in the mass of the entire fill; $\psi_{\omega o} = \omega_o\sqrt{R/g}$ is the relative speed of rotation of the drum in self-oscillatory mode; ω_o is the angular velocity of drum under a self-oscillatory mode; $\psi_{\omega op} = 2\pi\sqrt{R/g}\,f_{op}$ is the relative circular frequency of the filling self-oscillations; f_{op} is the cyclic frequency of self-oscillations.

Parameters for self-oscillations can be determined experimentally using a method for the visualization of a filling's flow. The mass share of pulsating zone κ_{op} is calculated based on an analysis of flow patterns. The cyclic frequency of self-oscillations f_{op} is established from an analysis of transient flow modes. The absolute mass and the mass share of a non-free fall zone and a pulsating zone are variable parameters over the period of self-oscillations: $m_{of}^{tn} = var$, $m_{op}^{tn} = var$ and κ_{op}=var.

The mean, over a single drum rotation, value for the sum of the vertical components of impact pulses under a self-oscillatory mode per unit time is

$$S_{oh}^{ut} = S_{oh}^{tn} / T_o \ , \tag{16}$$

where $T_o = 2\pi / \psi_{\omega o} \sqrt{R/g}$ is the period of drum rotation under a self-oscillatory mode. Then, taking into consideration (15) and (16)

$$S_{oh}^{ut} = \frac{1}{\sqrt{2}\pi} gm \left[K_{to} \left(1 - \kappa_{op}\right) \psi_{\omega o} K_{vhof} \sqrt{\frac{h_{of}}{R}} + \kappa_{op} \psi_{\omega op} K_{vhop} \sqrt{\frac{h_{op}}{R}} \right]. \tag{17}$$

The average sum of the vertical components of impact pulses (17) over a period of self-oscillations varies from a minimum value when $\kappa_{op}=0$

$$S_{oh\,min}^{ut} = \left(gmK_{to} \psi_{\omega o} K_{vhof} \sqrt{h_{of} / R} \right) / \left(\sqrt{2}\pi \right), \tag{18}$$

to a maximal value when $\kappa_{op}=\kappa_{opmax}$

$$S_{ohmax}^{ut} = \frac{1}{\sqrt{2}\pi} gm \left[K_{to} \left(1 - \kappa_{opmax}\right) \psi_{\omega o} K_{vhof} \sqrt{\frac{h_{of}}{R}} + \kappa_{opmax} \psi_{\omega op} K_{vhop} \sqrt{\frac{h_{op}}{R}} \right]. \tag{19}$$

Expressions for the mean value of total power of the vertical components of the impact forces of fill can be derived by analogy to the above procedures for the formalization of average sums of the vertical components of impact pulses.

An expression for the total work of the vertical components of impact forces for a single drum rotation under a steady mode, taking into consideration (7), (8) and (11), takes the form

$$A_{sh}^{tn} = gm_s^{tn} K_{vhs}^2 h_{sf} \ . \tag{20}$$

The average total power of the vertical components of impact forces, taking into consideration (9), (12), (13) and (20), is

$$P_{sh} = \left(gmK_{ts} \psi_{\omega s} \sqrt{g/R} K_{vhs}^2 h_{sf} \right) / (2\pi). \tag{21}$$

An expression for the total work of the vertical components of impact forces for a single drum rotation under a self-oscillatory mode, taking into consideration (7), (8) and (15), takes the form

$$A_{oh}^{tn} = g \left(m_{of}^{tn} K_{vhof}^2 h_{of} + m_{op}^{tn} K_{vhop}^2 h_{op} \right). \tag{22}$$

The average total power of the vertical components of impact forces under a self-oscillatory mode, considering (9), (16), (17) and (22), is

$$P_{oh} = gm\sqrt{g/R}\left[K_{to}\left(1-\kappa_{op}\right)\psi_{\omega 0}K_{vhof}^{2}h_{of} + \kappa_{op}\psi_{\omega op}K_{vhop}^{2}h_{op}\right]/2\pi\ .\qquad(23)$$

The average sum of power (23) over a period of self-oscillations changes, similar to (18) and (19), from a minimum

$$P_{oh\,min} = \left(gm\sqrt{g/R}K_{to}\psi_{\omega o}K_{vhof}^{2}h_{of}\right)/(2\pi),\qquad(24)$$

to a maximal value

$$P_{ohmax} = \frac{1}{2\pi}gm\sqrt{\frac{g}{R}}\left[K_{to}\left(1-\kappa_{opmax}\right)\psi_{\omega o}K_{vhof}^{2}h_{of} + \kappa_{opmax}\psi_{\omega op}K_{vhop}^{2}h_{op}\right].\qquad(25)$$

The expression for the ratio of average values for the sum of the vertical components of impact pulses under a self-oscillatory and a steady mode per time unit, taking into consideration (13) and (17), takes the form

$$\frac{S_{oh}^{ut}}{S_{sh}^{ut}} = \frac{K_{to}\left(1-\kappa_{op}\right)\psi_{\omega o}K_{vhof}\sqrt{h_{of}} + \kappa_{op}\psi_{\omega op}K_{vhop}\sqrt{h_{op}}}{K_{ts}\psi_{\omega s}K_{vhs}\sqrt{h_{sf}}}.\qquad(26)$$

Ratio (26) over a period of self-oscillations changes ($S_{oh}^{ut}/S_{sh}^{ut} = var$), taking into consideration (18) and (19), from a minimum

$$\left(S_{oh}^{ut}/S_{sh}^{ut}\right)_{min} = \left(K_{to}\psi_{\omega o}K_{vhof}\sqrt{h_{of}/h_{sf}}\right)/\left(K_{ts}\psi_{\omega s}K_{vhs}\right),\qquad(27)$$

to a maximal value

$$\left(\frac{S_{oh}^{ut}}{S_{sh}^{ut}}\right)_{max} = \frac{K_{to}\left(1-\kappa_{op\,max}\right)\psi_{\omega o}K_{vhof}\sqrt{\frac{h_{of}}{h_{sf}}} + \kappa_{op\,max}\psi_{\omega op}K_{vhop}\sqrt{\frac{h_{op}}{h_{sf}}}}{K_{ts}\psi_{\omega s}K_{vhs}}.\qquad(28)$$

The expression for the ratio of average values for the total power of the vertical components of impact pulses under a self-oscillatory and a steady mode, taking into consideration (21) and (23), takes the form

$$\frac{P_{oh}}{P_{sh}} = \frac{K_{to}\left(1-\kappa_{op}\right)\psi_{\omega o}K_{vhof}^{2}h_{of} + \kappa_{op}\psi_{\omega op}K_{vhop}^{2}h_{op}}{K_{ts}\psi_{\omega s}K_{vhs}^{2}h_{sf}}.\qquad(29)$$

Ratio (29) over a period of self-oscillations changes (P_{oh}/P_{sh}=var), taking into account (24) and (25), from a minimum

$$\left(P_{oh}/P_{sh}\right)_{min}=\left(K_{to}\psi_{\omega o}K_{vhof}^2 h_{of}\right)/\left(K_{ts}\psi_{\omega s}K_{vhs}^2 h_{sf}\right), \qquad (30)$$

to a maximal value

$$\left(\frac{P_{oh}}{P_{sh}}\right)_{max}=\frac{K_{to}\left(1-\kappa_{op\,max}\right)\psi_{\omega o}K_{vhof}^2\dfrac{h_{of}}{h_{sf}}+\kappa_{op\,max}\psi_{\omega op}K_{vhop}^2\dfrac{h_{op}}{h_{sf}}}{K_{ts}\psi_{\omega s}K_{vhs}^2}. \qquad (31)$$

Values of the variables κ_{op}, K_{to}, K_{ts}, K_{vhs}, K_{vhof}, h_{sf}, h_{of} and h_{op} in expressions (26)-(31) can be determined by visual analysis of the fill motion patterns. Values of variables $\psi_{\omega o}$ and $\psi_{\omega op}$ can be determined by visual analysis of transient self-oscillating modes of the fill motion. The value of $\psi_{\omega s}$ is specified.

Fig. 7 shows the obtained quasi-static dependences of the K_t turnover on the relative rotational velocity ψ_ω for a discrete charge with a relative particle size $d/(2R)$=0.01-0.03 for the values of the chamber filling degree κ=0.25, 0.35 and 0.45.

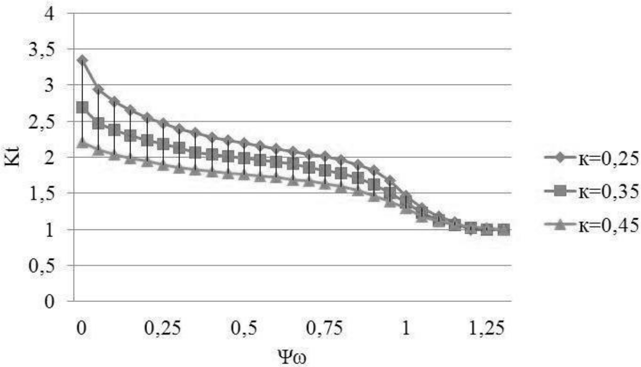

Fig. 7. Dependence of fill turnover K_t on relative rotational velocity ψ_ω at $d/(2R)$=0.01-0.03, κ=0.25, 0.35 and 0.45

The ratio of the collision parameters for self-oscillating and conventional steady-state operation modes (26) to (31) is convenient to use for comparative assessment of the impact of the grinding fill at different of the chamber filling degrees κ.

4. Results obtained in determining the impact action of the fill

According to the experimental data obtained, geometric and kinematic parameters of the fill motion at the chamber filling degree κ=0.45 had the following values: K_{vhs}=0.4, K_{vhof}=0.75, K_{vhop}=1,

h_{of}/h_{sf}=1.54, h_{op}/h_{sf}=1.46, $\psi_{\omega s}$=0.75, $\psi_{\omega o}$=1.1, $\psi_{\omega op}$=1.31 (at f_{op}≈2 Hz), K_{ts}=1.63 (at $\psi_{\omega s}$=0.75), K_{to}=1.11 (at $\psi_{\omega o}$=1.1), κ_{opmax}=0.0662.

Parameters of the fill motion at the fill κ=0.35 acquired the following values: K_{vhs}=0.3, K_{vhof}=0.75, K_{vhop}=1, h_{of}/h_{sf}=1.48, h_{op}/h_{sf}=1.6, $\psi_{\omega s}$=0.75, $\psi_{\omega o}$=1.075, $\psi_{\omega op}$=1.31 (at f_{op}≈2 Hz), K_{ts}=1.82 (at $\psi_{\omega s}$=0.75), K_{to}=1.19 (at $\psi_{\omega o}$=1.075), κ_{opmax}=0.12.

Parameters of the fill motion at the fill κ=0.25 acquired the following values: K_{vhs}=0.2, K_{vhof}=0, K_{vhop}=1, h_{of}/h_{sf}=0, h_{op}/h_{sf}=2.46, $\psi_{\omega s}$=0.75, $\psi_{\omega o}$=1.05, $\psi_{\omega op}$=1.31 (at f_{op}≈2 Hz), K_{ts}=2.01 (at $\psi_{\omega s}$=0.75), K_{to}=1.29 (at $\psi_{\omega o}$=1.05), κ_{opmax}=0.25.

Relation of average sums of vertical components of the collision momenta and the power of these momenta for self-oscillating and steady-state modes per unit time were determined. According to (27), (28), (30) and (31), extreme values of these relations at the fill κ=0.45 had the following values: $\left(S_{oh}^{ut}/S_{sh}^{ut}\right)_{min\,0.45}=2.32$, $\left(S_{oh}^{ut}/S_{sh}^{ut}\right)_{max\,0.45}=2.39$, $\left(P_{oh}/P_{sh}\right)_{min\,0,45}=5.41$, $\left(P_{oh}/P_{sh}\right)_{max\,0,45}=5.7$.

Extreme values of parameters of the collision momenta at the fill κ=0.35: $\left(S_{oh}^{ut}/S_{sh}^{ut}\right)_{min\,0.35}=2.96$, $\left(S_{oh}^{ut}/S_{sh}^{ut}\right)_{max\,0.35}=3.07$, $\left(P_{oh}/P_{sh}\right)_{min\,0,35}=9.35$, $\left(P_{oh}/P_{sh}\right)_{max\,0,35}=9.57$.

Extreme values of parameters at the fill κ=0.25: $\left(S_{oh}^{ut}/S_{sh}^{ut}\right)_{min\,0.25}=0$, $\left(S_{oh}^{ut}/S_{sh}^{ut}\right)_{max\,0.25}=5.79$, $\left(P_{oh}/P_{sh}\right)_{min\,0,25}=0$, $\left(P_{oh}/P_{sh}\right)_{max\,0,25}=45.5$.

The graphs of dependences S^{ut}/S_{sh}^{ut} for variable values of numerators S_{sh}^{ut} and S_{op}^{ut} for one period of self-oscillations at the degree of chamber filling with the fill κ=0.25, 0.35 and 0.45 are shown in Fig. 8.

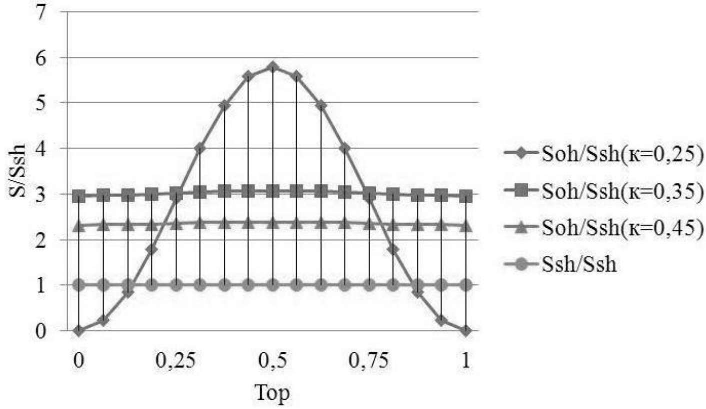

Fig. 8. Dependence of relations of average sums of vertical components of the collision momenta for self-oscillating and steady-state modes per unit time S_{oh}/S_{sh} and S_{sh}/S_{sh} on the period of self-oscillations T_{op} at $d/(2R)$=0.01-0.03, κ=0.25, 0.35 and 0.45

The graphs of dependences P/P_{sh} for the variable values of numerators P_{sh} and P_{oh} during the self-oscillation period at the chamber filling degree κ=0.25, 0.35 and 0.45 are shown in Fig. 9.

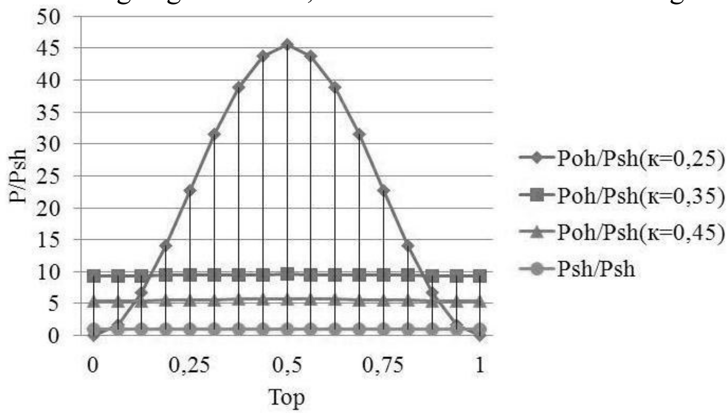

Fig. 9. Dependence of relations of average sums of powers of vertical components of the collision momenta for self-oscillating and steady-state modes per unit time P_{oh}/P_{sh} and P_{sh}/P_{sh} on the period of self-oscillations T_{op} at $d/(2R)$=0.01-0.03; κ=0.25, 0.35 and 0.45

Numerical values of dynamic parameters of the fill impact action indirectly characterize influence of the chamber filling degree κ on the grinding process course.

5. Experimental modeling of the grinding process

Effect of the degree of the chamber filling with the fill on efficiency of the self-oscillating grinding process in a tumbling mill was evaluated for the case of grinding cement clinker.

Particles of pre-crushed clinker with relative size $d_m/(2R)<0.0059$ completely filled in rest condition the gaps between steel ball grinding bodies with relative size $d_b/(2R)=0.026$. Productivity of the grinding process with duration of 30 min was determined by sieving through a 0.08 mm mesh sieve.

Technological efficiency of the self-oscillating grinding process was evaluated by relative productivity

$$\frac{C_o}{C_s}=\frac{1-m_{ro}/m_m}{1-m_{rs}/m_m}, \tag{32}$$

where C_o is productivity of the self-oscillating process; C_s is productivity of the conventional steady-state process; m_{ro} is weight of the sieve residue of the crushed material after sieving during the self-oscillating process; m_{rs} is weight of the sieve residue in the conventional steady-state process; m_m is total weight of a portion of ground material prior to sieving.

Power efficiency of the self-oscillating grinding process was evaluated by relative power intensity

$$\frac{P_{do}}{P_{ds}}=\frac{\psi_{P0,5d}}{\psi_{P0,5s}} \tag{33}$$

and relative specific power intensity

$$\frac{E_o}{E_s}=\frac{P_{do}}{P_{ds}}\bigg/\frac{C_o}{C_s}, \tag{34}$$

where P_{do} is the power of the drive rotating the filled drum in the self-oscillating process; P_{ds} is the power of the drive in the conventional steady-state process; $E_o=P_{do}/C_o$ is the specific power intensity of the self-oscillating process; $E_s=P_{ds}/C_s$ is the specific power intensity of the conventional steady-state process; $\psi_{P0,5o}=\psi_{M0,5o}\cdot\psi_{\omega o}$ is the relative power of the drive in the self-oscillating process (Fig. 10); $\psi_{P0,5s}=\psi_{M0,5s}\cdot\psi_{\omega s}$ is the relative power of the drive in the conventional steady-state process (Fig. 10); $\psi_{M0,5o}=M_0/M_{\max0,5}$ is the relative torque of the drive in the self-oscillating process (Fig. 11); $\psi_{M0,5s}=M_s/M_{\max0,5}$

is the relative torque of the drive in the conventional steady-state process (Fig. 11); M_0 is the absolute torque of the drive in the self-oscillating process; M_s is the absolute torque of the drive in the conventional steady-state process; $M_{\max 0,5}$ is the absolute value of the conditional maximum antitorque for the half-filled chamber (κ=0.5) which corresponds to the apparent load distribution over cross section of the drum chamber in a form of an ideal solid-state segment rotated at a right angle relative to the initial position; $\psi_{\omega o}$ is the relative velocity of the drum rotation during the self-oscillating process; $\psi_{\omega s}$ is the relative velocity of the drum rotation in the conventional steady-state process.

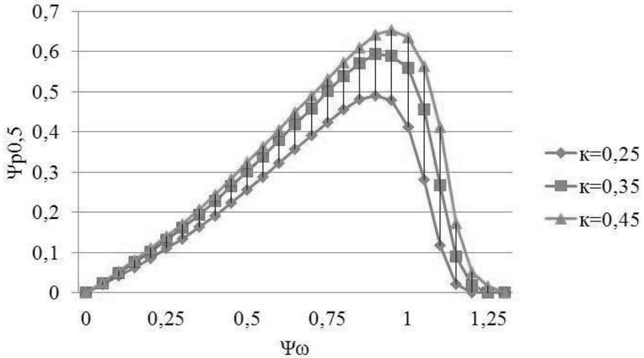

Fig. 10. Dependence of relative power of the drive in rotation of the filled drum, $\psi_{P0.5}$, on relative rotational velocity, ψ_{ω}, at $d/(2R)$=0.01-0.03, κ=0.25, 0.35 and 0.45

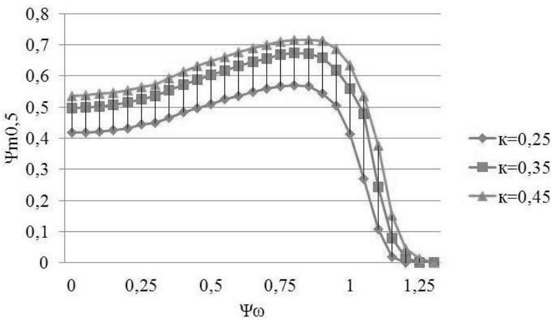

Fig. 11. Dependence of the relative drive torque in rotation of the filled drum, $\psi_{M0.5}$, on relative velocity of rotation, ψ_{ω}, at $d/(2R)$=0.01-0.03, κ=0.25, 0.35 and 0.45

According to the obtained experimental data, technological and power parameters of the process of grinding in a tumbling mill with the chamber filling degree κ=0.45 had the following values: C_s=0.435, C_o=0.464, P_{ds}=0.532 (at $\psi_{\omega s}$=0.75), P_{do}=0.413 (at $\psi_{\omega o}$=1.1).

Parameters of the grinding process at the chamber filling degree κ=0.35 acquired the following values: C_s=0.373, C_o=0.485, $P_{ds}=\psi_{P0,5s}$=0.501 (at $\psi_{\omega s}$=0.75), $P_{do}=\psi_{P0,5o}$=0.375 (at $\psi_{\omega o}$=1.075).

Parameters of the grinding process at the chamber filling degree κ=0.25 acquired the following values: C_s=0.323, C_0=0.472, $P_{ds}=\psi_{P0.5s}$=0.424 (at $\psi_{\omega s}$=0.75), $P_{do}=\psi_{P0.5o}$=0.282 (at $\psi_{\omega o}$=1.05).

Then, according to (32) and (34), relative productivity, power intensity and specific power intensity of the self-oscillating grinding process at the chamber filling degree κ=0.45 have the following values: C_0/C_s=1.067, P_{do}/P_{ds}=0.776, E_o/E_s=0.728.

Relative characteristics of the self-oscillatory process at the chamber filling degree κ=0,35 acquired the following values: C_0/C_s=1.3, P_{do}/P_{ds}=0.749, E_0/E_s=0.576.

Relative characteristics of the self-oscillatory process at the chamber filling degree κ=0,25 acquired the following values: C_0/C_s=1.46, P_{do}/P_{ds}=0.665, E_o/E_s=0.455.

Graphs of the obtained dependences of C_o/C_s, P_{do}/P_{ds} and E_o/E_s on the chamber filling degree κ are shown in Fig. 12.

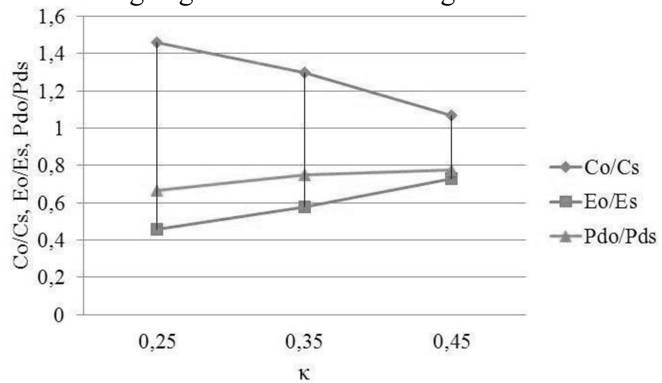

Fig. 12. Dependences of C_0/C_s, E_o/E_s and P_{do}/P_{ds} of the self-oscillating process of grinding cement clinker at $d_b/(2R)$=0.026 and $d_m/(2R)$<0.0059 on κ

Dependences of relative productivity, power intensity and specific power intensity characterize influence of the chamber filling degree on efficiency of the self-oscillating process of grinding.

Conclusions

1. Ratio of the average sums of the vertical components of the collision momenta per unit time for self-oscillating and steady-state motion modes was approximately determined numerically. For the relative size of the ball elements of the fill of 0.01-0.03, this ratio was approximately 2.32-2.39 at the chamber filling degree κ=0.45, 2.96-3.07 at κ=0.35 and 0-5.79 at κ=0.25. Ratio of the average sums of powers of such impacts per unit of time under such conditions was 5.41-5.7 at κ=0.45, 9.35-9.57 at κ=0.35 and 0-45.5 at κ=0.25.

2. With a decrease in a chamber fill, κ, the self-oscillating impact action increases significantly due to the decrease in the passive quasi-solid zone and a significant increase in the active pulsation zone of the fill motion in cross section of the rotating chamber. This is due to manifestation of the dynamic effect of significant increase in the self-oscillations swing with a decrease in κ.

3. A decrease in relative specific power intensity in grinding cement clinker was established experimentally for an innovative self-oscillating process compared to the conventional process. For the relative size of the fill particles of 0.026 and complete filling of the gaps between the grinding bodies, this reduction was approximately 27.2 % at κ=0.45, 42.4 % at κ=0.35 and 54.5 % at κ=0.25. Under these conditions, relative productivity of the self-oscillating process has increased by about 6.7 % at κ=0.45, by 30 % at κ=0.35 and by 46 % at κ=0.25.

4. It was established that with a decrease in the chamber fill, κ, specific power intensity and productivity of the self-oscillating grinding process carried out in the tumbling mill significantly decrease. This is due to manifestation of the technological effect of a significant increase in the impact action of the grinding bodies on particles of the crushed material and some decrease in the drive power spent for rotation of the filled drum with a decrease in κ.

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RAW MATERIALS ASPECTS OF NATIONAL SECURITY OF UKRAINE

Zhukov S.O.,
Doctor of Sciences (Engineering), professor,
Kryvyi Rih National University, Ukraine

Palyvoda O.A.,
Ph.D. (Engineering), associate professor,
Kryvyi Rih National University, Ukraine

Lutsenko S.O.,
Ph.D. (Engineering), associate professor,
Kryvyi Rih National University, Ukraine

Abstract. Improving the national security of Ukraine is possible by expanding raw materials base of mining enterprises by substantiating prospective quarries contours. When determining the quarry boundaries, economic stripping ratio is calculated according to technical and economic indicators achieved at the time of design and its value is constant. However, the analysis of mining and processing plants operation showed that their economic indicators and stripping ratios change over time. It is proved that economic stripping ratio is not constant, but varies over time. The analytical method substantiates the condition of competitiveness of the designed quarry. On the example of quarries, which reflect the characteristic features of the development of steeply dipping deposits in Ukraine, the influence of technological indicators (current stripping ratio) of the existing quarries operation on the value of the economic stripping ratio and, accordingly, on the quarry final depth is graphically substantiated.

The methodology was developed for determining the boundaries of quarries, which provides for the application of the economic stripping ratio as a value that is not constant, but varies over time, and one that depends on changes in the current stripping ratio at competing quarries. The result of the study is that the theory in the field of determining quarries final contours has been improved. It is proved that the deviation of the final mining depth of the designed quarry, determined by comparing its current stripping ratio with the current stripping ratios of the conditional base competing quarries, from the final mining depth determined by the constant economic stripping ratio can reach 50%.

Keywords: raw materials base, iron ore quarry, economic stripping ratio, quarry boundaries.

Introduction. One of the main parameters in the design of open pit mining is the boundaries of the quarry. The economic efficiency and duration of the enterprise operation, the full use of mineral reserves and investment volumes directly depend on the selected depth of open-pit mining.

As practice shows, the depth and position of the final contours of most major quarries, as they develop mineral deposits, are repeatedly reviewed and adjusted. However, it is mandatory to determine the final contours of quarrying, in which open pit mining will be effective.

Recent applied and scientific studies on this issue clearly indicate the need for reevaluation of the final contours of open pit mining in Ukraine. At the moment, the working boards of almost all quarries on the surface have approached or are approaching the project contours and are entering the reclamation process (in the process of transferring the working open pit side to the designed one). An example of such quarries can be powerful quarries of the Kryvyi Rih iron ore basin. The development of mining occurs only when it is lowered. Development within the boundaries approved by existing projects may lead to the rapid reclamation of the working open pit sides at the final design contours in the nearest future. If during this period we do not decide on the feasibility of expanding the design contours and allow the reclamation of the working sides, then in the future huge investments and a long time will be required to re-preserve the reclaimed sides. Therefore, it is now necessary to determine the possible prospects for the development of quarries and the feasibility of their reconstruction. In addition, this will entail a significant increase in the cost of open pit mining due to the re-conservation of non-working sides if the boundaries of the quarries are revised in the future.

In such conditions of mining, it is necessary to assess the capabilities of the raw materials base for the further development of the mining enterprise. Therefore, it is now necessary to determine the possible prospects for quarry development and the feasibility of revising and adjusting their boundaries. The further it is delayed in time, the greater will be the price and duration of the restoration of the design productivity of the raw ore quarries during the reconstruction of the quarries in case of further revision of the boundaries. In this regard, the interest of the owners of mining enterprises in the mining capabilities of the deposits transferred to them for exploitation increased: the final depth of the quarry, the remaining ore reserve in its contours, the maximum possible productivity of the quarry for ore, the volume of overburden operations, ensuring its achievement and others. These parameters determine the development prospect of the mining and processing plant and its industrial potential, the necessary investments and possible profits.

According to article 13 of the Constitution of Ukraine, natural resources are the subject of property rights of the Ukrainian people. Therefore, the state should control that the owners of the mining enterprises conduct the most complete development of the deposits that they have taken for use. From the above it follows that the problem of determining the prospective boundaries of quarries is relevant both for business entities that exploit mineral deposits and for the state and society overall.

Currently, quarries are being designed, in accordance with the decree of the Ministry of Industrial Policy of Ukraine; in accordance with the Technological design standards for mining enterprises with open pit method of mineral deposits development. This document does not actually reflect the latest scientific achievements in determining the boundaries of quarries, and does not fully take into account the current state of technology and technology of open pit mining.

Based on the above mentioned, increasing the national security of Ukraine is possible by expanding the raw materials base of mining enterprises through the development of a theoretical apparatus for quarry designing, as well as the development of specific applied methods and instructions for determining the boundaries of open pit mining, providing the most complete and cost-effective mining development of mineral deposits. Therefore, the purpose of the research described in the paper is to improve existing methods for determining the boundaries of quarries, focusing on the work of mining enterprises in a market economy.

Currently, in the theory and practice of designing, there is no single method for determining the boundaries of quarries developing steeply falling deposits. There are several well-known principles for substantiating the depth of quarries, based on a comparison of one of the stripping ratios or the sum of these ratios with the economic stripping ratio.

A major contribution to the development of the theory of designing the boundaries of open pit mining belongs to V.V. Rzhevsky [1], A.I. Arsentiev [2, 3], B.P. Yumatov [4], A.K. Polishchuk [5], V.S.

Khokhriakov [6] V.G. Blizniukov [7]. In these works, the determination of the economic stripping ratio was carried out based on comparing the performance indicators of the enterprise under study with the performance indicators of enterprises with an underground mining method, as well as with indicators of similar mining and processing plants. The price of marketable ore (products), which depends on its quality, has always determined the boundaries of open pit mining. However, when determining the boundaries of quarries according to the proposed calculation principles, the economic stripping ratio is calculated according to the technical and economic indicators achieved at the basic mining enterprises at the time of design, and its value is constant, while it does not take into account that the economic indicators and stripping ratios of the basic enterprises (competitive quarries) change over time.

Calculation principles. The economic basis of all the calculation principles for determining the boundaries of quarries is a comparison of the allowable cost price of ore mining (c_a) with the expected of the designed career ($c_{design.}$) [8], which is expressed by the inequality

$$c_a \geq c_{design.}, \text{ UAH/t.} \tag{1}$$

To determine the boundaries of the quarry, ensuring the competitiveness of iron ore products, it is necessary to take into account the time-varying economic indicators of the base enterprises (competing quarries).

Therefore, the calculation principle is based on the fact that at any development period, the economic indicators of the production and sale of iron ore products of the designed enterprise should be less than or equal to the similar indicators of the existing (base) enterprise.

Thus, if quarries develop deposits with the same quality of minerals, then the cost price of ore is taken as the cost price of ore of one of the main competing mining competitive enterprises with an open pit method of developing deposits

$$C_{act.} \geq C_{design.} \tag{2}$$

In this case, the allowable cost price of ore is accepted as it is at the time of designing a new quarry or reconstruction of an existing one.

The expected cost price of ore for the designed quarry is calculated depending on the value of the stripping ratio ($n_{design.}$) according to the formula, UAH/t

$$c_{design.} = a + b \cdot n_{design.}, \text{ UAH/t,} \tag{3}$$

where a – cost price of ore mining without stripping soils expenses, UAH/t; b – stripping expenses, UAH/m^3; $n_{design.}$ – stripping ratio in the designed quarry, m^3/t.

After substituting the expected value of the cost price of ore mining and transforming inequality (2), formula (3) will take the following form, m^3/t

$$\frac{C_{act.} - a}{b} \geq n_{design.}, \tag{4}$$

The left-hand side in expression (4) determines the value of the economic stripping ratio (n_e) and then, taking into account formulae (1, 3 and 4), we obtain the inequality, m^3/t

$$n_e \geq n_{design.}, \tag{5}$$

Based on the foregoing, the condition for the competitiveness of the designed quarry can be formulated as follows: *the stripping ratio for the designed quarry should not exceed the economic stripping ratio, which takes into account the economic indicators of the development of the existing (base) enterprise.*

Determining the economic stripping ratio. Methods for determining the economic stripping ratio are classified according to the methods for determining the allowable cost price of ore.

Technological design standards [9] for mining enterprises with open pit method of mineral deposits development regulate determining the economic stripping ratio based on the comparison of the underground and open pit methods of development.

However, it should be understood that a comparison of the cost prices of the open pit and underground mining is not correct due to different volumes of production; the productivity of quarries significantly exceeds the productivity of mines, which covers the needs of the market to different extents. In addition, the cost price of production by underground mining remains equally high throughout the life of the mine, while the quarry reaches a peak in production cost price only in one of the operating periods, when the surface boundaries reach the final contour. Therefore, it is advisable to determine the economic stripping ratio at the wholesale price per unit of commodity products, its output from raw ore, and the expenses on enriching a unit of raw ore.

The use of the wholesale price as a criterion for determining the economic stripping ratio is not reliable when planning for long time periods. Thus, for example, when designing a quarry for 20-50 years, it is not possible to determine market demand and selling prices for minerals. In addition, the economic indicators of existing (base) enterprises, at the time of the new design preparation, are used in the calculations of the economic stripping ratio. In this case, the economic stripping ratio will be a constant value for the designed quarry. However, the analysis of the economic indicators of most operating mining and processing plants, even for a relatively short period of their work, showed their significant change, Table 1.

Table 1

Economic indicators of the production and sale of the concentrate at the mining and processing plants of Ukraine

Enterprise	Years	Cost price, UAH/t	Price, UAH/t	Expenses on the 1 UAH of commercial products, UAH/UAH	The iron content in the concentrate, %
Northern Ore Mining and Processing Plant (Northern GOK)	2005	118,45	312,75	0,38	65,64
	2010	272,67	907,25	0,30	
	2013	343	858	0,40	
	2015	430	1150	0,37	
	2020	689,90	1850	0,37	
Central Ore Mining and Processing Plant (Central GOK)	2005	148,74	321,74	0,46	66,8
	2010	350,72	957,74	0,37	
	2013	397,18	900	0,44	
	2015	540	1190	0,45	
	2020	868,4	1920	0,44	
Southern Ore Mining and Processing Plant (Southern GOK)	2005	107,7	258,74	0,42	65,36
	2010	337,3	1012	0,33	
	2013	312,85	958	0,33	
	2015	409,44	1233	0,33	
	2020	622,08	1870	0,34	
Inguletsky Ore Mining and Processing Plant (Inguletsky GOK)	2005	120,36	287,11	0,42	63,89
	2010	327	921	0,36	
	2013	364,71	1000	0,37	
	2015	485	1210	0,40	
	2020	721,6	1780	0,39	

The table shows that from 2005 to 2020, the cost price and price of marketable mineral products increased on average respectively by 600 and 700%.

From this, we can conclude that the economic stripping ratio for the designed quarry will change.

In the regulatory documents governing the operation of mining enterprises with an open pit mining method, the calculated economic stripping ratio for determining the final mining depth of the designed quarry is assumed constant. However, the competitive base enterprise continues its work, and over time its economic indicators will change, in our case, the cost price of ore will change. The reason for this is the change in the current stripping ratios up or down.

Let us consider the condition for changes in the current stripping ratios over time using two conventional base competing quarries, which are presented at characteristic sections of the developed deposits (Fig. 1 and Fig. 2).

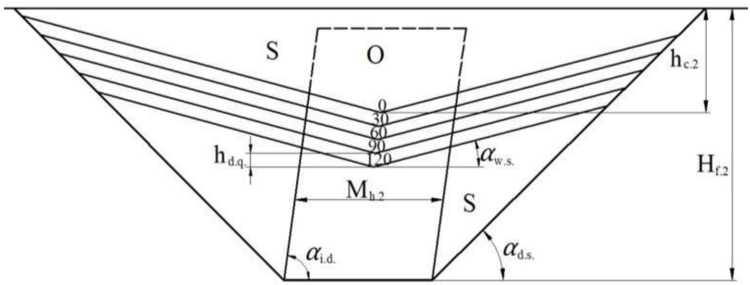

Fig. 1. Quarry №1: S – stripping soils; O – ore; $h_{c.1}$ – current depth of mining, m; $h_{d.q.}$ – step of dipping mining operations, m; $\alpha_{w.s.}$ – slope angle of the working side of the quarry, degrees; $\alpha_{d.s.}$ – slope angle of the designed side of the quarry, degrees; $\alpha_{i.d.}$ – angle of inclination of ore deposit, degrees; $M_{h.1}$ – horizontal deposit thickness, m; $H_{f.1}$ – final depth of the quarry, m; 0, 30, 60, 90, 120 – stages of lowering mining operations in the quarry

Conventional basic quarry №1 (Fig. 1) is developing an ore deposit in which: the angle of inclination is 80°; horizontal thickness – 350 m; length – 2700 m. The parameters of this quarry are slope angle of the working side – 16°; slope of the designed side – 45°; final quarry depth – 630 m. It should be noted that at the quarry №1, the surface boundaries have already approached the design position and further development of the open pit is carried out with the development of mining operations only in depth.

Conditional basic quarry №2 (Fig. 2) is developing an ore deposit in which: the angle of inclination is 80°; horizontal power – 120 m;

length – 4500 m. The parameters of this quarry are slope angle of the working side – 16°; slope of the designed side – 45°; final quarry depth – 345 m.

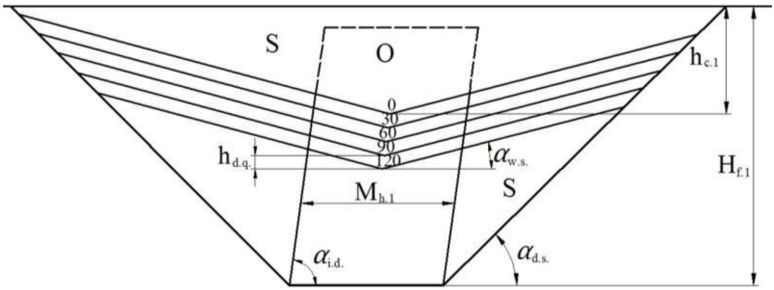

Fig. 2. Quarry №2: S – stripping soils; O – ore; $h_{c.2.}$ – current depth of mining operations, m; $h_{d.q.}$ – step of dipping mining operations, m; $\alpha_{w.s.}$ – slope angle of the working side of the quarry, degrees; $\alpha_{d.s.}$ – slope angle of the designed side of the quarry, degrees; $\alpha_{i.d.}$ – angle of inclination of ore deposit, degrees; $M_{h.2}$ – horizontal deposit thickness, m; $H_{f.2}$ – final depth of the quarry, m; 0, 30, 60, 90, 120 – stages of lowering the mining operations in the quarry

It should be noted that at conditional base quarry №2, the surface boundaries have not yet reached the designed position and further development of the quarry is carried out with the development of mining operations, both in the horizontal direction and in depth.

In the presented figures 1 and 2, the numbers 0, 30, 60, 90, 120 indicate the stages of calculating the volume of mining operations when they lower for every 30 m. The position of the working side of the quarry at stage “0” characterizes the state of mining operations in the quarry and serves as a starting point for further studies of their development.

To simplify the study, the position of the working sides of quarries №1 and №2 at the zero stage was chosen so that the value of the current stripping ratios of these quarries was the same.

When designing a new quarry, firstly it is necessary to determine the prospective depth of open pit mining: it will determine the boundaries of the quarry in terms of surface and the possible productivity of the quarry in terms of a mineral deposit. A condition for determining the prospective boundaries of the designed quarry is to obtain economic indicators of the designed quarry no worse than the economic indicators of existing quarries. This condition is written by inequality (18).

To determine the final depth of the quarry of the future (conditional designed) quarry, a characteristic section is selected on the deposit intended for operation in which: the angle of inclination is 85°; horizontal thickness – 230 m; length – 3200 m. The other parameters are taken similar to quarries №1 and №2. All quarries exploit the deposits of the same type of mineral with the same quality, and the existing technology allows them to work with the conditionally equal expenses on extracting one ton of stripping soils and extracting one ton of ore.

Both at conditional base quarries №1 and №2, and for a conditional designed quarry, from the position of working sides at the zero stage, their further position is rebuild according to the stages of development of mining operations every 30 m in depth. At each stage, the volumes of ore and stripping soils are measured with the determination of the current stripping ratios, the calculation results are listed in Table 2.

For the designed quarry and the base operating quarries, we construct a graph (Fig. 3) for the change in the largest current stripping ratio, depending on the increase in the designed mining depth of the quarry ($H_{des.q.}$). The line of the calculated stripping ratio is also shown in the graph.

Table 2

Volumes of the ore and stripping soils, as well as current stripping ratios according to study options

Quarries	Volumes	Quarry dipping, m				
		0	30	60	90	120
Conditional base quarry №1	Ore, m^3	28080000	28080000	28080000	28080000	28080000
	Stripping soils, m^3	101250000	95445000	89775000	84240000	78435000
	Stripping ratio, m^3/m^3	3,6	3,4	3,2	3,0	2,8
Conditional base quarry №2	Ore, m^3	16425000	16425000	16425000	16425000	16425000
	Stripping soils, m^3	59400000	80325000	88650000	92025000	90000000
	Stripping ratio, m^3/m^3	3,6	4,9	5,4	5,6	5,5
Conditional designed quarry	Ore, m^3	21920000	21920000	21920000	21920000	21920000
	Stripping soils, m^3	47680000	68800000	87520000	109760000	126880000
	Stripping ratio, m^3/m^3	2,2	3,1	4,0	5,0	5,8

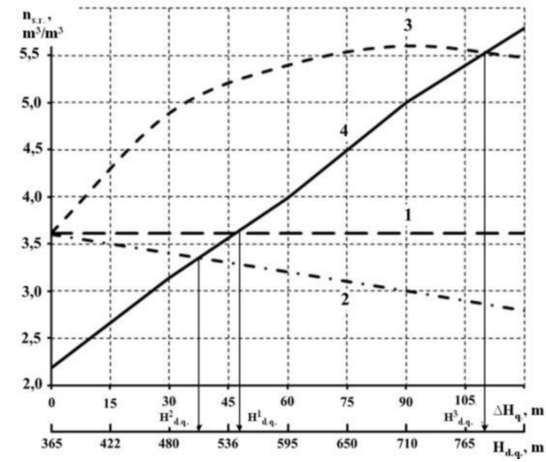

Fig. 3. A graph of the dependence of stripping ratios on lowering mining operations at the stages of dipping quarries with determining the final mining depth of a conditional designed quarry: 1 – economic stripping ratio; 2 – current stripping ratio in conditional base quarry №1; 3 – current stripping ratio in conditional base career №2; 4 – current stripping ratio according to the conditional designed quarry

In these cases, the current stripping ratios of the conditional base quarries №1 and №2 will also be economic stripping ratios for the conditional designed quarry.

If we calculate the economic stripping ratio for a newly designed quarry without taking into account the time-varying current stripping ratios of the base quarries №1 and №2, as accepted before the present studies, it will be constant, line 1 in Fig. 3.

The intersection of these lines (lines 1, 2, 3 in Fig. 3) with the line of the current stripping ratio at the conditionally designed quarry (line 4 in Fig. 3) will give a solution to the final depth of the designed quarry with various methods for determining the economic stripping ratio.

Thus, for the designed quarry, three values of the final mining depth were obtained:

- $H^1_{des.q.}$ – final mining depth of the quarry, determined by the economic stripping ratio without taking into account its changes in time, is 560 m;

- $H^2_{des.q.}$ – final mining depth of the quarry, determined by the time-varying current stripping ratio of the quarry №1 and being the economic stripping ratio for the designed quarry, is 485 m;

- $H^3_{des.q.}$ – final mining depth of the quarry, determined by the time-varying current stripping ratio of the conventional base quarry №2 and being the economic stripping ratio for the designed quarry, is 800 m

The graphical solution for determining the final depth of the conditional designed quarry is presented in Fig. 4.

Fig. 3 shows that the deviation of the final mining depth of the conditional designed quarry, determined by comparing its current stripping ratios with the current stripping ratios of the conditional base quarries, from the final mining depth determined by the economic stripping ratio (by the outdated method) is from 14 to 45%.

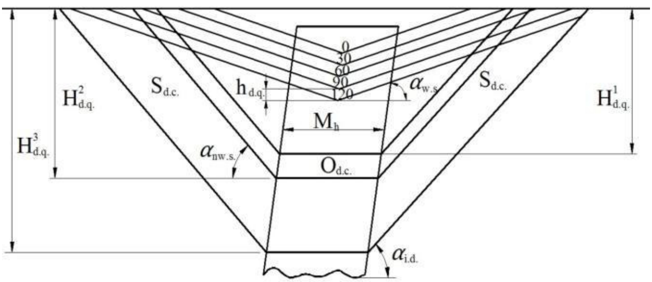

Fig. 4 Cross-section of the deposit, which will be developed by the conditional designed quarry: $S_{d.c.}$ - stripping soils in the designed contours of the quarry; $O_{d.c.}$ - ore in the designed contours of the quarry; $h_{d.q.}$ - step of dipping mining operations, m; $\alpha_{w.s.}$ - angle of the working side of the quarry; $\alpha_{nw.s.}$ - angle of the non-working side of the quarry; $\alpha_{i.d.}$ - angle of inclination of ore deposit; M_h - horizontal deposit thickness; 0, 30, 60, 90, 120 - stages of lowering mining operations in the quarry; $H^1_{d.q.}$ - final mining depth of the quarry, determined by the economic stripping ratio; $H^2_{d.q.}$ - final mining depth of the quarry, determined by the current stripping ratios of the conditional base quarry № 1; $H^3_{d.q.}$ - final mining depth of the quarry, determined by the current stripping ratios of the conventional base quarry № 2

Mainly two indicators - the cost price of marketable mineral products and its price, determine the competitiveness of a mining enterprise, like any other. The cost price of marketable mineral products, even of a similar type, depends on the natural conditions of the deposit, the existing structure of the enterprise, the achieved level of organization and production technology, and other factors that depend on the qualifications of the management. Therefore, the difference in the quality and cost price of marketable mineral products (concentrate as its first type) at various mining and processing plants is quite understandable and explainable. However, the price of marketable mineral products on the sales markets should depend only on the quality of the products.

Previously, when there was a planned economy and economic indicators were stable, the price of marketable mineral products was determined solely on the content of the useful component in it. Currently, the price of marketable mineral products depends not only on objective factors (environmental conditions, production technology, etc.); subjective factors also influence it a lot: the skills and competence of the management team, agreements between the buyer and seller of products, the use of methods to minimize tax obligations, and others.

If the price of the concentrate depends only on its objective factors (qualitative characteristics), then the price of 1% iron in the concentrate would be relatively the same at all plants. Therefore, a methodology was proposed [10] for determining the economic stripping ratio, which avoids the influence of unnatural and non-technological factors on the price of the concentrate. The bottom line is that the price of the concentrate of the designed mining and processing plant is provided at the price of the percentage of iron per tonne of concentrate of the base mining and processing plant (competitive enterprise),UAH/t

$$P'_{design.} = \frac{P_{bas.} \cdot Fe_{design.}}{Fe_{bas.}}, \tag{6}$$

where $P'_{design.}$ - provided price of the concentrate of the designed mining and processing plant;

$Fe_{bas.}$ and $Fe_{design.}$ - the content of iron in the concentrate of the base designed mining and processing plant, %.

Table 1 shows that the expenses of 1 UAH of marketable mineral products have a slight change - by 2-3%. Therefore, if quarry contours are designed under the condition that the economic indicators of concentrate production at the designed mining and processing plant should be no worse than the similar indicators of the compared plants, then it is necessary for the calculations to take (as the base enterprise) the indicators of the mining and processing plants with the smallest expenses per 1 UAH of produced and sold concentrate

$$E_{bas.} \geq E_{design.} \tag{7}$$

where $E_{bas.}$ and $E_{design.}$ - expenses per 1 UAH of produced and sold concentrate at the base and designed mining and processing plants, UAH/UAH.

Then,

$$\frac{C_{bas.}}{P_{bas.}} \geq \frac{C_{design.}}{P_{design.}} \tag{8}$$

where $C_{bas.}$ and $C_{design.}$ – the cost price of the concentrate production at the base and designed mining and processing plants, UAH/t; $P_{bas.}$ and $P_{design.}$ - the price of the sale concentrate at the base and designed mining and processing plants relatively, UAH/t.

From here, the economic stripping ratio for the quarry of the designed mining and processing plant, determined through comparing the economic efficiency of production and sale of concentrate at the basic mining and processing plant, will be equal to, m^3/t

$$n_e = \frac{E_{bas.} \cdot P'_{design.} \cdot \gamma_q - (a_{Da} + a_{Dp})}{в_D}, \tag{9}$$

where $E_{bas.}$ - expenses per 1 UAH of the marketable mineral products of the base enterprise, UAH.

Based on the results of the performed studies, we can confidently state the following. In order to determine the boundaries of the designed quarry, the economic stripping ratio must be determined taking into account the possible changes in the extraction volumes of stripping soils and ore at the base competitive enterprises which perform mining operations both in an open pit and underground method, i.e. taking into account changes in their current stripping ratios.

It should be noted that ignoring changes in the economic stripping ratio over time can lead to both an excess and an underestimation of the economically feasible depth of the designed and existing quarries. The error in the accuracy of determining the boundaries of quarries that do not take into account market conditions reaches 50%.

The conducted studies made it possible to justify the new calculation principle for determining the boundaries of quarries. It can be formulated as follows: the final depth of the quarry should be such that during its operation the sum of the initial and maximum current stripping ratio does not exceed the time varying appropriate economic stripping ratio, which depends on the technical and economic indicators of the base enterprises operation. At the same time, the current stripping ratio reaches its maximum value at the time of mining operations on the level of the daily surface at the final design contours.

The algorithm for determining the quarry boundaries according to this calculation principle is as follows:

1. On a characteristic transverse geological section or overall plan for mining operations, several options for quarry contours along the surface are outlined and the corresponding values for the final quarry depth are determined in accordance with the accepted slope angles of the non-working sides.

2. For each planned option, the position of mining operations is rebuilt at the moment of their exit by the level of the day surface to the design contours of the quarry and the values of the largest current stripping ratios are determined.

3. Based on the calculation results, is built up a graph (Fig. 5) of changes in the largest current stripping ratios depending on changes in the depth of mining operations for the same time period $n_{\text{т}max}=f(h_{\text{г.р.}})$. To determine the final depth of the quarry under the abscissa axis, we construct an additional scale for changing the final depth of the quarry $n_{\text{т}max}=f(H_{\text{к}})$. While determining the boundaries of quarries developing elongated deposits, when the design is carried out at several transverse sections for different areas of the deposit, we construct graphs of changes in the largest current stripping ratios depending on changes in the depth of mining operations for each area $n_{\text{т}max}=\mathrm{f}(\mathrm{h}_{\text{г.р.}})$ and from an increase in the final depth of the quarry $n_{\text{т}max}=f(H_{\text{к}})$ in the same areas.

4. The values of the economic stripping ratio are calculated taking into account the time varying technical and economic indicators of the base enterprise, as well as the reduction of prices to one type of marketable mineral product at the price of one percent of iron per ton of concentrate.

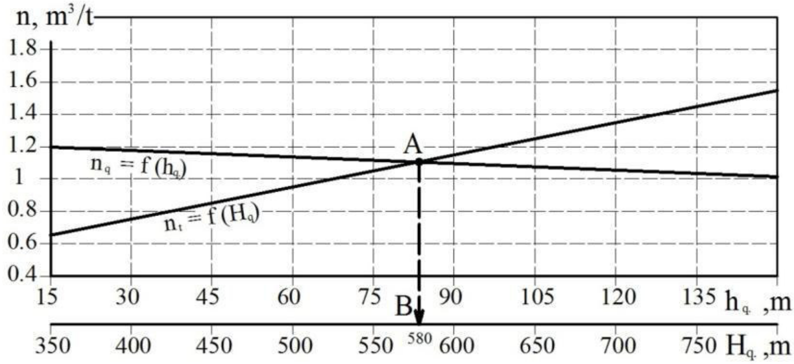

Fig. 5. Graphs of changes in the largest current tripping ratios depending on the increase in the depth of mining operations of the designed quarry from the current state

5. The intersection point of the curve reflecting the change in the largest current stripping ratio of the designed quarry with the lines of the economic stripping boundary (point A in Fig. 5) is determined and the prospective quarry depth is determined in comparison with the base enterprise (point B in Fig. 5).

Conclusions. A new calculation principle has been developed for determining the boundaries of quarries, which provides for the application of the economic stripping ratio as a value that is not constant, but varies over time, and one that depends on changes in the current stripping ratios at competing quarries.

The economic basis for the calculation principle for determining the prospective depth of a designed quarry is the following condition: at any period of development, the economic indicators of the production and sale of iron ore products of the designed enterprise should be better than or equal to those of the existing (base enterprise for comparison) enterprise.

The error in the accuracy of determining the boundaries of quarries that do not take into account market conditions reaches 50%. This calculation principle allows you to expand the raw materials base of mining enterprises and thereby increase the national security of Ukraine.

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SUBSTANTIATION OF INVESTMENT IN STUDY AND INDUSTRIAL DEVELOPMENT OF THE "ZHYRYCHI" COPPER ORE OCCURRENCE IN VOLYN

Melnychuk V.G.
National University of Water and Environmental Engineering (NUWEE), Professor, Doctor of Geological Sciences, the Head of the Department of Geology and Hydrology, Ukraine

Polishchuk A.M.
Institute of Geological Sciences, NAS of Ukraine, Candidate of Science Degree of the Department of Geotectonics and general geology, Ukraine

Melnychuk G.V.
National University of Water and Environmental Engineering (NUWEE), Candidate of Geological Sciences, Associate Professor of the Department of Geology and Hydrology, Ukraine

Krynickaya M.V.
National University of Water and Environmental Engineering (NUWEE), Candidate of Geological Sciences, Associate Professor of the Department of Geology and Hydrology, Ukraine

Abstract

The stratiform deposits of native copper were found among the trap complexes of the lower Wend In the cover of the Volyno-Podilskiy Plate of the Volyn territory in the southwestern part of the East European Platform. They corresponding to the largest mineralogical provinces of the World by resources.

In the article economical efficiency of the copper deposits geological study and industrial development of the Volynian Vendian traprocks are substantiated by example of the "Zhyrychi" ore occurrence.

The ore occurrence's general characteristics, information about copper-bearing horizons, ores and ore bodies and project data of its prospecting are given. Technical and economical indicators of its industrial development were identified. An approach to the complex processing of copper basalt and basalt tuffs, which involves the extraction from them, in addition to copper, and precious metals, as well as the use of enrichment tails as a peturgical raw material.

The current market value of metallurgical copper averages $ 5,000 per ton.

Considering that, possible income from development of the copper deposits may amount several billion $ from each ones.

The conclusion is that the biggest income will be obtained if 0.2% cut-off grade ores be used, and the highest profitability – at 0.4% cut-off grade.

The obtained results may become a basis for further geological-economical researches of the other numerous ore occurrences in the Volynian traprocks. In total, they delineate a large copper-bearing province of European significance.

Key words: Volyn, traprocks, native copper, ore, prospecting, costs, income, profitability.

Introduction

The scale of modern need for mineral resources, up-to-date technologies of its extraction from ores, demands of expensive infrastructure creation lead to increase of industrial development deposits with significant reserves that belong to the category of large and super large ones.

The greatest prospects of those deposits detection have territories that buried under the platform cover to 1000-1500 meters depth.

The Volyno-Podilska Plate of the Southwestern part Eastern European platform (figure 1) is one of those areas.

Its cover contains large stratiform native copper deposits among Lower-Vend trappean complexes [8,10]. Those deposits belong to the largest mineragenetic provinces of the world. "Zhyrychi" ore occurrence is one of them.

Large native copper deposits are known on the Keweenaw Peninsula since the century before last (Michigan, USA) [3,4].

Those deposits gave over 5 million tons of copper and 500 tons of silver for 120 years of their development.

Discoveries of new large deposits suitable for developing in British Columbia [12] and Yunnan and Guizhou provinces in the Southern China [13] give reason to have a new look at the prospects of native copper mineralization in platform covers, in the Volynian traprocks in particular, where deposits of copper located that discovered by Polish geologists [9] in the last century.

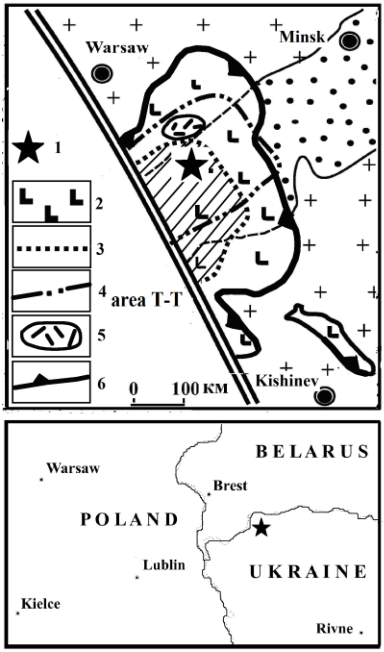

Fig. 1. Position of the "Zhyrychi" copper ore occurrence among the Lower-Vendian trappean complexes of the Southwestern part Eastern European platform:
1 - "Zhyrychi" copper ore occurrence; *2* - Lower Vendian traprocks; *3-6* - dissemination contours of Lower-Vendian trappean complexes: *3* - Zakhidnobuzkiy, *4* - Verkhnopripyatskiy, *5* - Brestskiy, *6* - Bilovezko-Podilskiy

A large volume and lateral spread of ore-containing rocks, a presence of several volcanic stratiform ore levels with an industrial significant copper content, a native character of copper mineralization, an accompanying noble-metal mineralization, an ability of ore-containing rocks complex usage, a satisfactory ecological safety of native copper ore processing evidence of Volyn copper-ore region great prospects [9]. The copper deposits are not inferior to known foreign analogues for those indicators and might surpass all known ones in the Europe.

Recent times 12 prospective ore fields were defined by prospecting and evaluation works [10]. Their resources (mostly Inferred Resources) estimated in 16 billion tons of copper. The copper reserve of Pivdennorafalivskiy ore occurrence was estimated by the category C_2 (Probable Reserves) and resources of the Zhyrytskiy ore occurrence – by the category P_1 and P_2 (Inferred Resources).The scientific ground of technology of the complex processing of raw material that contains metals at development of basaltic deposits is executed [6].

Significant capital costs in geological study and industrial development of the Volyn copper deposits foresaw by preliminary technical and economical considerations (M.I. Zhykov and others, 2008). At current market price of metallurgical copper approximately 5,000 USD per ton, the possible income form potential deposits exploitation might consist of several billion USD in each of them.

The problem is the efficiency of multi-million investments in further geological study and industrial development of the copper deposits in the Volyn at the current market conditions. The advisability of such investments needs economical substantiation that made in this work by the example of the "Zhyrychi" ore occurrence considering current approaches [5].

General Characteristics of the ore occurrence

The "Zhyrychi" ore occurrence is located in the Ratne Region of the Volynska Oblast between Zhyrychi and Tur villages. It has 40 square kilometers area. Its surface has a plain relief with 155-162 meters altitude. Near 55% of the area belongs to the drained agricultural land and settlements infrastructure. The rest is forestry. There are a drainage canal and a gravel road, which passes through the area. There are gas pipelines of large (800 and 1200 millimeters) diameter along the road. As power-lines are LEP-10kW and LEP-35kW. The fresh water supply is provided by wells and artesian down holes. Sand and wood use as local building materials.

The "Zhyrychi" ore occurrence was studied by prospecting works (M. I. Zhuykov and others, 2008). In the geological structure of the region (figure 2) takes part eroded to various degree Lower-Vendian volcanogenic formation of the Volynska Series: Zabolotivska, Babynska, Ratnivska Suites and their subdivisions, that nave been studying by Polish [1,2] and Ukrainian [8,10] geologists. They are covered Upper-Vendian terrigenous sediments of the Mogyliv-Podilska and Kanylivska Series in the western part ore and the marl-chalky stratums of the Upper Cretaceous (to 40 meters) in the places of their erosion. There are quaternary sediments above them.

The Lagozhanskiy sublatitudinal fault is a limit of the ore occurrence in the North and Northwest. The possible feather-out line of the middle and lower Babynska Series rock complex is in the West. The Pivdennoratnivskiy fault is in the South.

At the limit of the ore occurrence the volcanic rocks strata has mainly a gently dipping seam (to 3^{o}) to the West-Southwest and the presence of a multidirectional mainly steeply dipping to vertical disjunctive dislocations with the displacement amplitude to 20-30 meters, to 80-100 meters rarely. Basalts and tuffs represent the Volynska

Series of volcanogenic rocks, which have mineralized copper intervals.

Different structures of Volynska Series were mapped by their age and facial affiliation due to block structure, conditioned mainly by Hercynian tectogenesis and Pre-Mesozoic denudation on the same surface.

Counted by actual basis geological maps of the area appropriate units of the Volynska Series (Ratnivska, Babynska and Zabolotivska Suites) amount 29.8, 35.2; i 36.4 square km accordingly.

Ratnivska Suite is represented in its ore-bearing part by basalts and lava breccias. The lasts are separated by individual flows. This separation is due to the low thickness tuff layers, within tuff conglomerates, tuff gravestones with exotic petrochemical and petrogparhical rocks variations among ones with the largest thickness. It causes possibility of identification the three different parts by age of the Suite. That is (from bottom to up by cross-section): Luchichivski, Zoryanski (transitive) and Yakushivski Layers. Their thickness stay more or less constant within the area.

Basalt pyroclastic rocks with basalts subordinate role present the Babynska Suite. The total thickness is up to 180 m. 40% of it is effusive basalt surface. These basalts lay approximately in the middle of the tuffaceous rock mass. However, in the overwhelming majority of cases the “under basalt tuffs” thickness slightly exceed (10-15%) the “on basalt tuffs” one.

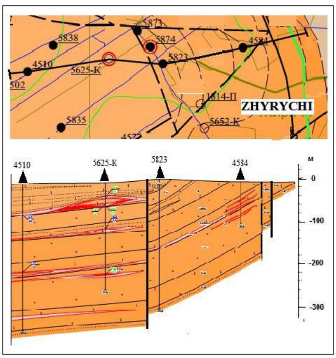

Fig. 2. Geological structure the of "Zhyrychi" copper ore occurrence (ore bodies in the section are shown in red)

Zabolotivska Suite in lateral relation is most distributed stratigraphic unit in the Volynska Series composition within the Zhyrychi area. Its thickness practically completely consists of basalts with amygdaloidal type with total absence of breccias which separating individual basalt flows. Comparably deep rocks bedding of the Suite cause weaker knowledge of its copper-bearing.

Since native copper mineralization occurrences in the underlying (Gorbashivska Suite) and overlying (Chartoryiska Suite) volcanogenic-terrigenous deposits are not discovered, the volcanogenic rocks of the Volynska Series limit the age range.

Copper-bearing horizons, ores and ore bodies

There are 5 ore-bearing horizons defined on the "Zhyrychi" site. In its turn, particular ore bodies with 0.2% content on ≥1.0 meter interval are defined in those horizons.

The productive horizons (from top to bottom): 1A (Zabolitvska Suite), 2A, 2B (Babynska Suite) and 3A (Luchichivska Suite), 3B (Ratnivska Suite) [8,11]. At the same time, only 2A horizon is presented by tuffs, and the others – by basalts. The occurrence depth of the ore horizons is between 164 and 530 meters. The dipping seam of the ore bodies is subhorizontal. There are possible minor changes of inclination angles in tectonic fragmentation zones.

The basalts of the horizons 1A and 2B often have amygdaloidal texture except their low-crystalline, aphanite types, which distinguish them in volume density from Ratne generally cryptocrystalline types.

Mineralogical composition is feldspar (40-55 %), monoclinic pyroxene (25-45%), magnetite (5-6%), basalt glass (up to 10%) and chlorite (up to 5%). In small amount, it contains ilmenite, hematite, zircon, staurolite, garnet. Of the secondary elements are present zeolite, palagonite, carbonates.

The tuffs are presented by different clastic (from pelitic to agglomerate) types with fragments of basalt, volcanic glass, which are cemented with altered (chloritization, zeolitization, ferrum oxides) pyroclastic material. The terrigenous impurity is less than 1.5-2%. Pyroclasts amount 80-85%, cement mass - 40% from the rock mass.

The lava-breccia part, which separate define basalt flows, is insignificant. They present only in horizons 3A and 3B. In general, ore-bearing rocks are monolithic. Although, differently oriented fracturing (sometime cured with secondary minerals) is detected on some near-fault areas.

The area of the ore horizons distribution is a few tens of meters. The length of particular ore bodies reaches hundreds of meters by both strike and dip. The ore bodies have generally strata shape. The copper-containing interval thickness ranges from several decimeters to 1-1.5 meters, 19.3 meters in particular cases (down hole 5827). The highest average weighted copper content is 2.01% per 1.1 meter in some places. The maximal content is 4.15% per 0.3 meter (5815 down hole) - 2B horizon.

By the technical and economical considerations for the "Zhyrychi" ore occurrence have been accepted the following temporary conditions: 0.2% cut-off grade copper content in a sample; 0.172% minimum mining copper content in the estimation block; for these conditions the ore bodies have been delineated by 0.2%, 0.3% and 0.4% cut-off grades. The copper resources belong to a medium deposit (over 0.5 million tons) with 0.379%; 0.457% and 0.596% average contents and with 1.26-5.7; 1.6-4.5 and 1.0-3.75 meters ore body thickness accordingly.

Under the division of copper by rock types [8] the basalt has 63%, and tuff has 37% from general amount (n=113) of the productive intervals (with over 0.2% copper content). The division of copper by concentration conditions (morphotypes) looks as follows: blotches in main rock mass - 55%, streaks and faces of fissures - 20%, amygdales - 10%, combinations of above types - 15% (figure 3).

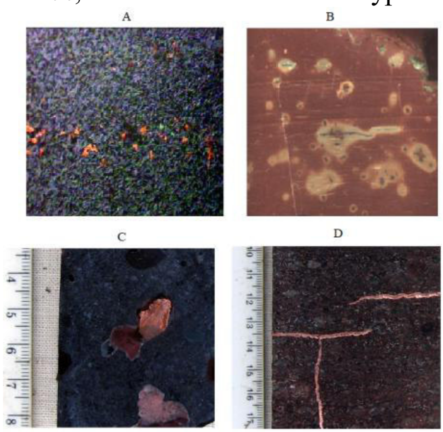

Fig. 3. Native copper mineralization morphotypes: A – banded blotch in phaneritic basalt (5-time zoom), B − nest-interspersed in spots of illumination of siltstones (increase in 6 times; C − basalt's amygdales, D – streaks among tuffs

The practical significance of copper mineral is native copper. Its ore formation and geological and industrial deposit type are identified [11]. Other copper minerals in particular chalcocite, digenit, chalcopyrite, bornite, covellite, cuprite, and tenorite present as accessory minerals and make no influence on the mineralization scale.

According to the microprobe analysis, (the total number of measures is 48) native copper has a high purity. The copper contents vary from 99.23 to 99.95%. The main impurity elements are Ag (0.01-0.37%) and Fe (0.01-1.04%).

A quite stable geochemical association of copper and silver was identified. The correlation coefficient between their contents is usually over +0.75. Silver is often detected in a native mineral form and characterized by high purity [3,5]. Based on the *X*-ray (microprobe) analysis data the average copper content in the native copper monofractions (n=51) is 706 grams per ton [10].

In the last years the ore occurrences prospects in relation to other noble elements - gold [9] and platinum group were confirmed.

Presence of native gold is revealed during developing of sample preparation scheme by IMR specialists (Simferopol) and in the crushed samples of tuffs (dawn hole 4382) and in the volcanic-sedimentary rocks, which superpose the traprock mass. The fineness of gold is within from 816 to 987. From the impurities are detected Ag (up to 10%), Cu (up to 15%), Fe (up to 0.4%), and Pt (up to 0.77%). They are presented as small spheres, unideal isomorphous crystals and films on native copper.

Except the native gold in the traprocks of the considering area during polished section studying detect such gold-containing minerals as cuproaurite (Cu_3Au), electrum (Au, Ag), rozhkovite (Cu, Au, Pd) and gold-containing pyrite (30 grams per ton).

Presence of gold (as chemical element) detected with all used analytic methods. Clear trends toward positive correlative geochemical relations with copper and silver are remarked. Gold content by the present microprobe analysis in native copper monofractions (n=49), extracted from the ore-bearing rocks of the occurrence, is 8.2 grams per ton in average.

The platinum group elements in relation to traprocks of the considering area and especially their copper-bearing are studied much weaker in comparison to before-mentioned noble metals. That is explained by underdeveloped analytic base for mass and express analysis in the volcanic intervals and very low threshold of sensitivity most of the used laboratory methods.

Platinoid concentration in mineral form have not detected already. Perhaps some volume of native iron, that is often present as accessory mineral, might be tetraferroplatinum as a result of more detailed research.

Presence of platinum, palladium, rhodium and iridium is revealed by the results of microprobe analysis and ICP-MS method of copper-containing rocks studying. The first two elements are detected most often. Their contains for Zhyrychi and Rafalivka ore occurrences are 0.03 and 0.01 grams per ton for Pt and Pd accordingly (by the results of assay tests, n=42).

There total 53 down holes have been drilled on the ore occurrence territory (23850 linear meters). Reached network density is 1600×600×800 meters with concentration on some profiles to 800×800 meters. The evaluation and direct ore resources estimation for the "Zhyrychi" ore occurrence made in 2008 (M. I. Zhuykov and others, 2008). The prospective copper resources for the "Zhyrychi" ore occurrence counted by the P_1 and P_2 (Inferred Resources) categories. They approved in UkrRNRP and accepted in author's count. The potential copper deposit has prospects for increasing because it borders with the "Shmenky-Zalisy" ore occurrence and the Pivnichno-Girnytske copper-bearing field.

Its geometrization made on the grounds of structural and tectonic position in relation to Girnytskyi block. From the North and North-west it is limited with the Lagozhanskyi sublatitudinal fault, from the west – with the feathering-out line ore-containing horizon 1A (Zabolotivska Suite), from the east - with feathering-line of the horizons 2A and 2B that coincide with ore-containing of the volcanic rocks mass (over 500 m thickness) submerge, from the South – the faults that belong to the Zabolottya-Bugska and Pivdenno-Ratnivska fault node.

The total area of the ore occurrence amount 36.4 square km that meets the area of horizon 1A distribution.

Considering all available geological-prospecting data, it was advisable, that the reserve estimation of copper and associated noble metals within Zhyrychi ore occurrence was calculated by three variant. It made by accepting as basis three gradation of cut-off grade, weight-average copper contain on the ore intervals thickness less than 1 m that is 0.2, 0.3, and 0.4%.

The resources estimation made with direct calculation by the following formula

$$P = \frac{S * m * c * d * k}{100},$$

where

P - copper resources (thousand tons);

S - ore body distribution square (square meters);

m - average thickness (m);

C - weight-average copper contain in ore body (%);

d - ore-containing rocks volume density (tons per cubic meter);

k - reliability (veracity) coefficient.

The ore-containing squares mapping which used during copper resources estimation made with considering of the dawn holes data and their position in definite blocks (structures) of the territory. In addition, geological-prospecting features (mineralization character, its genetic features, individual aspects of the volcanic rock mass and its separate parts, syn-ore fault confinedness, possible volcanic processes developing etc.) was used.

The ore bodies thickness accepted by the results of the drilling. The copper contains was calculated with considering the achieved analytical data as results of chemical drill core sampling.

Project data of ore occurrence prospecting

Intended purpose of the ore occurrence further geological study works consists of the prospecting evaluation completion and its exploration with preliminary geological and economical assessment and copper balance reserves estimation by C_1 and C_2 categories (Probable Reserves) to the depth of 500-600 meters with delineation of mineralization by the copper content from 0.1% and its industrial parameters determination concerning the developed conditions. The geological prospecting project realization will be committed in a staged manner.

At the preliminary phase of Stage 1 (prospecting evaluation), the works will be concentrated in the limits of the site of 1 square kilometer area. There the drilling network density will reach from 400x400 meters to 200x200 meters.

At the final phase of Stage 1 the accomplishment of drilling works foreseen according to the planned networks of 400×400 meters in the ore occurrence border and 200x200 meters (16 square kilometers) on the most prospective part. That will give an opportunity to complete the prospecting-evaluation works with resources evaluation by P_1 category (Inferred Resources) and reserves estimation by C_2 category (Probable Reserves) accordingly.

The drilling will be accompanied by the assaying complexes, geophysical researches (which have been determinated at first phase work as rational ones), analytical, laboratory and technological researches, mineralization features modeling with Mining GIS "MICROMINE".

During Stage 2 (exploration) the down holes drilling by 100x100 meters network (reserves of C_1 category, Probable Reserves) determined on the area of approximately 6 square kilometers. The evaluation will be by P_1 category after Stage 1 (Inferred Resources, 16 square kilometers area), C_2 (Probable Reserves) category (10 square kilometers) and C_1 (Proved Reserves) category in the limit of researched square (1 square kilometers).

The cost of the exploration works is 28.84 million USD as on November 2019. The approximate cost of Stage 1 (prospecting evaluation) is 15-16 million USD, Stage 2 (the exploration) - 13-14 USD.

The calculated (indicative) cost of a exploring mine for research and industrial extraction is 5.2-6.4 million USD (with mine depth of 450 meters and 2 horizontal working galleries of 1 kilometer each on 2 horizons).

Technical and commercial indicators of the ore occurrence industrial development

Geological and commercial indicators were calculated based on technical and commercial data for the "Zhyrychi" ore occurrence (M.I. Zhuykov and others, 2008) according to currency rate as of November 2019 (25 UAH per 1 USD) and the market prices of designed mining camp's commodity products considering modern methods [5].

The depleted deposit will be conducted by complex mining and beneficiation plant (MBP). The copper concentrate will be presented as a commodity product with 80.0% copper content. It will be exposed metallurgical procession for copper and noble metals (gold, silver, platinum and palladium) extraction. At the MBP it is possible to receive basalt raw materials that applicable for stone casting (the beneficiation tailings meet the requirements of the TU-14-12-190-02. Stone-casted chutes). The deposit development anticipated in underground way.

Ore beneficiation technology. The main ore beneficiation operations are crushing (to the limit piece of 0.04 millimeters), screening, hydraulic classification, gravity beneficiation, radiometric, electrical and magnetic separation. As a result of technology, probe beneficiation (0.313% mass fraction) was received the copper-containing concentrate with mass fraction of 81.18% and with initial ore extraction of 80.4% and the titanium-magnetite product with weight particles: Fe_2O_3-26.77%, FeO-26.1% and TiO_2-14.25% that is possible to use as ferriferous rare materials.

Using samples demonstrated by basalts (mass fraction of 0.14%) and their tuffs (mass fraction 0.46%) the beneficiation scheme was designed based on gravity (with receiving the coarse copper concentrate of 0.044 millimeters) and flotation (with receiving the fine copper concentrate) beneficiation cycles.

Expected capital costs of the industrial construction. In figure 4 the capital costs of the ore occurrence industrial development is given according to the basic (the first) evaluation option.

The mining camp construction investments calculated by analogy to the Bachtynske fluorite deposit.

The mining equipment cost calculated by analogy to the Saulyak gold deposit, the Bachtynske deposit, the Manuylivskiy iron ore deposit with appropriate amendment to the underground mines productivity and different time of the evaluation.

The capital investments cost in the beneficiation plant construction calculated by analogy to the Saulyak (the plant productivity is 200 000 tons of ore per year) and to the acting plant of the Muzhiivske mining and processing enterprise.

The capital investments cost in the Pobuzkiy nickel plant reconstruction for metallurgical processing of Volyn deposits is 23.1 million USD.

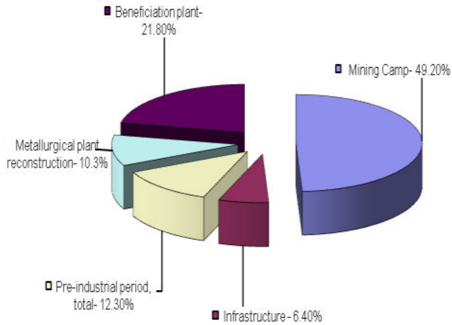

Fig. 4. The capital costs of the "Zhyrychi" ore occurrence industrial development

The possible operating cost of commodity production receiving calculated as the sum of cost mineral resource extraction and transportation from mine to beneficiation plant, its beneficiation, ore concentrate transportation to metallurgical plant where the final product is received (metallurgical copper and pure noble metals and alloys). The volumes of operating costs for commodity production receiving (commodity copper and gold, silver, platinum and palladium) are shown in the table and their structure is demonstrated in figure 5.

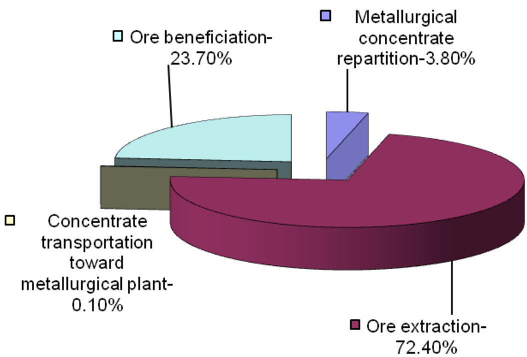

Fig. 5. The operation costs structure for commodity production receiving

The economic efficiency indicators of the ore occurrence industrial development

The commodity production price determined by three options of evaluation as a product of mining camp annual productivity figures, terms of deposit operation and the commodity production wholesale price which was on the global and domestic markets as of November 2019 (Table 1).

The balance (gross) income of the company was calculated as the difference between the annual volume of commodity production in realization price and the cost price of extraction and finished product transportation to warehouse without fees, taxes and charges stipulated by Ukraine current legislature, which were included into the production cost price. The company income was calculated as the difference between the price of commodity production annual volume in realization prices and the annual cost of production (with considering of depreciation charges). The profitability level determined as the ratio of company income to total cost of fixed assets (investments cost) and operating costs. The recoupment term was calculated as the ratio of their prices to company income.

Table 1

The calculation of the mining camp commodity production price

Indicator name	Units	Reserves estimation options		
		I	II	III
Cut-off grade copper content in one sample	%	0.2	0.3	0.4
Annual mining camp productivity				
copper ore	thousand tons	closed information	2 000.0	1 500.0
noble metals				
gold	kilograms	61.6	50.0	48.3
silver	kilograms	5301.7	4302.8	4156.4
platinum	kilogram	43.7	28.5	21.1
palladium	kilogram	14.6	9.5	7.0
building rubble	thousand cubic meters	208.9	139.3	104.5
stone casting raw materials	thousand tons	1500.0	1000.0	750.0
deposit operation term	years	76.3	75.3	65.0
Wholesale final product price (without AVT)				
metallurgical copper (1 ton)	USD	4 810.00	4810.00	4810.00
gold (1 kilogram)	USD	43050.0	43050.0	43050.0
silver (1 kilogram)	USD	639.0	639.0	639.0
platinum (1 kilogram)	USD	37522.0	37522.0	37522.0
palladium (1 kilogram)	USD	22835.0	22835.0	22835.0

Continuation of table. 1

1 cubic meter of building rubble	USD	16.3	16.3	16.3
1 ton of stone casting raw materials	USD	17.5	17.5	17.5
Income from annual volume commodity product realization				
metallurgical copper	thousand USD	37037.0	28860.0	26936.0
noble metals	thousand USD	8012.77	6188.26	5686.814
building rubble	thousand USD	3405.07	2270.59	1703.35
stone casting raw materials	thousand USD	26 250.0	17 500.0	13 125.0
total gross income	thousand USD	74704.84	54818.85	47451.16
the same for entire operation term	million USD	5699.98	4127.86	3084.34

The results of income calculation, production profitability level by the evaluation options are demonstrated in table 2.

Table 2

The results of income calculation, production profitability level and recoupment term

Indicator name	Units	Ore reserves estimation options		
		I	II	III
cut-off grade copper content in one sample	%	0.2	0.3	0.4
capital costs in deposit industrial development	thousand USD	118611.332	85935.828	72103.6
annual operation costs of commodity production receiving	thousand USD	29368.11	19561.87	16728.53
the same for entire operation term	million USD	2240.788	1473.0112	1087.356
Indicator name	Units	Ore reserves estimation options		
		I	II	III
cost of company deriving from operation	thousand USD	5929.67	4296.29	3604.03
total costs for entire operation term	million USD	2365.328	1563.244	1163.064
income from annual volume commodity product realization	thousand USD	74704.84	54818.85	47451.16

Continuation of table. 1

the same for entire operation term	million USD	5699.98	4127.86	3084.33
Company income without taxes				
annual	thousand USD	45336.73	35256.98	30722.63
for entire operation term	million USD	3459.19	2654.85	1996.97
investments profitability level to profits taxation	%	146	169	172

As it can be seen from the table, the future development of the "Zhyrychi" native copper ore occurrence is profitable by all of considered options of cut-off grade copper content. The profitability level of the production to investments in industrial construction by considered options is higher the higher cut-off grade copper content in one sample is.

Conclusion

The best of considered economic evaluation options (without taxes) for the entire operation term of future deposit for the "Zhyrychi" ore occurrence is as follows. Option 1-0.2% cut-off grade copper content (by company income of 3459.19 million USD). Option 3-0.4% copper content in one assay (by profitability level 172%). The operation of the deposit by those cut-off grades will provide 2.6 times increase the recoupment of primary capital investments. Considering the received geological and economical indicators and the employment opportunity of several thousand people for the long term (65-80 years). We can conclude about indisputable industrial importance of the "Zhyrychi" ore occurrence and its advisability further geological study and industrial development.

Significant necessary investments urge to searching ways of improve the investment attractiveness of copper mining in the Volyn. That might be achieved by increasing of the commodity production price, decrease of operation costs and creation of a favorable investment climate.

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MODELLING AND DESIGN OF TECHNOLOGICAL SCHEMES OF UNDERGROUND DEVELOPMENT OF IRON ORE DEPOSITS AT THE INCREASE OF THE DIFFICULTY OF THE CONDITIONS OF THEIR FUNCTIONING

Popov S.O.
Doctor of Sciences (Engineering), professor, Department of Automation, Computer Science and Technology, Kryvyi Rih National University, Ukraine

Ishchenko M.O.
Ph.D. (Engineering), associae professor, Department of Computer Systems and Networks, Kryvyi Rih National University, Ukraine

Ishchenko L.F.
Ph.D. (Economics), Kryvyi Rih college, National Aviation University Kryvyi Rih, Ukraine

Kolosovskyi D.
Assistant of the Department of Economics, Organization and Management of Enterprises, Kryvyi Rih National University, Ukraine

Abstract. The article is devoted to solving a complex problem that arises in the process of preparation for the development of reserves of iron ores underground, in the conditions of great depths of development and difficult economic conditions of operation of iron ore mining enterprises. This problem concerns the development of reserves of mining sites of iron ore deposits and consists in the need to select and put into practice the most cost-effective design solutions of structures and technological schemes of extractive blocks, which represent the main production facilities for the extraction of ore. The solution to this problem requires the development of methodology, methods and specialized tools for modelling and designing the designs of technological schemes for the development of block reserves, which would allow choosing and profoundly justified and the most economical such solutions at the systemic level of underground mining. To solve this problem, the authors have developed and tested a system of computer simulation and design of the process of mining the reserves of extractive blocks, in a methodological basis and the concept of which is based systematic object-oriented

approach to modelling and design. The algorithm of modelling of technological and logical schemes of working out of stocks of blocks, the procedure of development and application of technical and economic passports of elements of structures of blocks, from which the variants of technological schemes of working out of their stocks are formed and the technical and economic characteristics of realization of these schemes are developed. The system of economic estimation of efficiency of working out of reserves of blocks is also developed. A promising direction for the development of this system is the expansion of its software for more detailed modelling of block elements and their technological schemes, expansion of the database of technical and economic passports of elements of block development schemes and development of graph analytics subsystem of block design support, which provides a significant amount of data for economic and mathematical modelling of block designs and technological schemes of working out their reserves.
Keywords: system, object-oriented approach, computer simulation, flow diagram, underground development.

1. Introduction. At present, underground mining of rich iron ore deposits is one of the bases for the functioning and further development of Ukraine's mining and metallurgical complex. With the use of this method of development, up to 20-25% of commodity iron ore products of Ukraine are produced. Its raw material base is represented by large reserves of rich hematite ores, well-known in the bowels of Ukraine (the confirmed volume of which is 1.4 billion tons and the forecast reserves exceed 3.0 billion tons). The iron content of the ores of this different species reaches 58-66%, which makes it possible to use them for metallurgical processing without enrichment and makes them a valuable raw material resource [1, 2].

However, the conditions for the deposit and mining of reserves of rich iron ores in Ukraine are characterized by increased complexity, namely: a large depth of distribution of deposits, up to 5.0-7.0 km, currently their reserves are being developed at depths of 1.4-1.5 km. The iron ore mines of Ukraine are the deepest mines for the extraction of this type of raw materials in the world and have the following features: deposits of rich iron ores are characterized by significant power fluctuations (20.0 ÷ 150.0 m); deposits do not have clear contours, which requires their special contouring at the stage of geological exploration by methods of economic and mathematical modeling; deposits have complex morphology (internal structure), sharp fluctuations and anisotropy of the physical properties of ores; there are severe restrictions on the permissible magnitudes of anthropogenic disturbance of subsoil by their specific geological structure and terrestrial surface in the presence of natural, industrial and social objects on the surface, as they are not subject to wear and tear [3].

All this significantly complicates the development and extremely and negatively affects the very possibility of its implementation and economic results of development.

The prolonged use of underground mining of rich iron ores in Ukraine, which has been going on for over 120 years, has allowed accumulating a large number of various technical and technological solutions that provide both the possibility of their development in these conditions and the economic feasibility of development [4]. These solutions are: technological schemes of development; construction of extractive blocks; Means of mechanization of mining operations; schemes for the discovery of deposits, floors, preparation of ore bodies, cutting and refining of ore, construction of elements of extractive units and methods of performing various types of mining-related to the extraction of ore and support its implementation [5].

Instead, in a market economy, with its fierce competition for the economic and quality characteristics of commodity iron ore products, it is necessary to ensure not only the technical feasibility and its economic feasibility of development in difficult conditions, but also to achieve its highest economic efficiency. However, the mere use of these solutions does not yet guarantee the achievement of this level of efficiency. This aspect is explained by the fact that the economic results of the implementation of these decisions depend not only on the fundamental ideas underlying it but also determined by the level of their technological, parametric and organizational consistency in the integrated production and technological scheme of the mine under specific geological and mining conditions and mining reserves, as well as the specific economic conditions in which each underground mining enterprise operates [6].

The choice of such solutions and the determination of their effectiveness are carried out at one of the most important stages in the process of preparation for development, namely, the stage of designing mining blocks, which represent the main production objects of mining enterprises [7].

The choice of a complex of such solutions and the formation of high-performance technological schemes for them for specific conditions of development is an urgent problem.

This problem can be solved only by detailed economic and mathematical modeling of the production and technological system of mining the reserves of extractive blocks in the structure and dynamics of the operation of the mining enterprise. These dynamics are extremely uneven (rapid change in the parameters of the block and the process of its development in stages) in the life cycle of construction and operation of the mining block. This cycle begins from the moment the ore reserves are discovered in the depths of the mine field and the determination of its characteristics (volume, quality, conditions of occurrence) until the production of commercial iron ore products from the initial mass. This life cycle is multi-stage and complex in structure.

Such modeling is a complex and time-consuming process. The design and modeling of the construction and operation of mining sites are performed by specialized design departments of iron ore enterprises at the systemic level of mining.

Thus, in the field of underground mining of iron ore deposits, there is a serious and complex problem, which consists in the need to select and put into practice the optimal design solutions for the implementation of ore mining in specific mining and economic conditions based on modeling the development process for the full structure of production technological system of the mining enterprise.

A considerable number of domestic and foreign scientists and specialists dealt with the development of methods and tools for modeling and designing the process of development of mining blocks for the conditions of iron ore mining, namely: Malakhov G.M., Faustov G.T., Stupnik M.I., Korzh V.A., Kucheryavenko I.A., Martynov V.K., Fedko M.B., Pluzhnik Y.A., Baranov A.O., Khavzor L.B., Levinson A.A. and a number of others. At the same time, it should be noted that the works of these authors were mainly aimed at developing methods and software for solving local problems of only certain nodes of internal technological schemes of mining blocks without considering them as elements of the general production and technological system of the mine, ie not at the systemic level of production. The purpose of the article is to develop theoretical foundations and software for modeling and designing the main production facilities of iron ore mines that would consider these facilities at the systemic level of production of the mining enterprise.

2. Formal problem statement. The above problem can only be solved by solving a number of the following problems:

- development of technical and economic models of underground mining and technological objects with high detail of their structural and technological schemes;
- ensuring high accuracy of determination of the parameters calculated on these models;
- a correct economic evaluation of the options of design solutions that are considered and modeled to select the most effective of them;
- determination of conformity of actual final technical and economic results of the development of reserves of extractive blocks with their design values.

It is necessary to wash that the solution of all these tasks is difficult enough. This is since the mining blocks represent production objects, which are characterized by: the large size, complexity of structures and technology of working out their reserves, the complexity of the underground conditions of their construction and operation.

To give an idea of these blocks and the specifics of their work in Fig. In Fig. 1 shows a diagram of the design of the extractive block, the stock of which is worked out by the system of open space development. This variant of the development system is widely used in the iron ore mines of Ukraine. more than 2000 development systems are being developed for the mining of ore deposits, which are being implemented in mining units. The iron ore mines of Ukraine use more than 50 such options [8]. The choice of the optimal variant of the development system according to the technical and economic criteria for the efficiency of their work for the conditions of each particular excavation site is carried out by performing technical and economic modeling of a number of their variants, which is a complex task of increased complexity.

The mining block is an underground mining and technological structure, which has the following basic elements: 1 - extraction section within which the mining block is created (in the simultaneous operation of such sections at the mine may be from 4 to 20, and also 2 - 4 sections are spare); 2 - recoil floor screed (provides access from the shaft of the mines to the ore deposit; 3- surface ortho (access from the screed to the sites of cleaning ore extraction); 4- running / economic lifting ortho (provides ventilation of the block and access to drilling sites); 5 drill bit; 6-section of explosive wells; 7-extraction chamber; 8-ceiling over the chamber (the stock of the ceiling is spent after the ore is extracted from the chamber; 9-between the chamber whole (delimits the camera from the adjacent previously used block, the whole of the stock is spent together with the ceiling); 10-production of the extracted ore from the chamber.

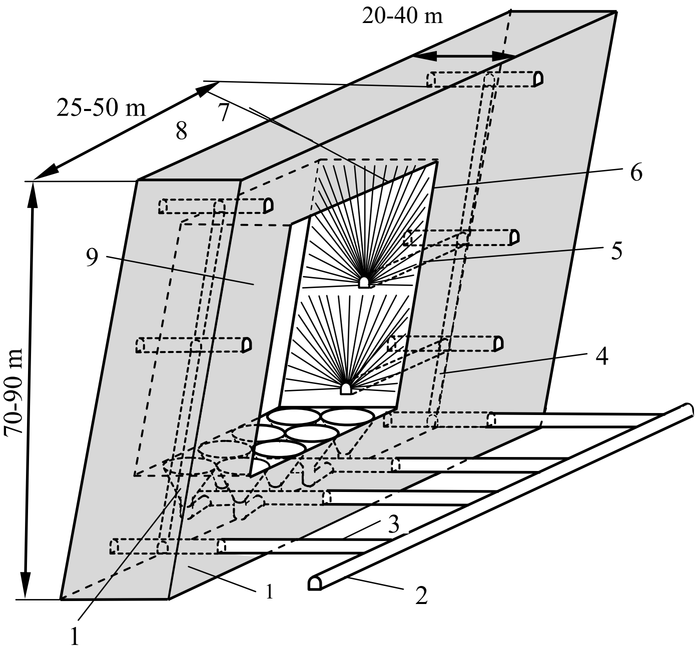

Fig. 3. The scheme of construction of the mining block

The structural and technological parameters of all these elements must be determined by appropriate calculations and justified by a number of factors, namely: geological (by geological structure of a particular excavation site), hydrogeological (by nature of the watering of the array); geomechanical (by parameters of action of mountain pressure, presence of tectonic disturbances, presence of zones of weakening of rocks); technological (on necessary extraction of ore, on technologies of performance of different processes on preparation and refining of ore, on technological connections and dependencies between different processes of development; the rate of profitability of production.

3. Algorithm for system object-oriented modeling and design of technological schemes for mining reserves of mining blocks.To solve this problem, the authors propose the development based on the principle of systematic object-oriented modeling and design of technological schemes for mining reserves of extractive blocks (SOOMAD), which implements the approaches and means of the problem- and object-oriented and system design [9, 10].

The essence of SOOMAD is to solve the problems of designing mining and technological objects with mutual technological and parametric harmonization of all their structural elements, taking into account their role and share in the formation of technical and economic characteristics of the final production of the mining company, implementation of forecasting, planning and control of project parameters. production facilities during their complete life cycle, which makes it possible to take into account the instability of the design objects parameters in the period of their construction and production activities, which is limited by the natural lifecycle of objects (2-5 years depending on their size).

The implementation of this approach should be carried out according to the algorithm shown in (Fig. 2).

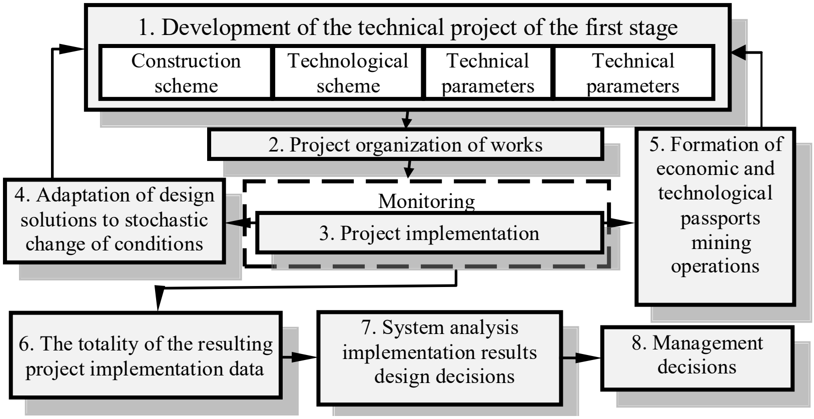

Fig. 4. SOOMAD implementation algorithm

This algorithm includes the following steps:

1. Development of a technical project of a mining and technological object (mining block) according to the theoretical methods (block 1). To prepare the project, economic and mathematical models of several technically acceptable variants of the scheme of the object are formed, their technical parameters are calculated, an economic model is being constructed, the forecast economic characteristics are determined, the timeframe and stages of the project realization are calculated. From these options, the best option is chosen based on the feasibility assessment of the specific system of proposed indicators (described below). Based on this variant the technical design of the block of the first stage is developed.

2. Development of a technical project of a mining and technological object (mining block) according to the theoretical methods (block 1). To prepare the project, economic and mathematical models of several technically acceptable variants of the scheme of the object are formed, their technical parameters are calculated, an economic model is being constructed, the forecast economic characteristics are determined, the timeframe and stages of the project realization are calculated. From these options, the best option is chosen based on the feasibility assessment of the specific system of proposed indicators (described below). Based on this variant, the technical design of the block of the first stage is developed.

3. According to the technical design, the preparation of the project organization of work (POW) (block 2), which develops organizational measures for the construction of a specific unit and the implementation of its basic production functions.

4. After approval of the first stage project, the process of its implementation begins (block 3). Technical and economic monitoring is carried out during the implementation.

5. In the process of project implementation at certain stages, based on monitoring data, an additional adaptation of the already designed scheme of the facility to the actual mining conditions is performed (block 4), since the first stage project is developed on new data on the conditions of implementation of development, which are largely probable. the complexity and imperfection of modern methods of geological prospecting at great depths [8]. The input data for this adaptation is obtained through operational exploration and feasibility monitoring, which establishes:

6. the actual technical and economic characteristics of the facility and its construction conditions; the degree of deviation of the actual geotechnical conditions of its functioning from the design values;

7. deviation of actual object parameters from design. In addition, the actual parameters of resource consumption during the construction and operation of the facility are set.

8. Precise technical and economic data bases (block 5) are formed on this basis. These data are hereinafter accepted as standards and used for further economic-mathematical modeling of such objects.

9. Adaptation of the first stage project to the scale of changes in the project leads to the development of the second stage project directly in the process of practical implementation of the first stage project. Such changes, while maintaining the overall concept of the first stage project, can significantly alter the design and parametric characteristics of the design object with the corresponding changes in the economic results of its operation. These results also need to be forecasted in the second stage of design, choosing options for local solutions based on their predictive performance, that is, the modeling process extends throughout the life of the design object.

10. After completion of the project, the final technical and economic characteristics (block 6) are calculated as a result of the accumulation and analysis of information in the monitoring process. The

obtained data are analyzed (block 7) to determine the actual performance of the object and the magnitude of its deviation from the design and identify the reasons for these deviations.

11. According to the results of the analysis, management decisions (block 8) are made to improve the production activity of the enterprise, the methodological basis of the design process and its information and regulatory framework.

12. One of the main elements in the above algorithm is to determine the number of financial costs required to implement specific design decisions. Based on the calculation of the forecast value of this parameter, the economic effectiveness of the decisions is determined and substantiated, the economic feasibility (or impracticality of their implementation) and many technical and economic indicators characterizing the efficiency of the development are calculated.

4. Mathematical model. To ensure the accuracy of the values of these indicators, the modeling system of mining and technological objects should make economic calculations on the basis of the proposed special logic and mathematical model, which relates the geotechnical conditions of construction and operation of objects and their parameters with the characteristics of resource consumption and economic results of implementation. This model is described by this logic-mathematical function

$$S_o = \sum_{i=1}^{V_Z} \left[\underset{V_i \in V_Z}{V_i} \cdot \sum_{j=1}^{N} \left(x_o @ K_p \Rightarrow \underset{P_x \in P_N}{P_x} \Leftrightarrow \sum_{c=1}^{H} \underset{R_c \in R_P}{R_c} \cdot P_{R_C} \right) \right] \quad (1)$$

According to this function, the amount of financial costs S_o for the implementation of a project decision of a particular object is defined as the sum of the outputs of the volumes of V_i specific types of mining i of all their species V_Z, regulated by this scheme, for the sum of the output of the specific costs of resources of the species R_c from all kinds of resources R_P, identified (as equivalent to $\Leftrightarrow$) the decision taken P_x, at its current price P_{Rc}.

Decision Px presenting economic and technological passport (ETP) each specific elementary mining-technological object, from which the scheme of the projected object is formed. This ETP is selected from the set P_N all possible passports, according to this type of site of the object of design and the corresponding process in the general scheme of mining ($P_x \in P_N$).

Decision Px is an implication (conclusion ⇒) operations of logical relation @ between the selection key xo, which describes the specific mining conditions and the criterion K_p, which describes the conditions of use of each ETP of the respective object.

Economic and technological passport represents a new concept in the field of modeling of mining and technological objects. ETPs are developed in the form of a structured electronic document that reflects the technical and technological solutions, as well as the conditions for the use of elementary mining and technological objects - stripping, preparatory, threaded and other technological work and performing all types of mining operations. In addition, the ETP is used as a software to calculate the unit cost of resources and their value (at current market prices) per unit of output that results from the execution of specific types of mining, namely: the cost of extraction of 1.0 m3 of rock in the workings, drilling of 1.0 linear meter of wells, 1.0 linear meter of drill holes in drilling operations, 1.0 ton of production and delivery of ore from the clearing space, etc.

Construction of models of the structure of large mining and technological objects on the basis of complexes of ETP allows to change at any time local and global technical and technological decisions (schemes, technology, parameters), input data (resource consumption, specific volumes of expenses of specific types of resources, prices for resources), quickly calculate the predicted technical and economic characteristics of objects and final products in the implementation of a specific variant of technical and technological solutions to the above-described logic and mathematical function.

Economic and mathematical models, the structure of which is formed on the basis of the ETP, allow to follow the dynamics of changes in the technical and economic characteristics of objects, that is, to simulate the process of realization of projects, due to the gradual formation of their structure, in accordance with the volumes and types of works by stages of construction and gradual functioning of the object. This allows taking into account the dynamics of expenditures of resources and financial resources, which is important for accurate planning of production activity of a mining enterprise.

The formation of the ETP is based on the actual data of elementary mining and technological objects obtained through object-oriented technical and economic monitoring of the implementation of previously designed objects that are used in this project, which ensures high reliability of the data. These data form the regulatory information base for design modeling of new production facilities using the direct calculation method (rather than the statistical modeling currently used [6]).

One of the most important issues in the design of mining and technological facilities is the correct assessment of the technical and economic efficiency of their design solutions, which underlies the optimization of production.

At present, such an assessment is carried out on the direct characteristics of the objects themselves according to their types, that is, according to the schemes of disclosure, schemes of preparation and cutting of stocks, technological schemes of sewage extraction. This approach is not correct and often leads to poor decisions. For example, when designing a technological scheme for the treatment of extraction, a certain solution at the level of performance of a particular mining unit is effective, but taking into account the factors that operate in the processing of ore into commodity iron ore products, these decisions may lead to low production efficiency. This situation is due to the fact that the final technical and economic characteristics of the products are affected not only by the internal characteristics of the adopted design decision but also by the features of all other processes that are performed to obtain the product. At the same time, the characteristics of all these processes can be in some way interdependent and have both a positive and a negative impact on the end result of mining.

Thus, the correct estimation of design decisions of specific mining and technological objects should be carried out only on the final results of development, ie commodity iron ore products. This approach causes a systematic design. For example, when the high productivity of ore extraction and its entry into the system of its processing into a commodity product, which is a positive factor for the economy of the operation of a particular unit can lead to increased losses of hematite during its processing into a commodity product. In this way, the performance of the unit and the performance of the ore processing must be consistent, and this should be foreseen by the project itself for the refinement of each specific mining unit.

The authors have developed a system of assessment of design mining and technological solutions, the basis of which is the technical and economic indicators, the values of which are calculated according to the economic and mathematical models below.

These metrics include: the total financial cost of building and operating the facility C_t; the quantity of ore that will be obtained during its operation Q_o, or with the use of this facility (when designing disclosure schemes, preparation and slicing); projected balance profit from the sale of P_r ore; total cost of commodity ore C_t; efficiency of financial investments in construction and operation of the projected object F; ore loss ratios of the balance sheet k_l; clogging factor of the ore extracted kcf at its extraction; loss of quality of the extracted ore ΔC compared to the quality of the ore balance sheet; coefficient of removal of commodity product from the subsoil K_r; coefficient of compensation for financial loss from ore losses K_c; specific mine costs MC_s; the cost of extracting commercial ore from the subsoil C_s; the specific cost of building the facility C_b; technical efficiency of financial investments in the construction and operation of the facility E_{ef}; a level of economic efficiency of exploitation of the natural stock of commodity product E_{mn}.

The effectiveness of design decisions is based on this approach. There are two major economic factors that reflect and limit the level of technical and economic efficiency of development, namely: the number of financial costs for the production of commodity ore ΣC_{co} and cost-effectivity of development investments P. These indicators are related to the profit generated from the sale of commodity ore P_{co} and the regulatory cost-effectiveness of development P_n, (14...15%) this relation

$$P = \frac{100 P_{co}}{\sum C_{co}} \geq P_n, \% \tag{2}$$

Value P_{co}, planned by the amount of necessary financial costs of enterprises to ensure the necessary economic efficiency of production. This makes it possible to calculate the predictive value of profits P_i, which must be obtained when the particular projected object is functioning

$$P_i = \frac{P_{co} B_i C_b}{\sum B_j C_{cj}} \geq P_n, \% \qquad (3)$$

where B_i – ore balance will using this facility (opening, preparation and slicing schemes, refining), thousand tons; I_k - the iron content of the ore balance sheet, %; $\Sigma(B_j \cdot C_j)$ – total metal reserves in the ore of all j blocks, scheduled for completion within a specified time, thousand tons, with metal content in the ore of each block C_b, %, which is part of the field operated under the floor plan.

From expression (2), we derive a mathematical model for complex evaluation of the development efficiency.

$$P = \frac{100 \cdot K_f \cdot \frac{100 - k_e}{100 - k_l} k_{em} B \cdot \left[P_m - \frac{\sum C_{oe} + K_n \frac{100 - k_e}{100 - k_l} k_{em} \cdot B \cdot S_t}{K_n \cdot \frac{100 - k_в}{100 - k_{cf}} k_{ot} \cdot B} \right]}{\sum C_{oe} + K_n \cdot \frac{100 - k_e}{100 - k_l} \cdot k_{em} \cdot B \cdot S_t} \geq P_n \qquad (4)$$

where k_{em} - output coefficient of commodity ore from the ore mass, which will be obtained upon realization of this project, UAH; K_f – conversion factor of thousand tons in tons. k_l – predicted (or normative) value of the ore loss factor that will be obtained when the field site is being mined for this project, %; k_l – clogging factor, %; P_m – contractual (market) price of 1 ton of commodity ore, UAH/ton; ΣC_{oe} - costs for ore extraction for this project are determined by the expression modeling method (1), thousand UAH.

In formula (4), particular attention should be paid to the expression (100-k_e/100-k_l). This expression is recognized as the value of the so-called coefficient of visible extraction of the K_{vv}. The particular importance of this coefficient is due to the fact that the volume of the extracted ore extracted from a certain volume of the balance reserve of ore B depends on its value. This value depends on two indices k_l and k_{cf}). Considering that at present the value of technological losses of ore at domestic iron ore mines are high enough and make k_e=9-20% by weight, k_l =7-18%, this has a very negative impact on the economic results of development. For these reasons, up to 20-25% of the value represented by the ore balance sheet is lost at domestic mines, which is a very significant problem. Therefore, when designing the forecasting values of k_e and k_l, special attention should be paid. For this purpose, special techniques have been developed for modeling technological schemes of soldering of reserves of extractive blocks, according to which it is possible to fulfil such forecast and indicators k_e and k_l are one of the main criteria for the choice of rational development systems for specific conditions of its implementation [11].

The k_{lm} value is calculated according to the condition $k_{lm}=k_c \cdot k_f$, where k_c - depends on the allowable difference ΔC_n between the iron content of the ore extracted C_l and its content in commodity ore C_m, %; k_f - depends on the strength factor of the ore f (f= 0,1-30)

$$k_c = \exp\left[\left(426{,}3\Delta C + 16{,}7 e^{\Delta C_n}\right)\cdot 10^{-6} + 9{,}4 \cdot 10^{-3} \Delta C_n - 0{,}01\right] \quad (5)$$

$$k_f = \exp\left[-3{,}3767 \cdot 10^{-6} \cdot \exp(f)\right] \quad (6)$$

Specific mine costs of S_t are determined by the costs of construction of the disclosure schemes C_s and preparing C_{pr}, the rates of depreciation of the openings k_{ap} and preparing k_{an}; costs for their operation, respectively, C_{ep}, C_{en}; maintenance costs of department services C_∂, *.UAH/ton*

$$MC_c = \sum C_t / Q_{mpt} = \left[\left(C_s \cdot k_{ap} + C_{ep}\right) + \left(C_{pr} \cdot k_{an} + C_{en}\right) + C_\partial\right] / Q_{mpt}, \quad (7)$$

where Q_{mpt} - a quantity of commodity ore, realized within the specified period, thousand tons.

The main difficulty in assessing the effectiveness of design mining and technological solutions is to determine the amount of financial costs for the following mining operations: construction workings opening C_s, preparing C_{pr}, and for the extraction of ore ΣC_t, which includes mining and cleaning works (blasting, ore production and delivery). The amount of these costs depends on the specific design and technological decisions of the projected objects. These values can be determined by economically and mathematically modeling their circuits by function (1) and verifying that such a constraint is satisfied

$$\Sigma C_t \le k_{lm} \cdot D \cdot \left(S_{mp} - S_t\right), \quad (8)$$

where D - mined ore, thousand tons; C_p - the permissible cost of commodity ore;

$$S_{mp} = \left(K_n \cdot Q_{mp} \cdot P_{ro} - P_{co}\right) / K_n \cdot Q_{mp}. \quad (9)$$

The difference (S_{mp}-C_t) determines the permissible cost price of C_a of a commodity product contained in a specific volume of ore balance sheet) and is part of the extracted ore mass $Q_{mp}=k_{lm}\cdot D$.

$$S_{amp} = S_{mp}\text{-}C_t \ . \tag{10}$$

The ratio between the values S_{amp} and ΣC_{oe} form the basis for the normalization of ore extraction rates by the criteria of economic efficiency of mining a specific mining block, due to their relationship with the value of the coefficient of visible ore extraction K_{ve}

$$\frac{C_{oe}}{K_n \cdot B \cdot k_{em} \cdot S_{omm}} = K_{ve} = \frac{100 - k_e}{100 - k_l} \ . \tag{11}$$

Normative value k_l can be defined by from the allowable loss of quality of the balance stock in the extraction of ore $\Delta C_p=C_b$-C_{min}, %. From here $k_l=\Delta C_p/(C_b$-$C_a)$, where C_a - is the average iron content of the ore clogging rocks, %. Thereby, the normative meaning k_n can be defined by the expression, %

$$k_n = 1 - \left[\left(1 - k_l\right) \cdot \Sigma C_{oe} / \left(K_n \cdot B \cdot k_{lm} \cdot S_{omp}\right)\right] \ . \tag{12}$$

The following indicators are calculated:

Loss compensation factor K_{cf} for the loss from the loss of commercial ore. This indicator determines the degree of compensation for the profit received from the implementation of the project decision, the amount of financial loss from the inevitable technological losses of ore, \$ units

$$K_{cf} = \frac{100 \cdot P_{co} \cdot \left(100 - k_e\right)}{P_{ro} \cdot k_{lm} \cdot B \cdot \left(100 - k_l\right)}, \tag{13}$$

The unit construction costs of the facility are projected taking into account the ratio (14). This indicator reflects S_c in financial terms, the level of structural and technological complexity of the mining and technological object, as well as the construction conditions for a particular variant of its design decision. The necessity of its definition is caused by the fact that mining and construction works are one of the most resource-intensive and costly works.

$$S_c = \frac{C_r + C_p + C_t + C_{tw}}{B \cdot \dfrac{100 - k_e}{100 - k_l} \cdot k_{em}} = \frac{\sum\limits_{i=1}^{4}\sum\limits_{j=1}^{M}\left[L_{ij} S_{ij}\left(C_{BPDij} + C_{\kappa ij} + C_{oij}\right)\right]}{B \cdot \dfrac{100 - k_e}{100 - k_l} \cdot k_{em}} \ , \tag{14}$$

where C_r, C_p, C_t, C_{tw} – costs for conducting and equipment, respectively: revealing, preparatory, threaded, technological workings, are determined by function (1), thousand UAH.; i - work type identifier; j - i production site identifier with specific geotechnical conditions; M – number of plots; L_{ij} - working length of the *i-th* species in the *j-th* plots; S_{ij} - a cross-section of production; C_{BPD}, $C_к$, C_o, - specific costs, respectively, for the implementation of the BPD, mounting, the equipment production side of the appropriate type, are determined by the selected ETP, in accordance with the conditions of development and the required characteristics of workings, UAH/t.

When evaluating the design of mining blocks, this indicator is an alternative to the indicator "costs of pre-cut work per 1000 tons of balance sheet", which is currently widely used to compare the options of block schemes, but from an economic point of view, it is incorrect because takes into account the individual characteristics of the workings, which differ significantly in different types of workings.

The technical and economic efficiency of financial investments is determined by the amount of commodity ore that will be received per unit of investment in the implementation of a specific project decision. This indicator is one of the most important and reflects how, from a technical point of view, the financial resources invested in the construction and operation of this facility will be used by the amount of commodity ore received per unit of these funds.

$$E_{ee} = \frac{1}{S_{omp} + C_t} = \frac{B \cdot k_{em} \cdot (100 - k_e)/(100 - k_l)}{\Sigma C_{oe} + K_n \cdot B \cdot \dfrac{100 - k_e}{100 - k_l} \cdot k_{em} \cdot C_t} \qquad (15)$$

The level of economic efficiency of the exploitation of the natural stock of commodity ore, determines how economically efficient was extracted and part of the ore Q'_{mp}, determines how economically efficient was extracted and the part of the ore which in the natural state (in the balance reserve) met the requirements for commodity production. This figure is calculated as the ratio of estimated profit from the sale of commodity ore P_i to that part of the ore balance sheet B_i, which has a metal content equal to that of commodity ore. It is taken into account that the profit will be obtained also from that part of commodity ore which will get into ore mass with impurities of other breeds, and also that in the balance stock the metal content C_b can be equal to commodity $C_b=C_c$, or less $C_b<C_c$, or more $C_b>C_c$.

$$E_e = \frac{P_i}{Q'_c} = \frac{Q_{mp} \cdot P_{ro} - \sum S_{mp}\left(100 - k_e\right)}{B_i\left(C_b / C_c\right)\left(100 - k_l\right)}. \tag{16}$$

where Q'_c - commodity ore, which was contained in the redeemed balance sheets, tons.

The technical efficiency of the exploitation of the industrial ore stock should be estimated by the coefficient of extraction from the bowels of commercial ore K_{bco}. This figure reflects the amount of commodity ore that will be received from a given site of the deposit, compared to the amount of ore contained in the natural balance stock and meets the consumer's requirements for the commodity product.

$$K_{dco} = \frac{B_i \cdot \left(C_b / C_{mp}\right)}{Q_{mp}} = \frac{B_i \cdot \left(C_b / C_{mp}\right)}{K_n \cdot B \cdot \frac{100 - k_e}{100 - k_l} \cdot k_{em}}. \tag{17}$$

In the aggregate, all of the indicators described reflecting the effectiveness of the design decision of a particular mining and technological object in sufficient volume to substantiate the feasibility (or impracticality) of its selection and implementation in practice, especially when comparing complex construction and structure and technology of objects. Sufficiency of the described descriptions of the interests SOOMAD implemented by the authors of the automated system of the « Geotechnologist » project. The elements of the system have passed approbation in a number of mines of the Krivorizhsky Zalizruznogo pool. In the second house, a number of important project-related mining-technological tasks were solved, and the following: the optimal scheme for the preparation of a family home at the field of Ternivsk PJSC «Krivbaszalizrudcom» block was completed, the experimental project of a 149-154 in the minds of the «Zhovtneva» mine of OJSC KZHRK; the optimum geometric parameters of the structural elements of the view blocks at the Yuvleyna mine and the PJSC “Suha Balka” mine are marked; The optimal technological scheme for the production of the “Golovna” mine of the “Yuvileyna” mine was formed.

Today, a more advanced data system and software development support for the model of high-tech elements of industrial circuits and additional mine and technological systems of mines for expanding the number of design tasks are needed.

5. Conclusion. Based on the above materials, we can draw the following conclusions:

The problem of selection, implementation in practice and determining the actual effectiveness of design solutions for the development of reserves of extractive blocks, which represent the main production facilities of mines, and is currently one of the most important, complex and not yet fully resolved the field of underground mining of rich ores in difficult mining-technical and economic conditions.

The need to solve this problem requires the development of methodology, methods and specialized tools for modeling and designing structures and technological schemes for the development of block reserves, which would allow to select and substantiate the technical and economic efficiency of development solutions at the systemic level of production at underground iron-ore mining.

To solve this problem, the authors have developed a system of computer simulation and design of the process of working off the belts of mining blocks, in a methodological basis and the concept of which laid an approach to considering each mining block as an element of a complex and branched production and technological system of the mine, which is unique in terms of working out its stock and parameters, has a clear life cycle, stage of implementation. The efficiency of this process can be correctly evaluated only by the parameters of the final iron ore product, which will be produced from the block ore reserve with its specific parameters (volume, natural quality of the ore, geological and mining conditions of occurrence and mining of the reserve).

The authors have developed an algorithm for modeling technological schemes of mining reserves of extractive blocks, the procedure for the development and use of technical and economic passports of elements of block structures, from which the variants of technological schemes of working out of their reserves are formed and the choice of the optimal option is determined, the technical and economic characteristics of its implementation are determined and substantiated. developed a special system for economic evaluation of the effectiveness of the application of technological and logical schemes of development and economic and mathematical models to determine values of economic indicators that make up the system.

Further development of the developed systems of modeling and designing is the expansion of its software to solve more specific problems of modeling of block elements and their technological schemes and their design, expansion of the base of technical and economic passports of elements of technological schemes of working out of stocks of blocks and development of graph analytic subsystem of support for mining of production blocks. , which provides a large amount of input data for the economic and mathematical modeling of block structures and technological schemes of working out their reserves.

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PREDICTION OF ROCK PRESSURE EFFECTS IN MINE WORKINGS IN THE ZONE OF LONGWALL FACE INFLUENCE

Didenko, M.
Candidate of Science, associate professor of Mining sub faculty of Volodymyr Dahl East Ukrainian National University, Severodonetsk, Ukraine

When dredging flat coal seams using a pillar system of development, before the approach to the face, the preparatory workings go through three characteristic periods of their existence:

I – in the zone of influence of the face;

II – outside the zone of influence of the face;

III – in the zone of temporary abutment pressure in front of the longwall face.

Mine research indicates significant features of the rock pressure effect in these zones. To study the distribution of stresses around the mine working developed in the zone of temporary abutment pressure of the moving longwall, a three-dimensional elastic numerical problem was solved by the finite element method.

The object of the study was a rock mass in the vicinity of a single longwall developing with a pillar system a flat coal seam.

The subject of research is the stress-strain state of the rocks around the mine working in the zone of temporary abutment pressure of the longwall, depending on the type of roof (easily-, medium- and hard-to-break) and the length of the cantilever of suspended rocks (0-5 m for easy-breaking roof; 0-20 m for medium-breaking and 0-35 m for hard-breaking).

The study and prediction of geomechanical processes that is realized around the longwall face will significantly solve the problem of the stability of mine workings.

Introduction

As a result of the coal production in the undermined stratum, complex and largely random geomechanical processes occur. Part of the rocks, located directly above the developed space, is deformed and collapses, losing contact with the rock mass. The rock layers lying above bend over the worked out space and play the role of rock slabs redistributing the load from the overlying stratum.

Regardless of the technology of coal mining, the movement of the longwall face and the periodic process of hovering and collapse of suspended rocks change the stress-strain state of the formation and surrounding rocks. The stress distribution around the longwall, first of all, depends on the step of the main roof sedimentation and the length of the hanging rock console, reflecting the rhythmicity of the processes of coal extraction.

Currently used methods for controlling geomechanical processes in time of coal production have exhausted their capabilities and do not guarantee the safety of mining. A further increase in the efficiency of coal production and ensuring the reliability of the mine workings is possible only on the basis of the results of modern fundamental studies of the mechanism of formation of rock pressure around the mine workings and the laws of its effect [1].

In this case, the main tasks that need to be solved are determining the size of the zone of abutment pressure, studying the distribution of stresses in the abutment zone, evaluating additional displacements of the rock formation contour and loads on the lining. The solution to these problems is based, first of all, on the analysis of the stress-strain state of the rock mass around the face during the excavation of a flat coal seam by a long face with a complete collapse of the roof.

Currently, numerical methods for solving geomechanical problems, in particular, the finite element method (FEM) [2, etc.], which allows to implement the principles of simulation modeling of geomechanical processes, are most relevant to the specific task of assessing the properties of the medium under study and the mechanical state of the rock mass near the longwall face on a PC.

The volumetric numerical problem of the distribution of stresses in the rock mass around a single longwall face

The stress distribution in the zone of temporary abutmrnt pressure was studied in order to establish the dimensionless function of the influence of the longwall face on the stress state of the rocks around the mine working, which is supposed to be used as a multiplier to the components of the initial stress field when studying the effects of rock pressure in the mine on a flat model taking into account the fracture zone rocks formed at the time of the beginning of the coal extraction. Therefore, the problem was solved in an elastic setting for a transversely isotropic massif without taking into account the bedding and fracture of the roof rocks. On the one hand, such idealization leads to a simplification of the mathematical model, and on the other hand, unaccounted characteristics increase the stresses, which go into the margin of safety and are acceptable at the initial stage of research.

The mathematical model represents a prismatic fragment of the rock mass. The fragment sizes are taken taking into account the angles of movement of the undermined rock mass. The height of the model is 780 m, the width is 448 m. The size of the model in the direction of movement of the longwall face is 310 m. Due to the symmetry, the design model includes half of the longwall.

A coal seam with a thickness of 1 m lies at a depth of 650 m from the earth's surface to the soil of the seam. A working face with a half length of longwakk 129 m is located in the middle of the model, while the distance from the face to the end face of the fragment in front of the longwall is taken to be 100 m, the distance from the worked out space to the side face is 319 m, the width of the working space of the longwall is 5 m; the length of the cantilever of suspended rocks varies from 0 to 35 m, and the length of the worked out space where the collapse of the rocks was realized and their compaction occurs is 170 m.

The geological structure of the rock mass is selected in relation to the mines of Donbass. The cross-sectional shape of the mine has been adopted rectangular with a lower blasting of rocks, the method of protecting the preparatory mine from the mined-out side is a strip of prefabricated reinforced concrete blocks (БЖБТ), the mine working is reused, therefore it is maintained behind the longwall.

The calculated fragment is loaded with its own weight of rocks and coal under the following boundary conditions: on the lower surface of the model, there are no vertical or horizontal movements; along the lateral and end vertical surfaces of the model, there are no horizontal components of displacements and there are vertical as well as tangential displacements.

The modeling in the settlement schemes of the collapsed rock zone of the worked out space was based on the well-known assumption of the three-zonal nature of the rock displacement along the vertical of the undermined massif and the gradual compaction of the rocks of the random collapse zone (horizontal), which was limited by the size of the layer corresponding to the thickness of the formation. Based on the generated model, a calculation was implemented using the «Лира-9» computer software package [3, 4]. In total, 12 variants of the numerical problem were solved to determine the initial stress field, the stress field after the preparatory development of a rectangular cross section, as well as the stress-strain state of the rock mass during the coal production.

Function of the influence of the longwall face on the stress distribution in the zone of temporary abutment pressure

The studies showed that an adequate forecast of the effects of rock pressure in the mine working, which is exposed to the longwall influence, when modeling on an elastic model of the massif is impossible.

The development of a numerical spatial elastoplastic model taking into account the extreme deformation of rocks is a separate and very difficult task. The use of a flat numerical model in the zone of temporary reference pressure is incorrect, since the longitudinal strain is $\varepsilon_y \neq 0$. However, with practical accuracy it is possible to reproduce the patterns of deformation of the rock mass around the preparatory development in the zone of temporary abutment pressure of the longwall by replacing the spatial problem with a series of successively solved plane problems when the initial and boundary conditions at each subsequent step are set based on the solution of the problem at the previous step.

To maximize the display of processes of stress redistribution and displacement of a layered rock mass in the vicinity of a mine working, the following factors are taken into account: an alternating field of stress distribution at various distances to the longwall face; the real structure of the layered carbon-bearing stratum with heterogeneous mechanical characteristics and broken contacts between the layers; complete rock deformation diagram; joint deformation of the array and lining.

The correction of boundary conditions in the zone of temporary longwall abutment pressure is performed by multiplying the stresses by the concentration coefficient k_σ. The dependence of this coefficient on the distance to the longwall face is a function of the influence of the face.

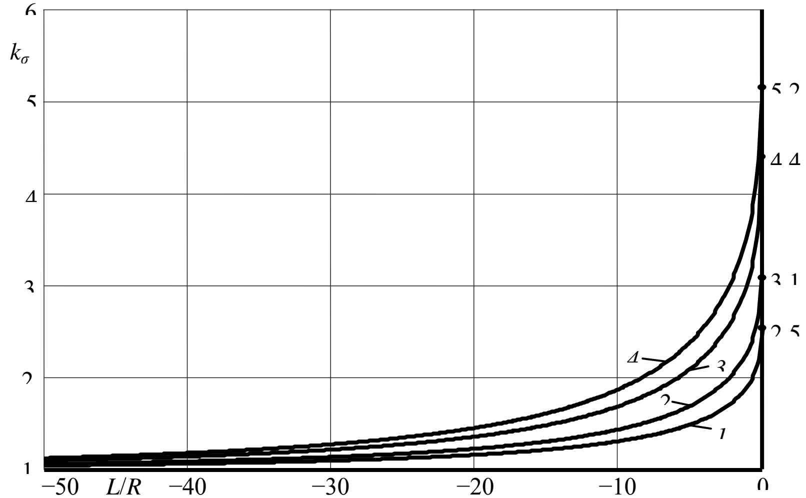

Fig. 1. The influence function of the longwall face in the zone of temporary abutment pressure at $l_к = 1$ (*1*); $l_к = 2$ (*2*); $l_к = 5$ (*3*); $l_к = 8$ (*4*)

The type of this function was determined by calculating the three-dimensional elastic problem. Processing the results of the least squares simulation gives the following form of function

$$k_\sigma = 1 + \frac{b}{\exp(0{,}5 \cdot \sqrt{-L/R})} \tag{1}$$

L is the distance to the longwall face,

R is the reduced radius of the mine working.

The value of coefficient b depends on the type of roof and the length of the cantilever of suspended rocks $l_к$, reduced to a dimensionless form:

$$l_k = 1 + l_з / l_p \tag{2}$$

l_p is the width of the working space of the longwall, m;

$l_з$ is the length of the cantilever of suspended rocks (for an easily collapsed roof it is 0 to 5 m, for a medium collapsed 2 to 20 m, for a hard collapsed 4.5 to 35 m).

The value of coefficient b is approximated with a theoretical correlation ratio of 0.99, a power function

$$b = 1{,}503 \cdot l_k^{\,0{,}528} \tag{3}$$

Given formula (3), dependence (1) can be written as

$$k_\sigma = 1 + \frac{1{,}5 \cdot \sqrt{l_\kappa}}{\exp(0{,}5 \cdot \sqrt{-L/R})} \tag{4}$$

An analysis of the function (4) for various values of the relative length of the hanging rocks is presented in Fig. 1. On the face line, i.e. at $L = 0$, the range of variation of the stress concentration coefficient is from $k_\sigma = 2.5$ for an easily collapsible roof to $k_\sigma = 5.24$ for an hard collapsed roof.

Prediction of rock pressure effects in the mine working during its development

To predict the rock pressure effects, a specially developed program was used to obtain a stress field around a mine working on a flat model of the rock mass taking into account the extreme deformation of the rocks. The solution to the problem is performed on the same network of finite elements for the three stages of development:

- in the zone of influence of the mine working face during its implementation;
- outside the zone of influence of longwall, i.e. after the formation of a stationary stress field before the impact of the longwall face;
- in the zone of the longwall temporary abutment pressure.

At the construction stage of the mine working, a drift of a rectangular cross section was simulated using a tunneling machine on a coal seam with a lower blasting of rocks. The geomechanical conditions of the drift construction corresponded to the volumetric problem considered above. The forces specified along the periphery of the computational domain are selected so that the initial stress state, which is characteristic of the gravitational stress field in an undisturbed array, is correctly reproduced in the planar system under study.

The computational domain consists of 10931 triangular elements and 5543 nodes. To simplify the calculation, the selected area of the array was considered without taking into account the dead weight of the rocks. Along the lower boundary of the calculated fragment, knot displacements were forbidden.

To simulate the initial field of gravitational stresses along the external contour of the model, concentrated forces were applied corresponding to the acting stresses in the volumetric problem. At various distances from the bottom, the external load was corrected by the working face impact function [5]. The action of the lining on the rocks along the output circuit at this stage of the calculation was not taken into account. For all types of rocks and working sections, a unified model of elastoplastic deformation of rocks was adopted taking into account a linear decrease in rock resistance beyond the ultimate strength (ε_p = 3 ε_y, $\sigma_o = 0.1\ \sigma_c$).

According to the calculation results, the maximum principal stresses σ_1 are concentrated in the sides, as well as near the lower and upper corners of the mine working, and the minimum stresses σ_3 are in the mine roof. Already at a distance of 1.6 m from the face of the mine (before the installation of the lining), an inelastic deformation zone was formed along almost the entire contour of the mine working. The area occupied by the destroyed elements in the plane of the model was 90% of the cross-sectional area of the mine working. At a distance of 188 m from the face, it amounted to 120%, which is, increased 1.35 times. The greatest increase is observed in the soil of mine working. The maximum depth of the fracture zone is not founded in the coal seam, but under it.

Prediction of rock pressure effects while maintaining mine working outside the zone of influence of longwall face

After stabilization of the stress distribution field, deformation of the rocks around the mine takes place at a constant load, but with varying long-term rock strength, which was set in the layers of finite elements surrounding the mine working in accordance with the expression

$$\sigma_c(r) = \sigma_c \cdot (1 - a \cdot r^{-n}) \qquad (5)$$

where r is the radial coordinate referred to the reduced radius of the mine;

a, n are dimensionless approximation parameters taken equal to $a = 0.35$, $n = 3$).

The distribution of stresses around the mine outside the zone of influence of working and longwall faces is fully consistent with the distribution of stresses during mine working development. However, as a result of a decrease in the rock strength over time, the relative area of the destroyed elements reached 130%, i.e. the area of the destruction zone increased 1.1 times. Moreover, the growth of the destruction zone is observed in all directions, and the greatest increase occurs in the soil of the mine working. Consequently, soil heap will occur in the mine working.

Prediction of rock pressure effects while maintaining mine working in the zone of temporary abutment pressure

Prediction of rock pressure effects in the zone of influence of longwall face is performed taking into account the zone of rock destruction around the mine working, which was formed by the time the mine workins occurs into the zone of temporary abutment pressure. Two calculation options have been performed: for an easily collapsible roof (the minimum cantilever of suspended rocks) and for a hard-to-collapse roof (maximum cantilever of a suspended rock).

When the roof collapses with the approaching face near the working face, the appearance of the stress field is maintained, however, the values of both compressive and tensile (modulo) stresses increase, with the maximum principal stresses σ_1 increasing 2.36 times and the minimum σ_3 increasing 2.25 times. As a result of the general increase in stresses, the relative area of the fracture zone also increases, reaching 210% on the longwall face, which is 1.6 times the size of the zone of inelastic deformations outside the zone of influence of the longwall.

With a hard-to-collapse roof in the zone of temporary reference pressure, the maximum principal stresses σ_1 increase by 3.7 times, and the minimum principal stresses σ_3 increase by 3.9 times. At the same time, the relative area of the fracture zone on the longwall face line was 250%, which is 1.9 times larger than the size of the zone of inelastic deformations outside the zone of influence of the longwall face.

The range of changes in the relative area of the destruction zone with an easily collapsible roof with a minimum cantilever and a hard-to-collapse roof with a maximum cantilever was 210-250%, i.e. the area of rock destruction in the most difficult conditions of maintaining the development is 1.2 times higher than the same indicator for the lightest mining and geological conditions.

Prediction of rock pressure effects when maintaining a mine working with a spacer support

Prediction of rock pressure effects around a timbered mine working taking into account the active spacer support was made for the same conditions. In the development of a rectangular cross-sectional shape, the effect of adaptive support was simulated with a two-stage effect on rock outcrop due to the expansion of racks between the soil and the roof of the mine [6, 7]. The effect of the compressive force on the destroyed rocks of the zone of inelastic deformations of P = 560 kN was simulated, as well as the support resistance in the roof $P_{кр}$ = 435 kN/m and in the sides $P_{б}$ = 200 kN/m.

Calculation of the bearing capacity of the rock mass – support system in the area of the extreme deformation of the rocks, the depth of the rock compaction zone, the averaged coefficient of fracture voidness of the rocks, strength and deformation parameters of the finite elements within the compaction zone was carried out according to the method described in [8].

The depth of the compaction zone in the roof and sides of the mine was taken to be 1.25 m, and the values of the average coefficient of fracture voidness of the rocks at various distances to the bottom were:

1.6 m – k_{mp} = 0.0133; 8 m – k_{mp} = 0.0136; 20 m – k_{mp} = 0.0147; 188 m – k_{mp} = 0.0154.

To simulate the effect of the sealing force on the properties of the marginal rocks within the zone of influence of the lining (the roof and the working sides at a distance of 1.25 m from the contour), 3 layers were selected for which the residual strength and softening strains were assigned as follows. In the first layer (0-0.25 m from the contour)

ε_p =15 ε_y, σ_o =0.9 σ_1; in the second (0.25-0.75 m from the circuit)

ε_p =10 ε_y, σ_o =0.75 σ_1; in the third (0.75-1.25 m from the circuit)

ε_p =6.6 ε_y, σ_o =0.5 σ_1. For all other layers and types of rocks

$$\varepsilon_p = 3\, \varepsilon_y,\ \sigma_o = 0.1\, \sigma_1.$$

In fig. 2 and fig. 3 it is shown the configurations of rock destruction zones around a mine working with easily collapsible and hard-to-collapse roofs in the zone of influence of longwall face during compaction of the destroyed rocks and expansion of the lining at various distances from the longwall face line.

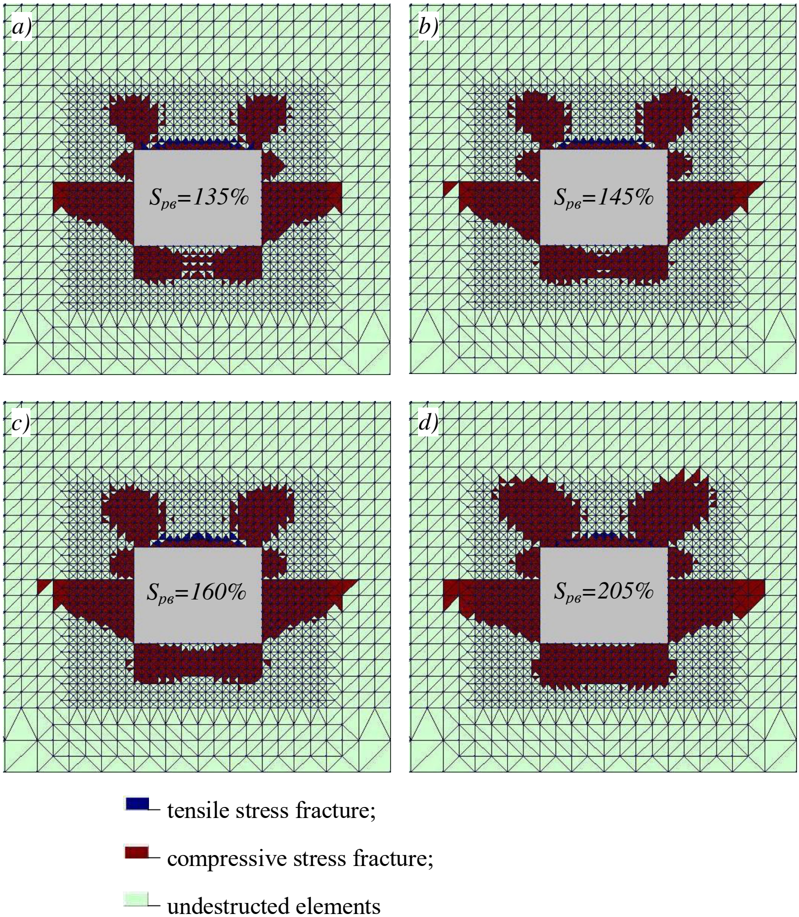

Fig. 2. The configurations of rock destruction zones around a mine working with easily collapsible roof in the zone of influence of longwall face during compaction of the destroyed rocks and expansion of the lining at various distances from the longwall face line: 20 m (*a*), 10 m (*b*), 5 m (*c*), 0 m (*d*)

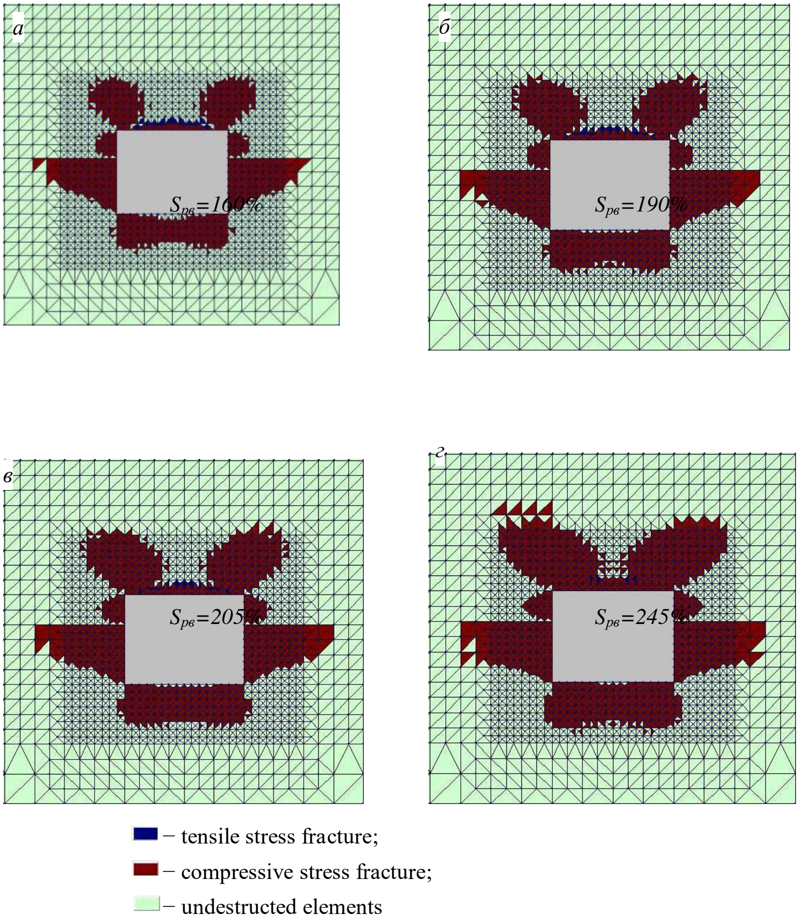

Fig. 3. The configurations of rock destruction zones around a mine working with hard-to-collapse roof in the zone of influence of longwall face during compaction of the destroyed rocks and expansion of the lining at various distances from the longwall face line: 20 m (*a*), 10 m (*b*), 5 m (*c*), 0 m (*d*)

With a hard-to-collapse roof in the zone of temporary abutment pressure, the relative area of the fracture zone on the longwall face line was $S_{рв}$ = 2.45, which is 1.01 times smaller than the size of the zone of inelastic deformations to produce under similar conditions. For an easily collapsible roof with a minimum cantilever length of suspended rocks, the decrease in the fracture zone on the face line due to compaction and lining action was 3%. Therefore, it is inefficient (useless) to compact the destroyed rocks and the spacer support in the zone of temporary abutment pressure of the longwall face. It has practically no effect on the change in the size of the fracture zone around the mine working.

Table 1

Change in the area of the destruction zone around the mine working

Conditions for maintaining the mine working		The relative area of destruction $S_{рв}$		The coefficient of reduction of the area of destruction
		excluding support	taking into account the impact of support	
In the zone of influence of mine working face at a distance:	1.6 м	0.88	0.69	1.27
	8 м	0.93	0.76	1.23
	20 м	1.08	0.95	1.14
	188 м	1.19	1.07	1.12
Outside the zone of influence of the mine working and longwall faces		1,31	1.19	1.1
In the zone of influence of the face with an easily collapsible roof and a minimum console at a distance:	0 м	2.08	2.03	1.03
	5 м	1.67	1.58	1.06
	10 м	1.57	1.47	1.07
	20 м	1.44	1.33	1.08
In the zone of influence of the working face with a hard-to-collapse roof and a maximum console at a distance:	0 м	2.48	2.45	1.01
	5 м	2.08	2.03	1.03
	10 м	1.92	1.87	1.03
	20 м	1.67	1.59	1.05

In the table 1 it is presented the results of calculating the relative area of destruction around the mine working for all options. The table shows that the maximum reduction in the area of destruction around the mine is achieved when installing spacers in near the face of the mine working. The effectiveness of compaction of rocks decreases as the distance from the face to the point of expansion of the support increases. It should be noted that the spacer of the support at a too small distance from the face does not provide effective compaction, since in this case the area of the fracture zone at the point of expansion of the support is small, which does not allow compaction of the destroyed rocks to a greater depth.

Conclusions

Thus, controlling the rock pressure effects with the help of preliminary expansion of the support frames and compaction of the destroyed rocks in roof of mine working near its face makes it possible to reduce the area of the inelastic deformation zone in the impact zone of the longwall. When the support is spread at an optimal distance, depending on the deformation-strength properties of the rocks and the initial stress state of the rock mass, the effectiveness of compaction of the destroyed rocks is much higher, which allows you to assign rational support parameters.

The above methods of prediction of rock pressure effects were used to develop recommendations for improving the reliability of the 9th western conveyor incline of the l_6 layer of the coal mine "Mashchinskaya".

During the 9th western conveyor incline construction, the displacements of the roof rocks according to the prediction technique were 181 mm, which is consistent with the results of displacement measurements using depth benchmarks (168 mm). Taking into account the subsidence of the roof, in the reference pressure zone in front of the first longwall face, during the reuse period behind the first and in the abutment pressure zone in front of the second longwall, determined in accordance with the standard [9], the total rock displacements the roofs will be 1267 mm, which exceeds the constructive compliance of even a five-link arched support and necessitates a double undermining of the soil (in the pressure zones in front of the first and second longwall faces).

Recommendations for improving the reliability of the 9th west conveyor incline provide for the use of a trapezoid support of increased bearing capacity (КПС) from СВП-27 with straight racks instead of a three-link arched support, and installation of the frames of this support with a spacer, carried out with the help of two hydraulic racks 2СУГ30 installed under the top, which corresponds to a sealing force of 560 kN. Using computer simulation the distance from the face is optimized, at which the lining spacing is carried out (28 m) and which corresponds to the minimum displacements of the roof (89 mm, which 2.03 times less than the base version). As a result of the expansion, compaction of the roof rocks has been achieved, due to which their total displacements over the entire service life of the mine working have been reduced to 677 mm, which corresponds to the constructive compliance of the trapezoidal support (700 mm). The cross-sectional area of the mine working when fastened with a КПС support $S_{св}$ = 11.4 m^2, which is smaller compared to the arched shape with the same width. The load on the lining, determined according to the standard [9], is 208 kN, and the density of the lining with its working resistance of 240 kN is 1.0 frame/m. The economic effect of the introduction of recommendations for the entire service life of the development is 2987.9 thousand UAH (roughly 100 thousand €).

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OPTIMIZATION OF ENVIRONMENTAL ENGINEERING PROTECTION AND WATER RESOURCE USE AT THE MOST WATERED MINE IN UKRAINE

Sadovenko I.O.
Dnipro University of Technology, Dnipro, Ukraine, Dr. of Tech. Sciences, Professor, Professor of Dept. Hydrogeology and Engineering Geology

Rudakov D.V.
Dnipro University of Technology, Dnipro, Ukraine, Dr. of Tech. Sciences, Professor, Head of Dept. Hydrogeology and Engineering Geology

Zahrytsenko A.M.
Dnipro University of Technology, Dnipro, Ukraine, PhD in Tech. Sciences, Assoc. Professor, Assoc. Professor of Dept. Hydrogeology and Engineering Geology

Derevaygina N.I.
Dnipro University of Technology, Dnipro, Ukraine, PhD in Tech. Sciences, Assoc. Professor of Dept. Hydrogeology and Engineering Geology

Abstract. This study deals with the environment protection of mining areas affected by flooding and waterlogging in Western Donbas. We used finite difference numerical modelling to analyse the hydrogeological regime of mining sites as complex geotechnical systems being transformed under the multifactorial influence and parameter uncertainty. The developed ground water flow model of the most water-abundant mine in Ukraine named after N.I. Stashkov has been validated using the available data on chronological stages of mining. This enabled to identify the patterns of temporal and spatial changes of flow parameters, the locations of high transmissivity zones in rocks and balance water flows in carboniferous and shallow aquifers.

Using the model we assessed the hydrodynamic conditions expected after closing and flooding the mine, particularly, the risks of near-by mine operation and environmental consequences to the ground surface. The groundwater level rebound is expected to take three years with emerging the zones of potential flooding and waterlogging in the Samara River floodplain. We compared four options of environmental engineering protection including the installation of stationary or submersible pumps in the mining horizon used for drainage, with evaluating the cost-effectiveness and anticipated environmental effect. The alternative option has been recommended to

protect the floodplain from waterlogging by installing a groundwater intake in high permeable sandstones in the paleochannel.

Introduction

Water regulation is one of the key issues at every stage of coal-mining, beginning from commissioning and pitting into operation to closure and the post-mining period. Water regulation influences both the company profitability and environmental and technical safety [1-3]. The water issues are the most challenging in terms of predictability and possibility to control due to intensive man-made transformations of rocks. A clear vertical zonation of the water exchange rate formed under natural conditions was violated during mining, with the zone of intensive water exchange extending down due to excavation roof collapse. This enhanced hydraulic connection between underground excavations, cover sediments, and surface watercourses; thus increasing the mine water inflow and the depletion of fresh water reserves.

Exhausting mineral reserves inevitably approaches the stage of closure designing and decommissioning that should take into account significant changes of ground water flow boundaries, flow and capacity parameters of rocks, ground surface subsidence, hydrochemical inversion etc. [4]. As a result of mine level rebound the flooded drainless areas appear in post-mining territories. Enhanced hydraulic connection among the mines determines the need to elaborate the measures on hydraulic protection of near-by active mines. Therefore, a tool for prediction and control over the processes accompanying flooding the mines as well as assessments for efficiency improvement of engineering solutions is getting of growing importance for mining areas of Ukraine.

A number of different methods and techniques have been used last decades to predict flooding the mines methods; they include finite difference method [5], finite element method [6], balance method in the form of the Box model [7, 8]. Finite-difference numerical models of ground water flow for the full life-cycle of a mine taking into account subgrid scale stochastic heterogeneity of affected rock and flow properties were proposed and tested in [9].

Besides, an analytical model of flooding the mines was first presented in [9] and further developed in [10]; it extended the known “big well” approach [11] to non-steady case and took into account vertical heterogeneity and, as distinct to existing analytical methods, temporal mine water level variation. The model allows easier and closer to reality analysing mine water rebound in case of hydraulically isolated mines but faces difficulties in simulating ground water flow in multiple adjacent coalfields with complex 3D geometries of mined out space.

Up to now, the widely used and most common and reliable method for predictive assessments of flooding multiple hydraulically connected mines is finite-difference modelling that takes into proper account 3D rock heterogeneity and complex geometry of geological formations, time-dependent boundary conditions and other factors, crucial for reliable evaluations and engineering protection design [12, 13].

In the Western Donbas, the challenges of the forthcoming mine closure become highly relevant in terms of emerging risks to the safety of active drained near-by mines. Beyond that, about 75% of the region's coal reserves are located in topographic lows and river floodplains with complicated hydrogeological conditions for mining.

The operating company DTEK plans to close the mine named after N.I. Stashkov as one of the most watered mine in Ukraine by 2021; the outflow of this mine significantly affects the hydrogeological conditions in near-by coalfields and the groundwater level in shallow aquifers. For this reason, the parameters of flooding the mine named after N.I. Stashkov should be optimized in the near future under the criteria of cost-effectiveness and mitigation of environmental impact including restoration of natural hydrogeological conditions.

In this regard, this study aims to optimize the environmental engineering protection and water resource use at the mine to be closed by adapting the method of predicting groundwater flow to the specifics of mined out coalfields.

Methods

To identify the patterns of temporal and spatial changes of the groundwater regime in the selected site we used the finite-difference method and the software MIF developed at the Ukrainian State Geological-Prospecting Institute (Dnipro, Ukraine). This software was applied to hydrogeological analysis of numerous sites in Ukraine including water regulation issues in mining areas [14, 15]. Numerical modelling of mining hydrodynamics included evaluation of groundwater regime parameters for the period of active coalmining and predictive calculations for closure and the post-mining period.

The developed numerical groundwater flow model reproduces the real mining site in terms of analogy to its geological settings and hydraulic response to man-made interventions including underground mining and operation of water storage ponds, water intakes, pumping stations, drainage facilities, etc. The model calculates groundwater flow parameters in the upper cover sediments and the lower water permeable carboniferous rocks with coal seams hydraulically connected by the shafts.

The model allows varying the technical parameters of water regulations and making assessments of the effect of water management measures on improving environmental safety on mining areas.

Model development and validation

The groundwater flow model covers the territory of two near-by coalfields (mines "Dniprovska" and named after N.I. Stashkov) hydraulically connected through the Buchak aquifer, and the coal-bearing rocks separated by a tectonic fault (Fig. 1).

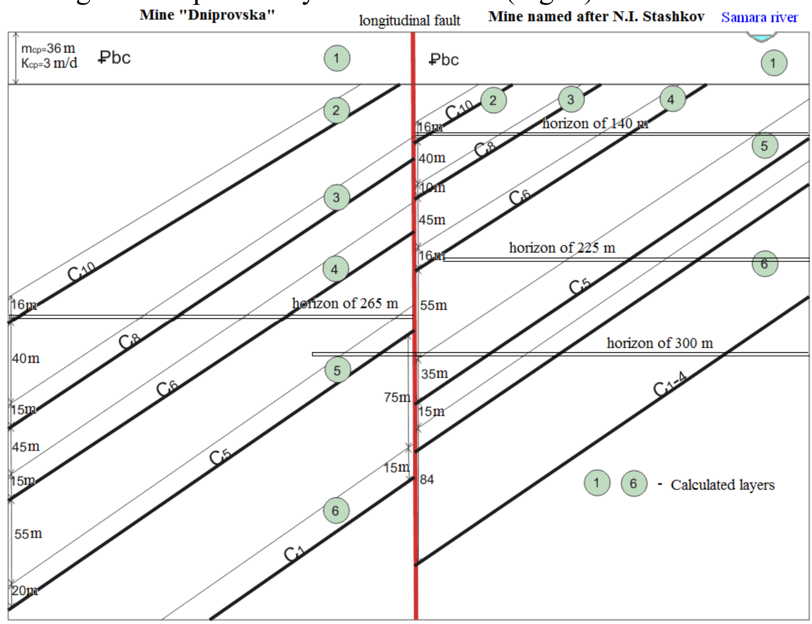

Fig. 1. Schematic vertical cross-section of the model

The modelled area of 16.6 km × 14 km was gridded by rectangular elements of 200 m × 200 m size. The mine "Dniprovska" is located northeast of the tectonic fault and the mine named after N.I. Stashkov is southwest of the fault.

In the vertical section, the model reproduces a 6-layer rock stratum, with the first (upper) layer simulating the aquifer in permeable Buchak sediments, and the lower layers inclined at the angle of 2-5°. The latter consist of rocks in the zone of water permeable cracks and completely or partially mined out coal seams (Fig. 1).

The model has been validated through a series of calculations using known natural and disturbed hydrodynamic conditions and available monitoring data. This allowed evaluating groundwater levels and water inflows as well as the coefficients of boundary conditions, flow and capacity parameters in all permeable layers from the beginning of mining to nowadays.

The overall evaluation of model parameters has been made according to the available data on mine operation, including the records of drainage contours for the periods when clear trends (growth or decline) of water inflow change were identified (Fig. 2), and the records of groundwater level observations.

An analysis of water inflow evolution during coalmining made with using a developed groundwater flow model enabled drawing the following conclusions.

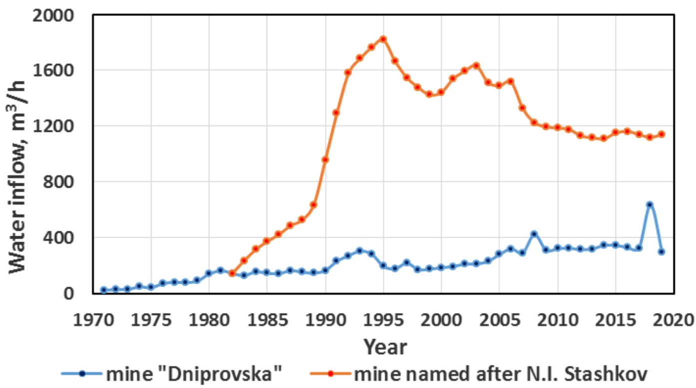

Fig. 2. Evolution of mine water inflows to the mine named after Stashkov and mine "Dniprovska" in time

1. The actual values of water inflow can be balanced in the model only in the presence of transit zones of enhanced permeability if the transmissivity of intact coal seams varies at 0,5-1,5 m^2/days. In the coalfield, the zones of enhanced permeability are located in the sandstone strata of thickness up to 30-50 m and transmissivity up to 30 m^2/day; these formations lie in the top of the coal seam C_5 contacting to water-abundant cover sediments. According to the lithological-facial analysis, these are alluvial sandstones in early carboniferous paleochannels and the patterns of their areal extension correlate with the zones of increased water seepages occurred during mining operations.

2. The water inflow to the mine depends by 70% on the resources withdrawn from the Meso-Cenozoic sediments, and the rest of water is accounted for the capacitive reserves in carboniferous rocks. As a result, a depression funnel up to 30 m depth appeared in the shallow Buchak aquifer, which led to changing groundwater flow directions near the Samara River. Under disturbed hydrogeological conditions, the river is transformed from the discharge zone into a recharge source for cover sediments and, consequently, provides an additional water source for flooding underground excavations.

3. Shifting the depression funnel in the Buchak aquifer correlates in time with the commissioning of coal seams in the direction of the zone where coal seams contact the sediment cover.

4. The maximum inflow of the Samara river runoff to the mine was recorded in course of mining of the C_5 coal seam under the river floodplain 200 m from the riverbed when approaching to the borders of safe mining. In this zone, the maximum water inflow to the coal seam of 822 m^3/h was provided owing to the river runoff of 570 m^3/h or 69% in 1994. 25 years later, the river runoff contribution to the outflow in the eastern coalfield part with the minimum thickness of subsoil sandstones far from the contact of the coal seam and Buchak-Kyiv sediments decreased to 63 m^3/h or 9% (Table 1).

5. As known, the water inflow into a mine with coal seams directly contacting an aquifer does not correlate to the increase in the area of mining operations. In the areas mined out by roof collapse the permeability of carboniferous sediments varies in time. According to our estimations [16] its value increases 10-15 times after rocks were

affected, then it is reducing by half in 5-10 years, and is approaching the initial permeability in 15-20 years.

The hydrodynamic conditions of the seam C_5 were assumed as the initial ones to predict rising groundwater level during flooding the mine. When mining the seam headway in 1994-1995 the maximum water inflows up to 650 m^3/h were recorded. According to the mining plans, in the last simulated period two coalfields have been connected by the cross-drifts at the horizon 300 m through a tectonic fault zone with a displacement amplitude up to 40 m, with no water occurrences having recorded during penetration in tectonically disturbed rocks. The water inflow to the stopes of the major and adjacent mining areas is evaluated by the model at 564 m^3/h (measured value 551 m^3/h).

Table 1

Temporal evolution of water inflow to the coal seam C_5

Year	1994	1998	2006	2015	2019
Total inflow to the mine, m^3/h	822	463	590	452	655
Inflow to the mine from the Buchak aquifer due to river runoff, m^3/h	570	250	159	76	63

After calibration, the modelled groundwater table in the Buchak aquifer was in good agreement with the available data of long-term groundwater monitoring records; the water inflows was also in good agreement with the measured values. This allowed to validate the groundwater flow model with the overall agreement up to 88% for all calculation layers, which was the ground to proceed with the predictive assessments of different flooding scenarios.

Modelling the mine flooding options

Option 1 means complete flooding of the mine named after N.I. Stashkov while keeping the inter-mine pillars. Modelling this option has been performed by disabling the first kind boundary condition in all calculation layers. The rate of flooding the mined out rocks and ground water rebound were found somewhat different. The mechanically affected rocks within the underground workings in the seams C_{10}, C_8, C_6 are expected to be flooded in a first year, since the mining has been shut down there long ago and the groundwater level has been partially restored to the initial elevation. As distinct from that, flooding the shafts and rebound of the ground and mine water levels in the seams C_5, C_4 is expected to take three years.

The groundwater level in carboniferous rocks will raise both in the main and additional parts of the coalfield above the C_5 seam. One month after flooding begins the gravitational capacity of the adjacent mining area will be exhausted and the level will be restoring under the elastic flow mode for three years (Fig. 3). Along with this, for the final period of flooding, a groundwater depression funnel will continue to remain in the adjacent area due to mining the C_{10} seam and draining the main haulage roads along the C_8 seam of the "Dniprovska" mine.

The hydraulic response of the Buchak aquifer to shut off the mine water drainage at the mine named after N.I. Stashkov manifests itself in groundwater level restoration during three years with the appearance of flooded zones along the Samara River. At the same time, the depression pit will be shifted to the "Dniprovska" mine with a maximum drawdown of 11.3 m (Fig. 4).

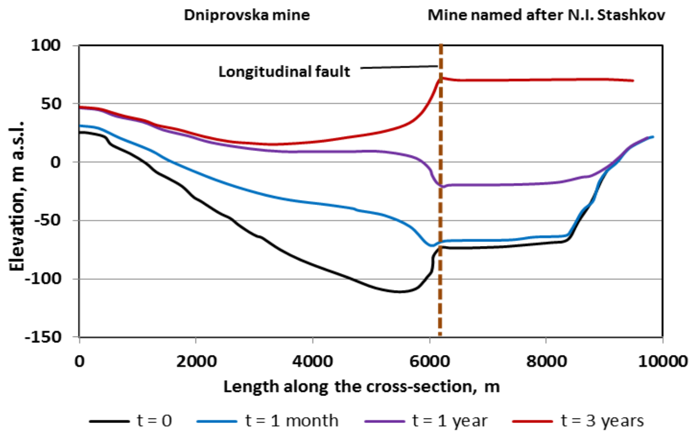

Fig. 3. Predicted groundwater level in the coalfield along the cross-section in Fig. 1 after drainage shut down at the moment $t = 0$

The rising groundwater level within the mine named after N.I. Stashkov will result in the increase of water inflow to the "Dniprovska" mine by 50 m^3/h. The expected water inflows when mining the C_5 seam at the "Dniprovska" mine after 2025 will vary from 80 m^3/h to 156 m^3/h depending on the intensity of mining operations.

The zone of potential flooding and waterlogging remains, as in the pre-operational period, within the Samara River floodplain where the topographic elevations of the ground surface are below +75 m a.s.l. These areas located outside the zone where rocks were mechanically affected by mining are remaining untouched.

Option 2 means flooding the mine named after N.I. Stashkov with maintaining the existing underground facilities of main drainage. On the one hand, keeping the drainage facilities on the horizon 225 m (–115 m a.s.l.) would enable lowering the groundwater level in cover sediments within the area of potential flooding typical for the pre-operational period by 4 m. On the other hand, this option implies keeping the existing design of pumping out mineralized mine water in the long run and its further discharge into the Samara River at the rate of 5,87 million m^3 per year, which should be considered environmentally unacceptable.

Option 3 means closing the mine with maintaining the drainage by submersible pumps installed in the auxiliary shaft. To ensure the environmental safety of shallow aquifers including prevention from waterlogging and keep the quality of groundwater and surface water it is necessary to maintain the mine water level below the bottom of cover sediments within the area of active mining.

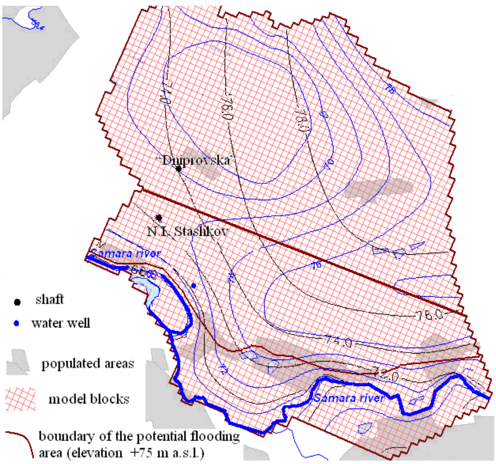

Fig. 4. Groundwater level in cover sediments in the pre-operational period (blue lines) and predicted for the period after mine flooding (black lines)

To maintain the level of mine water and groundwater in carboniferous rocks within mined out zones at the safe elevation of +20 m a.s.l. - that is the minimum elevation of the cover sediment bottom - three sub-options of pumping by submersible pumps were examined numerically. It was suggested that the mine water level in the shaft could be maintained with the flow rate of 300-350 m^3/h at three different positions above the sea level, namely +10 m, –10 m, and –30 m.

Comparison of responses of the aquifer in cover sediments to submersible pump operation shows that the radius of influence in all pump positions is much smaller than in option 2; the drawdown of the groundwater level in the zone of potential flooding is expected to vary from 0.1 m to 1.2 m. In this case, the area of potential flooding can be reduced at most by 15%.

The use of submersible pumps at the pumping level equivalent to the mining horizons 225 m (-115 m a.s.l.) or 300 m (-190 m a.s.l.) requires free water by-pass from the above horizons. Employing these drainage sub-options will maintain the hydrogeological conditions formed in the Samara River floodplain by 2020, i.e. the lowered groundwater level with respect to the pre-operational period. At the same time, these sub-options would require long-term drainage of mineralized mine water followed by its discharge to the Samara River, which is also environmentally unacceptable as in option 2 considered above.

Option 4 is an alternative option to protect the floodplain from flooding by installation of a water intake in the sandstone aquifer of high permeability in the paleochannel.

The option parameters including the water intake location were evaluated in course of model calibration taking into account the experience of mining the C_5 coal seam in the close vicinity of the zone where the seam contacts the cover sediments under the Samara riverbed. In 1994-1995, abnormally high inflows into the mine of 1817 m^3/h of which 649 m^3/h in the C_5 seam have been recorded, with the drawdown in the Buchak aquifer of 30 m reaching the aquifer bottom. In that time, the sandstone paleochannel in the cover sediments above the C_5 seam with a thickness of 14-22 m was the additional zone of groundwater transit from the shallow aquifer.

Identifying the causes and factors of abnormal water inflows allowed to recommend using the disturbed zone of sandstones under the riverbed in the interval of absolute elevations from –55 m a.s.l. to –50 m a.s.l. as an option of reducing the natural areas of flooding after mine closure. The proposed well location is shown in Fig. 4.

A sufficient drawdown in the zone of potential flooding up to 2 m can be reached by pumping water at the flow rate of 150 m^3/h or 3600 m^3/d. Regarding to the rate of mine water level rebound evaluated by modelling the water intake should be commissioned in 6 months after flooding the mine begins. The average mineralization of pumped water estimated by the calculated water balance is expected to be of 2 g/dm^3; this water can be used as process water.

This option of environmental protection is the most appropriate with the synchronous monitoring of groundwater and mine water levels and gradual increase of the flow rate *Q* according to the proposed correlation *f*(*S*) depending on drawdown *S* (Fig. 5); this enables using technical water for general use of the mining company and simultaneously draining 80-90% of the flooded area.

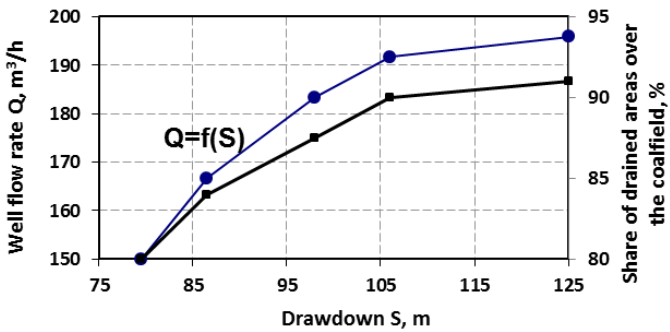

Fig. 5. Recommended flow rate of the water intake depending on the drawdown and estimation of its draining effect

Conclusions and recommendations

The study explored the temporal and spatial patterns of groundwater flow in the most watered mine in Ukraine named after N.I. Stashkov (Western Donbas) in order to optimize environment engineering measures to be apply during its forthcoming closure and the post-mining period.

Numerical modelling showed that the groundwater level rebound in carboniferous and cover sediments is expected to develop in the areas of potential flooding and waterlogging within the Samara River floodplain existed in the pre-operational period; these areas were not mined out and located outside the zone of mechanical impact on rocks.

Having applied the calibrated model we assessed the hydrodynamic conditions after the mine closure and compared three options to mitigate the environmental impacts of flooding. The use of submersible pumps to dewater the shaft above the horizon 140 m was found to be low effective in terms of reducing the area of flooding because of pump location on the periphery of the coalfield. Installing the drainage on the horizon of 225 m would allow approaching the hydrogeological conditions that existed in the pre-mining period, but it may require long-term pumping of mineralized mine water and its discharge into the Samara River, which is not environmentally accepted.

Based on the results of inverse modelling for the period of abnormally high water inflows into the mine, the option of installing the groundwater intake has been recommended. It allows using the water permeable sandstone stratum in the paleochannel in the top of the C_5 coal seam as a transit zone of high permeability to prevent the Samara river floodplain from flooding.

The results of ground water flow modelling and hydrodynamic assessments on the consequences of flooding the mine are the basis of the closure design being developed for the mine named after N.I. Stashkov.

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RESOURCE-SAVING TECHNOLOGIES OF RAW-MATERIAL BASE DEVELOPMENT IN MINERAL MINING AND PROCESSING

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Chief editor **Zinovii MALANCHUK,**
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Co-editor . **Roland MORARU,**
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Deputy chief editor **Serhii CHUKHAREV,**
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