

**TOPICAL ISSUES OF RESOURCE-
SAVING TECHNOLOGIES IN
MINERAL MINING AND
PROCESSING**

Multi-authored monograph

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The multi-authored monograph deals with urgent problems of introducing resource-saving technologies of mineral mining and processing and finds ways to solve them.

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 multi-authored monograph "Topical issues of resource-saving technologies in mineral mining and processing".

Nowadays, development and introduction of innovative resource-saving technologies is a key factor of increasing economic efficiency and satisfying growing needs for resources.

The multi-authored monograph is aimed at analyzing available resource-saving technologies of mineral mining and processing and finding ways to solve recurrent problems.

The monograph articles deal with some issues of applying resource-saving technologies to mining and processing, outline directions for resource-saving development and offer technological solutions of current problems.

The multi-authored monograph is intended for a broad audience of specialists engaged in solving problems of the mining and processing industry.

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DEVELOPMENT OF RESOURCE-SAVING TECHNOLOGY FOR MINING OF RESERVES BELOW THE PIT BOTTOM

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Abstract

The paper presents ways to solve the problem of mining of the reserves below the pit bottom at the mines that transitioned from open-pit to the underground mining and increasing the efficiency of mining with the use of the mining methods with backfilling the mined-out areas.

The authors of the paper have proposed a technology of backfilling of the mined-out areas with the mixtures of difference strength depending on the mining sequence of the stopes and the vertical open exposure.

In order to obtain filling mixtures that provide the necessary mobility and strength for the Maikain mine conditions and reduce costs of the materials, it was advised to use local flotation tailings from the tailing dump and screenings of the crusher, and as a binder - a mixture of portland cement and lime.

It was established that the addition of a plasticizer PozzolithMR 55 at a dosage of 1.0 kg/m^3 of the mixture, improves its mobility by 20% and strength by 12-15%.

Blasting technology at contact with the filling mass was proposed considering the discharge zone. Portions of the blast holes directed to the filled stopes in their bottom part can be undercharged to the length equal to the discharge zone. For example, for the blastholes with the length from 6m to 13m, the length of the undercharge should be from 0.4 m to 1.0 m.

Introduction

Relevance. There are more than 2 thousand deposits in the world today, which are transitioning or will transition from open-pit to underground mining. In the last 10 years, the number of such deposits has increased almost 1.5 times, mainly because pits are reaching their ultimate depths and the remaining reserves of deeper horizons can only be mined using underground mining methods. During the transition from open pit to underground, there is an ore pillar left between the two operations. Subsequent mining of this pillar will inevitably lead not only to heavy losses and ore dilution, but also a variety of processes and issues, which are practically impossible to predict and calculate.

At the same time, there are reserves left at the bottom and walls of the pit, which are not feasible to mine with the open pit.

The analysis have shown that the most effective way of mining of the reserves below the pit bottom is by using the mining methods with backfill.

Objective- Development of the technology of mining of the reserves below the pit bottom and in the pit walls, providing the most complete extraction of reserves, safety, economic and ecological efficiency of mining operations.

The scientific novelty of the work:

- dependence of the necessary strength of backfilling mass from the height of the stope opening and the stage of the stope mining was established;

- influence of additives-plasticizers on the rheological characteristics of the hardening mixtures was obtained;

- undercharged portion of the drill holes at their bottom in the blast hole rings at the contact with backfilled stopes was determined to depend on the length of the discharge zone and uniformity of the explosives distribution. Undercharging will reduce the ore dilution and damaging of the backfill.

In carrying out this work a complex method of research was used. It includes analysis and scientific generalization of research and technical information and practice of mining industry, experimental studies, statistical processing and calculations, economical evaluation of the technical decisions taken.

Mining with the use of backfill

There is a tendency of a significant increase of the underground mining methods with backfill in the world. This is primarily due to the need to increase the completeness of the extraction of minerals as well as reduce the dilution with waste rock. Practice shows that underground mining methods with backfill have average losses of ore is the range of 3-5%, and dilution does not exceed 7-10 %. The relatively high cost of mining with such method is compensated by the completeness of the extraction of minerals and improvement of their quality.

Using methods with backfilling can successfully solve a number of different in nature complex problems to ensure favorable conditions for mining operations. The most important out of them are: improving the safety of mining operations, preservation of the earth's surface, improving the ventilation of mine workings, environmental impact.

It should be noted that another trend in improving the technology of ore mining using methods with backfilling. Previously used waste rock filling was characterized by a large expenditure of manual labor; large backfill settling of the constructed artificial arrays, increased dust formation and this method was not widespread. In recent years, the most popular method is cemented backfill. Prepared on surface in specialized backfill plant cemented fill is delivered to the stope with the help of high-performance pipeline transport in gravity mode. Backfilling operations are easy to automate, provide stability of the composition of the filling mixture and the artificial filling array.

In the Republic of Kazakhstan almost all of the mining enterprises, which mine nonferrous metal ores by underground methods use systems with cemented fill. These enterprises are JSC Kazzinc – Ridder and Zyryanovsk mining and processing complexes, enterprises of LLP “East region” – Zhezkentski GOK, Artemyevsky mine, etc., these enterprises have high-performance filling plants with capacities of 100-200 m³/h. These enterprises use efficient methods of mining with the use of drilling and load-haul-dump vehicles.

The main types of mining method with cemented fill are currently:

- sub-level stoping;
- cut-and-fill.

Minig method of sub-level stoping with cemented backfill is used to mine flat and steeply dipping orebodies with high strength of host rock. These mining methods support many face on the level allowing to use the powerful self-propelled drilling and load-haul-dump equipment for high productivities.

In principle, the mining process by such method represents the following chain. Ore reserves within the level is mined out in stopes in several stages, leaving pillars of ore and filling the space with cemented fill. Subsequently, the ore pillars (stopes of the following phases) are excavated under the protection of the artificial pillars. Unified parameters of the stopes, which are mined out in different stages, allows to apply unified technology of mining and contributes to the improvement of technical and economic indicators for the system.

Below is a typical design of the sub-level stoping mining method with backfill, which is used at the Orlovsky mine LLP "Vostoksvetmet". Depending on the nature of the ore and the host rock, the stoping can be carried out to the full height of the level, or the level is divided into 2-4 sub-levels and the stoping is carried out in sequence

The design of the system is as follows (figure 1). The ore body is divided into sections up to 60 m long along the strike. Vertically, this section (block) is divided into three sub-levels by underground workings. The height of the sublevel is 17 m along the strike, the block is divided into stopes with a width of 10 m. Across the strike, depending on the thickness of the ore body block is divided into several panels, so that the ratio of the width of the stope to the size of the panel does not exceed 1:2.

Preparation of the block consists in ramp development at an angle of 8-12°, the ventilation raise, the transportation drift and of them at an angle of 5...6° cross-cuts. From the level of the haulage horizon the ore pass is sank to sublevels. Each cross-cut is connected with ventilation-backfill development to the backfill cross-cuts and the

ventilation drifts. Loading drives, drilling drifts, cut-out raises are developed after. Extraction begins with the cutting of the cut-off slots and with the subsequent ore breaking into this slot with the ring drill hole charges. The diameter of the drill holes is 56-70 mm. The development workings, depending on the stability of the enclosing rock, are supported with bolts with shotcrete or steel arches SVP-27 with special profile. Concrete fill into the stopes is supplied by pipeline to backfill the workings from the upper sublevels.

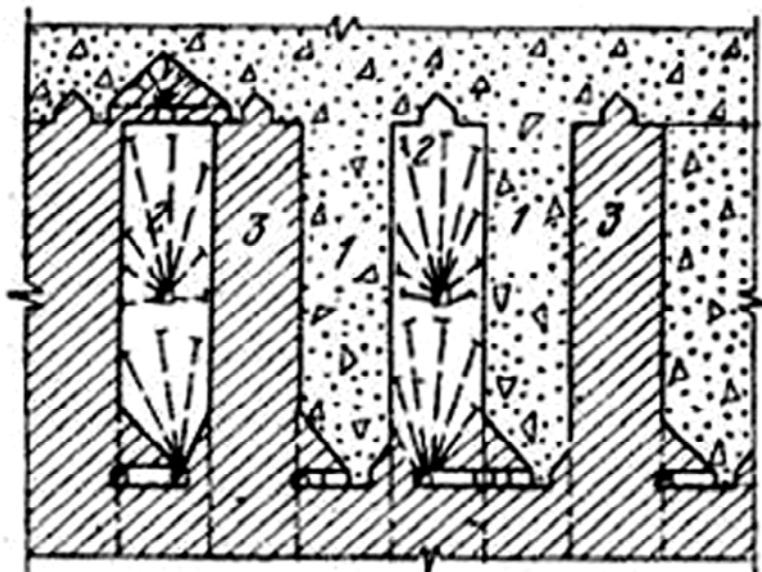


Fig. 1. Sub-level stoping with cemented backfill:
1-filled stopes; 2-stopes, drilled with the rings of blast holes; 3-ore

For secondary filling of the stopes and releasing the air in the process of filing 2-3 drill holes are drilled from the backfilling drift into the stope. The equipment at the drifting works is the same as during the mining process.

During the production mining, drilling of wells is performed by machines of the KBU-50 or PBU-80 types. Ore transportation is carried out by load-haul-dump vehicle "Toro-200" type. Hole blasting is sectional with electric detonators with a delay in the queue of 25-50 ms.

Ventilation of the drilling and stoping areas in the block is carried out with the air provided by the main ventilation fans of the mine. Simultaneous mining of adjacent panels is allowed with a space of two stopes between them, and simultaneous mining of stopes in two adjacent sublevels should be done with the lead of the upper stopes in relation to the lower ones by at least 30 m.

Cut-and-fill mining method with backfil is used for mining of highly valuable ores of steeply dipping vein, bedded, lenticular and other deposits. The main factor determining their use is reduce losses of the reserves of minerals and prevention of the deterioration of its quality.

Depending on the stability of the ore and host rocks cut-and-fill is used in the bottom-up and top-down orders.

With top-down cut-and-fill method (figure 2), the ore body length along the strike of up to 300 m is prepared with the ramp development, developed in the hanging side of the rock at an angle of 8-12° and a series of ore passes, developed from the cross-cuts (of ore or rock) to the level of the workings of the first layer in sublevel . In foot wall of the upper opening of the horizon or between sublevel ventilation drift is prepared. For each layer, the ramp development is connected with ore passes and cross-cuts at an angle of 3...5° to the middle of the ore body and then to the footwall at an angle of -3...5°. At the end of the cross-cut a ventilation raise is connected to the block ventilation workings of the upper (mined-out) layer.

Cutting works in the layer consist in penetration in both directions from the cross-cut of ore transport drifts along the strike of the ore body at an angle of 3 ... 5°. Transport drifts on each layer are connected with the ore passes and at the end with ventilation workings to the ventilation raises. The extraction in the block consists of drifting of the development at an angle of 3..5° in the layer with a section of 4X3 m² from the transport drifts towards the hanging or foot wall side. On the basis of the development, there is a reinforced mesh with bolts.

The equipment used for all works in the preparation and development of blocks is the same: drill carriages (of the type Minimatic-Universal) and loading and hauling machines of the Toro-200 type. To charge the blastholes in the faces, pneumatic chargers ZP-2 and ZP-12 were adopted.

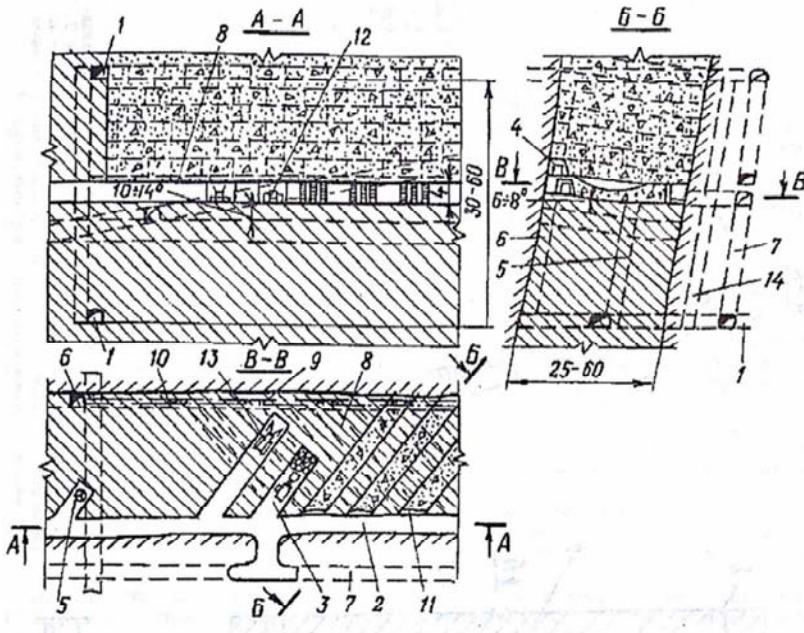


Fig. 2. Top-down cut-and-fill mining method:

- 1-cross-cut; 2-level-drift; 3-muck-bay; 4-backfill-ventilation drift; 5-orepass;
- 6-backfill-ventilation raise; 7-decline; 8-ventilation channel; 9-ventilation tube;
- 10-backfill pipe; 11-concrete stopping; 12-LHD; 13-mobile drill;
- 14-ventilation raise

Ventilation during the period of cleaning works in the block is carried out according to the following scheme. Fresh air in the block enters the layer cross-cuts and into the transportation drifts from the block decline. Contaminated air due to general depression is returned by ventilation raises to collecting ventilation passes and further along the level development is directed to the ventilation shaft. Supply of fresh air is carried out by fans of BM-8 type through slotted pipelines 800 mm in diameter.

Preparatory, cutting and cleaning workings in the underlying layer must be carried out with a shift of half of their width. The backfilling mixture on the layer can be fed through a pipeline laid along both the layer development (like cross-cut) or by a pipeline

laid in the service holes bored from overlying layer workings to the layer being worked.

The ascending order of the excavation of layers (Figure 3) is used to develop comparatively stable ores. Depending on the thickness of the ore bodies, the degree of stability of ores and rocks, can be different. For example, there are several variants with a ceil-layer excavation and their temporary underfill, with the development of mining operations in a layer over a large area etc.

The schemes of preparation of deposits and the delivery of ore in the boundaries of the panel are interlinked. There are two schemes of preparation with the delivery of ore: according to the cutting queue to the boundaries of the panel and along the queue to the flanks of the panel. The second scheme is advisable to use when the thickness of the ore body is less than 12 m. For these schemes, there are three main levels: a haulage (in rock, rail transport), transport (in ore, trackless transport) and ventilation-backfill (in rock). On the flanks of the panel transportation drifts for the loaded and empty transports, and at the borders - hauling drifts for the opening of block ore passes. On the route of haulage drifts, there are panel ore passes developed every 15-20 m. The panel ventilation and backfill drifts are developed every 50-60 m at the borders or in the center of the panel. Diagonal declines connecting the lower and upper layer drifts of the panels are developed every 70 m.

The extraction of the lenses by layers from the bottom to the top is carried out in three stages: working off of the lower (foot), main and upper layers. The lower stripping layer is mined out in three steps 3-4 m in height and 8 m in width.

After mining off the lower layer, the height of the following main layer is adjusted to 6 m. This is the most productive stage of mining works. Here a number of basic technological operations are combined, the blasting of the ore is performed by steeply inclined (60-70°) blast holes with a clearance height of 6 m. The ore is cleaned by loading and transporting machines.

After the excavation of the layer, the mined out space is filled at a height of 3.5-4 m, and the space of 2.5 m high is used for ventilation of the stops through the main mine depression. The breaking of ore in the next layer begins when the filling strength reaches 1-1.5 MPa.

The aforementioned mining methods allow to extract minerals with minimal losses, without disturbing the earth's surface and providing ventilation of the mining operations without air leaks. However, the use of such mining methods with backfill is also characterized by high costs of production, man power, which requires the development of a resource-saving, efficient technology for mining of ore reserves below the pit bottom and in its walls.

Proposed technology of mining of the reserves below the pit bottom

Research object is Maykain gold mine, which is situated in Bayanaulsky area of Pavlodar oblast, and a typical mine with an open-pit to underground transition.

Given the geological conditions of the deposit, the proposal was to use sublevel stoping underground mining method with backfill. In this case the ore body is divided into sections along the strike with the length of 60 m. Vertically blocks are divided into three sublevels with the height of 17 m. Along the strike blocks are divided into 10 m wide stopes, across the strike they are divided depending on the thickness of the ore, so that the ratio of the width of the stope to the thickness does not exceed the value of 1:2.

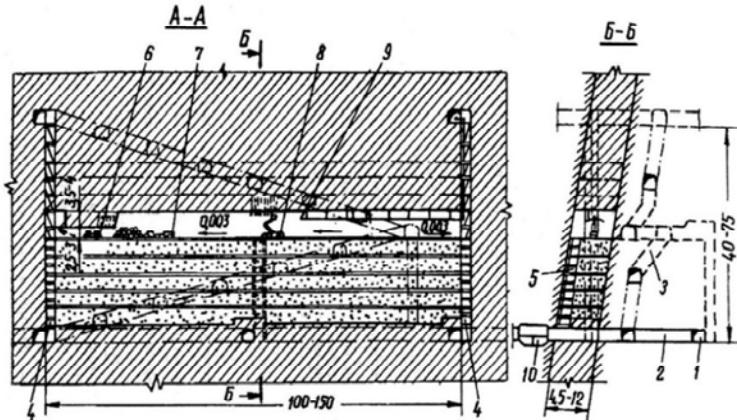


Fig. 3. Bottom-up cut-and-fill mining method:
 1-transport drift; 2-cross cut; 3-decline; 4-ventilation-manway raise; 5-dewatering
 raise; 6-drill rig; 7-loading-hauling dump; 8-inspection and scaling equipment;
 9-backfill pipe; 10-sump

Preparation of the block consists of drifting the decline at an angle 8-12°, ventilation-backfill raise, transportation drifts and crosscuts at an angle of 5-6° for loading and hauling. Ore pass is sunk from the haulage level to the sublevels. Each crosscut is connected with the ventilation-backfill crosscuts, which are developed from the ventilation-backfill drifts. Further, loading bays, drill drifts, slot raises are developed. Production starts with cutting a slot raise and subsequent blasting of ore using blast hole charges. The diameter of the holes is 56-70 mm. Operating development is supported with bolts with shotcrete or steel arches of special profile depending on the stability of the host rock. Backfill is pumped into the mined out stopes using pipelines laid in backfilled excavations of the upper sublevel.

There are several stages of the stopes proposed for mining of the reserves below the pit bottom:

- first stage – mining of the stope reserves under the protection of block pillars, where the artificial pillar (filled stope) is loaded almost exclusively by its own weight;
- second stage - the stope is mined with full exposure of an artificial pillar (filled stope) on one side;
- third stage - the exposure of an artificial pillar on both sides.

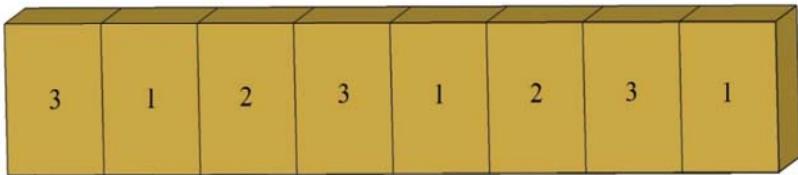


Fig. 4. Mining of the primary stopes

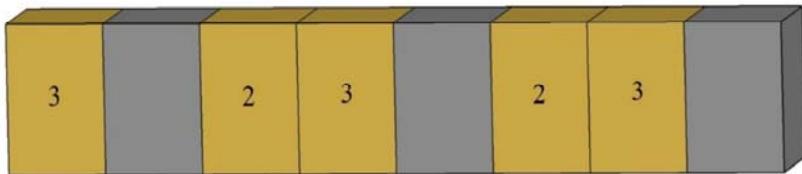


Fig. 5. Mining of the stopes with backfilled stope on one side

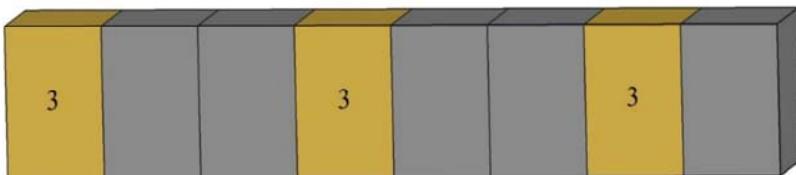


Fig. 6. Mining of the stopes with the backfill on both sides

The difference of this technology is that the stopes of different stage are filled with the mixture of different strength.

Расчет нормативной прочности искусственного закладочного массива был проведен по методике [3,4].

Based on studies, considering the geological conditions and the parameters of the stopes, it was determined that for the conditions of "Maikain" mine, the required strength of the artificial pillar depending on the stage of the stope and the area of its vertical exposure varies within 2,6-4,25 MPa and amounts to:

- stage I stopes - 2.6 MPa;
- stage II stopes - 3.0 MPa;
- stage III stopes - 4.25 MPa.

Necessary strength of the artificial pillar depending on the stability conditions with a specific vertical exposure (Table 1)

Table 1
Dependence of the required backfill strength on the stope vertical exposure

Stope vertical exposure (mean value), m	Required strength, MPa
10.5 (7.5)	1.2
10-15 (12.5)	1.5
15-20 (17.5)	2.0
20-30 (25)	2.5
30-40 (35)	3.0
greater than 40 (45)	4.0

Selection of the rational composition of the backfilling mixture

In order to prepare a mixture with the above properties, which would give 12 to 14 cm mixture mobility and shear stress of 20 to 25 Pa, allowing its gravity transportation and required strength there

were experiments conducted. To reduce the cost of preparation, it has been justified and suggested to mainly use locally available materials for the backfill. To do so, a research has been conducted on: flotation tailings; screenings of the crusher; rock from the drifting works, limestone of the "Keregetas" quarry.

The studies were carried out according to the following procedure. Theoretic calculation of the composition of the mixture following the criterion:

$$\Pi/\rho_{\Pi} + U/\rho_u + X/\rho_x + O/\rho_o + B/\rho_b = 1, \quad (1)$$

where Π - cement consumption, kg/m³; U – lime consumption, kg/m³; X – flotation tailings consumption, kg/m³; O – consumption of screenings of the crusher, kg/m³; B – water consumption, kg/m³; ρ_{Π} , ρ_u , ρ_x , ρ_o , ρ_b – respectively, the density of cement, lime, flotation tailings, screenings and water.

Given that currently the most common method of preparation of a back-filling mixture is a ball mill activation, it was adopted as a basis.

The laboratory mill was loaded with the components of the mixture and mixed for 5 minutes. Obtained mixture was tested for the mobility in the construction cone and the shear stresses on the Rebinder-Weiler instrument. Mixtures that met the mobility and shear stress requirements specified above were put in molds and after hardening were tested for strength at the age of 14, 28 and 90 days.

The composition of the tested mixtures differed in the ratio of the components, which subsequently made it possible to establish the dependence of the mobility, shear stress, and dynamics of the strength on the composition of the mixture.

Taking into account that the grains of the cement have a rather coarse grinding, and when they are milled in a ball mill, their regrind occurs, studies have been carried out on the effect of the fineness of cement grinding on the strength of the backfill, and it has been established experimentally that re-grinding of cement allows more complete utilization of its activity and, as a result, increases the strength of the resulting artificial pillar.

Backfill strength was determined in the laboratory by crushing test-samples using a hydraulic press (Fig. 2). There were three samples prepared for each test. Standard terms of testing were 28 and

90 days. Size of the mold for the fill mixtures was $100 \times 100 \times 100$ mm.

Results of experiments have shown that waste rock is not suitable for the preparation of the mixture, because the rock is not grindable in the mill and the mixture does not keep the water.

For conditions "Maikain" mine, it is proposed to use dry flotation tailings from the dump and screenings of the crusher.

Further studies have been directed to increase the strength, mobility of filling mixture and decrease material consumption. For these purposes, we have conducted a study on the effects of additives-plasticizers on the rheological properties of the mixtures.

There is a variety of different chemical additives produced worldwide and domestically. We have chosen PozzolothMR 25, 55 and 100 Pozzoloth XR, which are liquid additives used to improve fluidity of the mixture. They contain no chlorine and used for obtaining homogeneous and high quality concretes. Studies were conducted using three kinds of additives, wherein the amount of cement was 140 kg/m^3 , tailings in the range of $1193\text{-}1389 \text{ kg/m}^3$; water $439\text{-}511 \text{ kg/m}^3$. The quantity of additives ranged from $0,5 \text{ kg/m}^3$ to 1.5 kg/m^3 with an increment of 0.5 kg/m^3 .

The mobility of the mixture was initially determined using a standard cone. The process of determination of the mobility is very simple: the cone is submerged into the mixture and the depth of the submergence determines the mobility.

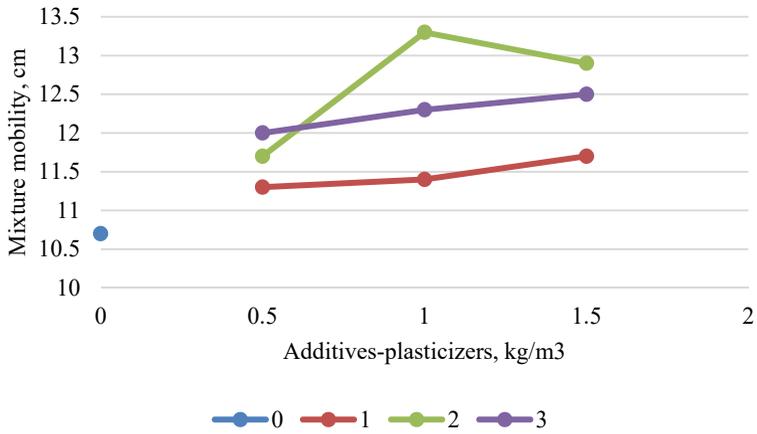
The results of such tests with and without the plasticizers are represented on the Figure 7.

Mobility of the mixture was also determined by flow time of 2 liters of mixture out of vibratory crucible.

Data processing resulted in determining the dependence of the changes of the mobility of the filling mixture and its outflow time from the vibratory crucible when the additives-plasticizers are used.

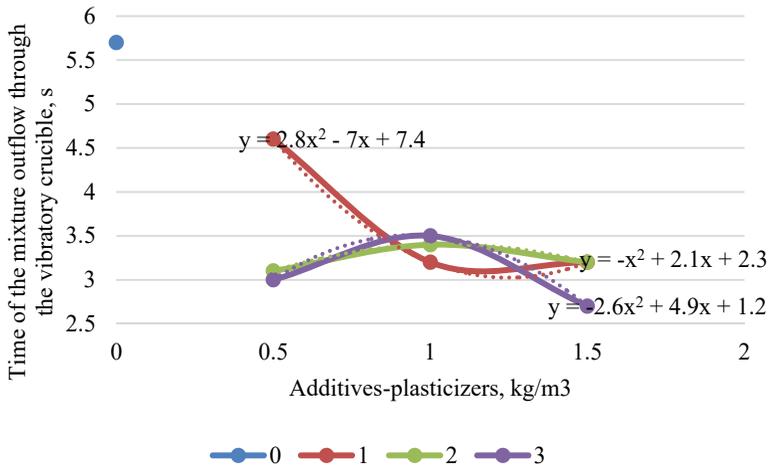
Figure 8 represents the results of such tests.

Increase in the mobility of the mixture due to the introduction of chemical additives can reduce the water content of the mixture, and this in turn leads to an increase in the strength of the hardened backfill massif.



0 - without additives; 1, 2, 3 - with additives-plasticizers, respectively, PozzolithMR 25, PozzolithMR 55 and Pozzolith 100 XR

Fig. 7. Change in filling mixture mobility with additives-plasticizers (composition of the mixture: cement - 140 kg/m³, tailings - 1288 kg/m³, and water - 476 kg/m³, the ratio S:L in the tailings - 73:27, calculated density of the mixture -1904 g/l)



0 - without additives; 1, 2, 3 - with additives-plasticizers, respectively, PozzolithMR 25, PozzolithMR 55 and Pozzolith 100 XR

Fig. 8. Chang in the time of outflow of the filling mixture from the vibratory crucible

For the purpose of analyzing the change in the strength of the resulting backfill massif several samples were tested for the standard uniaxial compression at the age of 14 and 28 days. (Figure 9).

As a result of the research following mixture compositions were proposed for the conditions of the Maikain mine (Table 2).

At the end of the laboratory studies, it was concluded that:

- the most appropriate additive to be used at the Maikain mine is chemical plasticizer PozzolithMR 55;
- use of the additive in the composition of filling mixture enables (depending on the quantity) to increase the mobility of the mixture for up to 20% in the laboratory conditions;
- strength of the backfill at the age of 28 days can be increased by 10-15% or in the case when the strength properties of the filling mass are kept at the same level, it can reduce the consumption of expensive cement.

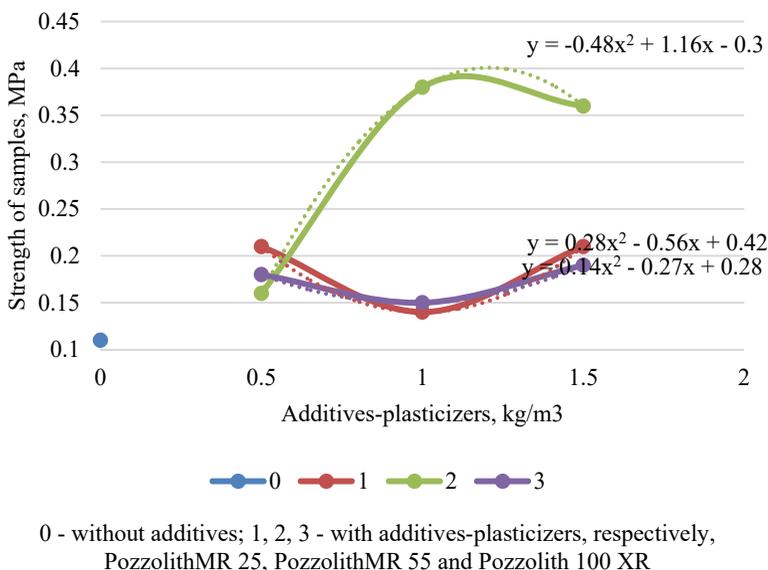


Fig. 9. Change in the strength of samples of the backing mixture with the use of plasticizer additives at the age of 14 days

Table 2

Recommended mixture compositions

Mixture code	Specific consumptions, kg/m ³							Mixture density, kg/m ³	Cube strength		Estimated strength in the stope, R90, MPa
	Binder			Filler			Water		MPa		
	Total	Including		Total	Including				R28	R90	
		cement	lime		tailings	screenings					
M301	240	240	-	1240	-	1240	500	1990	1,1	2,85	3,7
M302	200	200	-	1280	-	1280	450	1980	0,76	1,9	2,5
M303	150	150	-	1310	-	1310	450	1980	0,4	1,1	1,5
M304	200	200	-	1330	1050	285	450	1985	1,3	3,5	4,0
M305	150	150	-	1350	650	700	450	1980	1,5	3,4	3,8
M306	150	150	-	1330	1350	-	500	1980	0,85	1,8	2,3
M307	120	120		1350	1350	-	500	1985	0,4	0,95	1,25
M308	170	160	50	1250	1250	-	550	1990	1,2	3,0	3,8
M309	150	180	70	1280	1280	-	580	1990	1,0	2,4	3,1
M310	170	120	50	1310	-	1310	550	1985	1,25	2,75	3,5
M311	140	480	60	1310	-	1310	550	1985	0,65	1,7	2,1
M312	170	120	50	1300	950	350	550	1980	1,15	3,0	3,3
M313	140	170	70	1350	650	600	550	1980	0,6	1,75	2,2
M314	170	80	90	1350	600	650	550	1980	0,9	2,5	2,85
M315	220	100	120	1250	1250	-	550	1990	1,2	3,4	4,2

Technology of blasting at the ore-backfill contact

As it is known, when using mining methods with backfill and blasting with rings, backfill mass can be damaged by the explosion. Many mines have experienced that ore dilution with backfill material results in a sharp decrease in the recovery of the metals at the processing plant. Sometimes losses due to reduced extraction at the processing plant may greatly exceed the full costs of the backfill.

Stress-strain state of a rock mass in the vicinity to the backfilled stope is formed depending on the direction of the maximum and minimum stresses. On the other hand, during the blasting operations

of the rings at the ore-backfill contact, there is an effect of the stress waves arising during the detonation pressure that destroys both the ore and the backfill mass.

The stress-strain state of rock reduces in the vicinity to the backfill mass. Further away from the backfilled stope, the stress-strain state of the rock increases and reaches its maximum value and then comes down to its natural stress-strain state, and then increases again in the proximity to the drilling drift.

In order to determine the stress distribution along the drill holes placed in the direction of action of the mean stress, computer modelling was conducted using Examine2D software (Fig. 10).

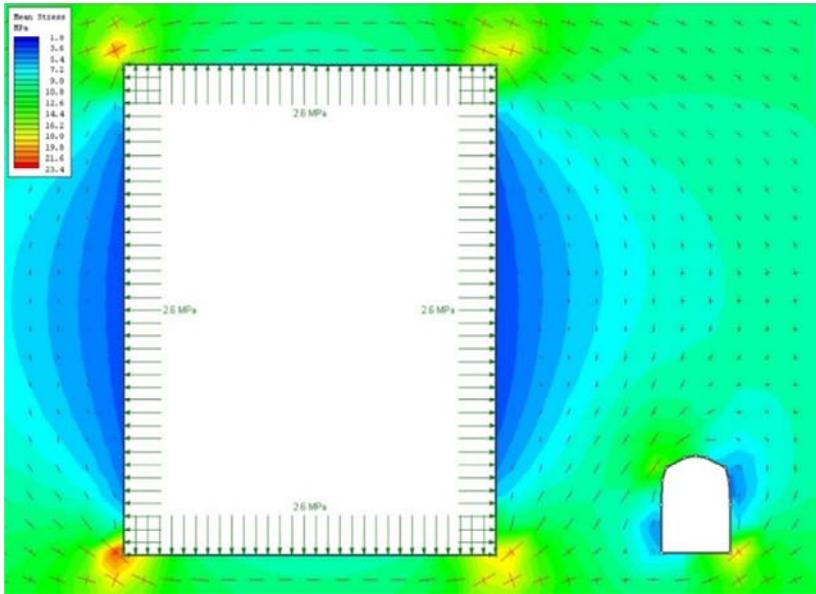


Fig. 10. Stresses around the stope and a drilling drift

Figure 11 reflects mean stresses (discharge area) of the second stage stopes after the first stages stope has been backfilled for the conditions of Maikain mine. As can be seen, lower stresses around the second stage stope are in its middle part, and maximum stresses are at the upper and lower corners of the first stage stope.

As a results, it can be concluded that the undercharge portion of some ring drill holes in their bottom can be determined according to the length of the discharge zone.

A layer of the backfill mass is in the most severe conditions at the contact with the ore. In the process of mining of ore reserves, the contact layer withstands the maximum loads from blasting, and after blasting, when it is left open, there is also a load (pressure) from the rock and the backfill mass itself. The effect of blasting in these specific conditions is usually reflected in the form of rockfalls, falls of the concrete backfill, which as consequence increases dilution of ore, and leads to increased mining safety risks.

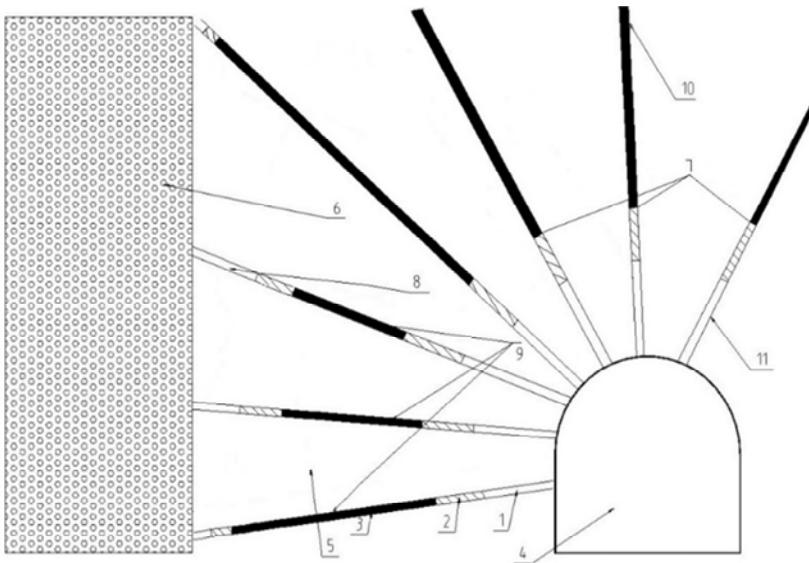


Fig. 11. The circuit arrangement of wells and placing them BB:

- 1 - unloading zone; 2 - zone of stress concentrations; 3 - ore zone; 4-drilling drift;
- 5 - ore body; 6 - backfill mass; 7 – blast holes; 8 - blast hole oriented in the direction of the minimum stress; 9 – blast holes in contact with the backfilled stope;
- 10 - undercharged portions of blast holes in their bottom parts;
- 11 - undercharged portions of blast holes in their collar parts

Obtained results can be represented as a diagram in the figure 12.

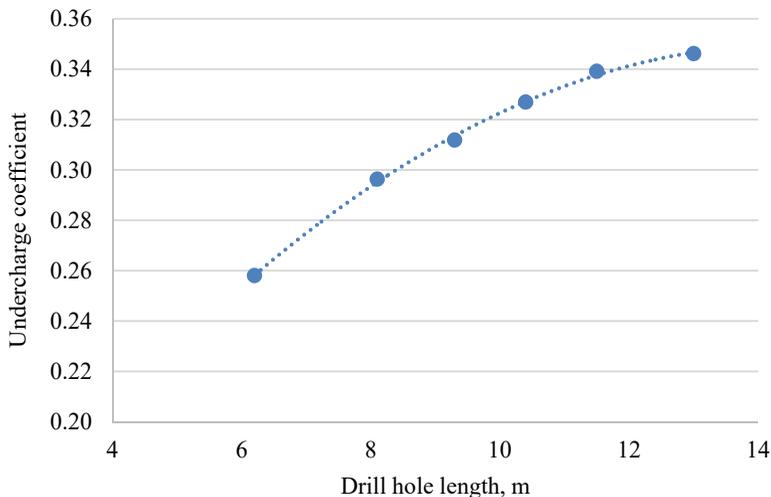


Fig. 12. Dependence of the undercharge coefficient of the drill holes in the direction of the backfilled stope from their length

Statistic interpretation of the dependence of the undercharge coefficient of the drill holes from their length resulted in the formula:

$$K_{\text{нед}} = -0.0013 \times L_{\text{СКВ}}^2 + 0.04 \times L_{\text{СКВ}} + 0.07 \quad (2)$$

$$(6 < L_{\text{СКВ}} < 13)$$

where $L_{\text{СКВ}}$ – drill hole length, m; $K_{\text{нед}}$ – total undercharge.

The proposed technology must perform the following sequence:

Using the data, including physical and mechanical properties of the ore and the surrounding rocks, depth of the stopes and their parameters, parameters of the drill drift, and the stresses around the drift and the stopes. Given the fact that the discharge zone is 0.2-0.25 of the value of the normal stress zone, the length of the undercharge can be determined at the bottom parts of the drill holes.

Thus, determination of the length of the discharge zone considering the stress-strain state allows to reduce the specific consumption of explosives, losses and dilution of ore with the backfill material.

Conclusion

This paper gives a solution to the critical problem of mining of ore reserves below the pit and in the pit walls for the mines with combined (open-pit-underground) mining.

As a result of the research work, the following conclusions can be drawn:

1. The analysis of international experience of mining of ore reserves below the pit and in the pit walls leads to the conclusion that for Maikain mine conditions effective way of mining such ore is proposed using stoping mining methods with backfill in several stages with different strength for every stage.

2. Installed nominal strength of backfill is based on the stage of the stope. The required strength of the backfill for the Maikain mine conditions:

- Stopes of the first stage - 2.6 MPa;
- Stopes of the first stage - 3.0 MPa;
- Stopes of the first stage - 4.25 MPa.

3. Strength of the backfilled stope depends of the vertical exposure and the stage of the stope. For example, vertical the stopes with the height from 5 m to 40 m, the strength of the filling mass varies from 1.2 MPa to 4.0 MPa.

4. For filling mixtures, in order to provide the necessary flexibility and strength of the filling mass for the Maikain mine conditions, as a filler, it is appropriate to use local flotation tailings and screenings of the crusher, and as a binder portland cement and lime.

5. It was established that addition of the plasticizers in the mixture, such as Pozzolith MR 55 at a dosage of 1.0 kg/m^3 of the filling mixture improves mobility by 20% and improves strength of the backfill by 12-15%.

6. Developed technology of mining of the reserves below the pit bottom and in the pit walls, provides a more complete extraction of reserves, safe and efficient mining operations.

7. Blasting technology at contact with the filling mass was proposed considering the discharge zone. Portions of the blast holes directed to the filled stopes in their bottom part can be undercharged to the length equal to the discharge zone. For example, for the

blastholes with the length from 6m to 13m, the length of the undercharge should be from 0.4 m to 1.0 m.

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SUBSTANTIATION OF STABLE CROWN SHAPES IN IRON ORE MINING

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Krivbass rich iron ores are mined by stoping at the depths of 1200-1450 m, preliminary development reaching 1600 m. Increased rock pressure decreases stability of mine workings, in particular stopes and compensating rooms, affecting stability of the most vulnerable element - exposed crowns of underground workings [1-8]. Due to this, the problem of stability of exposed stope and compensatory room crowns gains considerable importance when designing stopes at great depths.

Determining regularities of rock pressure impacts on the crown stability depending on its shapes, mining depths and iron ore hardness, therefore, will facilitate choosing and substantiating crown shapes.

The *Ansys 16.0* based finite element technique is applied to calculate stress and strain as well as triangulation with a 2 m side is performed to build stress and strain diagrams.

The computation process is conducted concurrently, including creation of a stiffness matrix, solution of linear equations, calculation of results when processing with memory sharing and distributing. The component synthesis mode, the analysis of the cyclic symmetry, submodelling techniques facilitate work with large models and systems that represent the stress-strain state of rocks.

The applied technologies parameters and the stoping sequence are considerably impacted by the stress-strain state of rocks, i.e. information on the character and values of active stresses in the

massif, reasons of their changes during stoping enables enhancing the applied flow sheet and developing new ones, choosing optimal parameters and sequence of stoping.

The experiment consists in creating models with horizontal, tent, vaulted and inclined crown shapes the dip of which varies within a wide range.

The values of caved rock pressure on the ore massif P_1 , P_2 and P_3 represent mining conditions of Krivbass deposits and correspond to mining at the depths of 1200, 1450 and 1700 m.

Physical and mechanical properties of the ore and waste ore studied are given in Table 1.

Table 1

Physical and mechanical properties of ore and waste ore

Parameters	Units	Ore				Rock			Waste rock
		$1P$ $f=3-5$	$2P$ $f=4-6$	$3P$ $f=5-7$	$4P$ $f=6-8$	1Π $f=4-6$	2Π $f=5-7$	3Π $f=8-10$	
Young modulus	MPa	22000	25000	28000	32000	22000	33000	40000	5000
Volume weight	kg/m ³	3700	3650	3600	3500	2800	2900	3000	2400
Compressive resistance	MPa	30	40	50	60	45	55	80	5
Ultimate tension stress	MPa	3	4	5	6	4.5	5.5	8	0.3
Poisson ratio	–	0.30	0.28	0.26	0.25	0.26	0.24	0.24	0.25

Table 2 presents values of rock pressure on the ore massif corresponding to mining conditions of Krivbass deposits.

Table 2

Parameters of rock pressure on the ore massif

Parameters	Unit of meas.	P_1	P_2	P_3
Rock pressure on massif: vertical / lateral	MPa	7.0/2.5	8.5/3.0	10.0/3.5

When studying the stress-strain state of the ore massif special attention is paid to shapes of stopes and to shapes of stope crowns in particular, stope sizes being assumed equal for comparison purposes.

The calculated stress fields values (MPa) for various crown shapes are given below.

Principal stress values in stopes with horizontal crowns are distributed according to the classical law of stress distribution in rocks. The maximum compressive stresses are seen in the corners of stopes whereas tension stresses are found in the central part of the crowns.

Being the most dangerous, tension stresses are the most frequent reason for destructive manifestations of rock pressure that affect mining engineering elements.

The value of tension stresses in a stope crown not exceeding the ultimate tension stress of the ore massif, the crown remains stable, especially in ores with low fracturing. The value of tension stresses in a stope crown exceeding the ultimate tension stress of the ore massif, the crown is predictably unstable and tends to form local falls or be completely destroyed.

The maximum stresses values in horizontal crowns of stopes σ_1 depend on the mining depth H_p and the corresponding value of rock pressure various iron ore hardness are described by the polynomial equations:

– for ore hardness $f=4-6$

$$\sigma_1 = 2 \cdot 10^{-6} H_p^2 - 0,0128 H_p + 3,056, \text{ MPa};$$

– for ore hardness $f=5-7$

$$\sigma_1 = 2 \cdot 10^{-6} H_p^2 - 0,0132 H_p + 2,836, \text{ MPa};$$

– for ore hardness $f=6-8$

$$\sigma_1 = 2 \cdot 10^{-6} H_p^2 - 0,0124 H_p + 1,076, \text{ MPa}.$$

The adaptive index of maximum stresses in horizontal crowns of stopes σ_1 depending on the mining depth H_p and various iron ore hardness can be determined by the expression, MPa:

$$\sigma_1 = (-4,4394 \ln(f_p) - 1,6208) \times \\ \times (2,2727 \cdot 10^{-7} H_p^2 - 1,4545 \cdot 10^{-3} H_p + 0,3472).$$

Principal stress values in stopes with tent crowns are distributed in the following way. As the result of crown contour transformations, principal stress inversion is observed.

The stress-strain state of the tent crown massif experiences decreased compressive stress values and no tension stresses.

The dependency of maximum stresses in tent crowns of stopes σ_1 on the mining depth H_p and the corresponding value of rock pressure for various iron ore hardness is described by the logarithmic equations:

- for ore hardness $f = 4-6$

$$\sigma_1 = 28,804 \ln(H_p) - 172,05,$$

where σ_1 is the maximum stress value in tent crowns, MPa;

H_p is the mining depth, m;

- for ore hardness $f = 5-7$

$$\sigma_1 = 28,804 \ln(H_p) - 171,05;$$

- for ore hardness $f = 6-8$

$$\sigma_1 = 25,858 \ln(H_p) - 148,3.$$

The following expression enables determining the universally adaptive index of maximum stresses in tent crowns of stopes σ_1 depending on the mining depth H_p and the corresponding rock pressure value for differentiated iron ore hardness, MPa:

$$\sigma_1 = (-0,1354 f^2 + 3,0691 f + 19,873) \cdot (0,9001 \ln(H_p) - 5,3766).$$

Based on the obtained results and fundamental laws, it is possible to conclude that tent shapes of crowns possess commensurately much higher stability as compared with horizontal exposures.

The following regularities result from calculating values of maximum stresses in vaulted stopes depending on the mining depth and rock pressure values for various iron ore hardness.

The crown maximum stresses vary depending on the value of rock pressure and the vaulted crown curvature.

The most stable crowns are the vaulted ones with the curvature radius equal to a half of the stope width.

The calculated maximum stresses σ_1 of rocks when forming a vaulted crown demonstrate that crown stress values vary depending on the rock pressure value and the crown curvature with the following dependencies.

Depending on ore hardness:

- for ore hardness $f = 4-6$:

$$\sigma_1 = 2,8804 \ln(H_p) - 10,405,$$

where σ_1 is values of maximum stresses in vaulted crowns of stopes, MPa; H_p is the mining depth, m;

- for ore hardness $f = 5-7$:

$$\sigma_1 = 5,4662 \ln(H_p) - 28,635;$$

- for ore hardness $f = 6-8$:

$$\sigma_1 = 5,2211 \ln(H_p) - 27,72.$$

The numerical simulation results allow the conclusion that the vaulted shapes of stope crowns facilitate decrease of maximum stress values. The increase of the vault curvature radius causes the decrease of the compressive stress values on the crown center. At that, the maximum stress values are observed in the vault spring.

With the crown curvature radius going to infinite values, the vault center experiences tension stresses. In this case the conceptual pattern of stress distribution in the massif is identical to the strain-stress state of the horizontal crown massifs.

The following expression determines the average dependency of values of maximum stresses in vaulted crowns of stopes depending on their relative curvature radius:

$$R_o = \frac{l_{KH}}{2R_n},$$

where R_o is the relative vault curvature radius of stope crowns;

l_{KH} is the normal width of the dead stope, m;

R_n is the crown curvature radius, m.

The dependency of values of maximum stresses in vaulted crowns of stopes on the vault curvature radius is described by the equation:

$$\sigma_{1R_o} = 6,636 \ln(R_o) + 9,4.$$

Multifactor experiments enable determining the universal adaptive index of maximum stresses in vaulted crowns of stopes σ_1 depending on the mining depth H_p , relative crown curvature radius R_o and iron ore hardness f , MPa:

$$\sigma_1 = (0,2880 \ln(H_p) - 1,0405) \cdot (6,636 \ln(R_o) + 9,4) \times (-5,83 \cdot 10^{-3} f^2 + 0,1325 f + 0,4582).$$

Dependencies of maximum stresses values in inclined crowns on the mining depth and the corresponding rock pressure values for various iron ore hardness enable the following results.

Maximum stress values in the crowns under study vary depending on rock pressure and the crown dip.

The calculated values of maximum stresses in inclined crowns of stopes σ_1 depend on the mining depth H_p and the corresponding value of rock pressure for various iron ore hardness f and are described by the logarithmic equations.

The determined dependencies are described by the following equations:

- for ore hardness $f = 4-6$

$$\sigma_1 = 12,144 \ln(H_p) - 73,945,$$

where σ_1 is values of maximum stresses in inclined crowns, MPa;

H_p is the mining depth, m;

- for ore hardness $f = 5-7$

$$\sigma_1 = 12,634 \ln(H_p) - 77,974,$$

- for ore hardness $f = 6-8$

$$\sigma_1 = 11,538 \ln(H_p) - 70,707.$$

Dependency of values of maximum stresses in inclined crowns on the dip angle λ_n of the inclined crown of the stope is described by the following polynomial expression:

$$\sigma_{1n} = -0,0031\lambda_n^2 + 0,5517\lambda_n - 9,1692,$$

where λ_n is the dip angle of the inclined crown, degrees.

Results of the multifactor experiments allows determining the universal adaptive index of maximum stress values in inclined crowns of stopes σ_1 depending on the mining depth H_p , the crown dip and the corresponding iron ore hardness, MPa:

$$\sigma_1 = (1,0469 \ln(H_p) - 6,3746) \cdot (-0,0031\lambda_n^2 + 0,5517\lambda_n - 9,1692) \times \\ \times (0,2566 \ln(f) + 0,5836)$$

Simulation analysis results in the following. The increased mining depth causes considerable rock pressure growth. Therefore, when stoping at great depths, it is necessary to specify requirements for stability of exposed stopes and compensating rooms as well as for accuracy of designing construction units.

So, when mining rich iron ores by sublevel caving at Krivbass underground mines, it is advisable to make wider use of vertical and inclined compensating rooms.

The technology of mining panels with caving ore onto the tent-shaped compensating room should be applied as well.

Stope crown stability is a key requirement when mining rich ores by the room-and-pillar method. It is critical to use vaulted crowns that provide maximum stability in complicated geological and mining conditions.

Sufficiently complete research of the technology of forming vaulted crowns allows recommending the suggested methods for substantiating stable crown shapes at underground mines of the PrJSC “Sukha Balka”.

Rich iron ore deposits of “Frunze” and “Yubileynaya” underground mines, which are part of the PrJSC “Sukha Balka”, are characterised by iron content of 46%. Low-grade ores (magnetite and oxidized types of ferruginous quartzite) occur between the mines (site No.6).

The deposits of “Frunze” and “Yubileynaya” are made of rocks of Krivorozhskaya metamorphic series of the lower Proterozoic (PR1) and the Archean rocks (AR). There are seven ferruginous and seven schist levels altogether.

The Fifth and the Sixth ferruginous levels are the main ore bearing thicknesses of the iron ore deposits.

Apart from rich iron ore deposits of the Fifth and the Sixth levels, there are also minable magnetite quartzite deposits “Yuzhnaya” and “Severnaya” which are part of the magnetite quartzite areas between sites No.5 and No.6. Other ferruginous levels are of insignificant thickness, high degree of oxidation and, therefore, not minable.

The “Frunze” deposit in the V ferruginous level includes ore bodies of the underground mine “Zapadnaya of underground mine No.8”, “V-VIII Severnaya” and “Saksaganka of underground mine No.2”. Ore bodies “Diagonalnaya”, “I-III Yuzhnaya” and “Tsentrlnaya” form the Sixth ferruginous level.

The ore bodies are pillar- and sheetlike in shape. The main and largest reserves are located in the Fifth ferruginous level. The ore bodies are mainly represented by martite metals. Gothite-hematite-martite, dispersed-hematite-martite, gothite-hematite ores are represented as “margins” of martite ores. The hardness ratio of martite ores varies from 3-4 to 11-13 points (Protodyakonov scale of hardness), that of the enclosing rocks is from 9-10 to 14-16 points.

The “Yubileynaya” deposit is located in the Fifth ferruginous level and contains ore bodies “Glavnaya” and “Shurfov 42-46”. The Sixth level contains ore bodies “Gnezdo 1-2”, “Gnezdo - 3” and “Tsybulko 76”.

The stock-, nest- and seemlike ore bodies are of 190-1530 m long along the strike. The horizontal thickness makes 2-42 m; the horizontal ore area is 750-30360 m². The ore bodies occur according to enclosing rocks and have a northeastern strike and a SW dip with the 50-60 ° angle.

Weakly fractured stable hydrohematite quartzite and unstable fall-prone along sheeting planes gothite-hematite and quartz-chlorite schists make the footwall rocks. The hanging wall contains gothite-hematite and martite weakly fractured stable quartzite. The hardness ratio of the ores varies from 4-6 to 11-13 points (Protodyakonov scale of hardness), that of the enclosing rocks is from 5-6 to 11-13 points.

In accordance with the standard design “Mining systems for Krivbass underground mines” (NIGRI, 1986) and geological and mining conditions of the ore body occurrence and physical and

mechanical properties of ores and enclosing rocks at “Yubileynaya” underground mine, the PrJSC “Sukha Balka”, the sublevel-stopping system with vertical rings of deep holes onto the horizontal compensating room should be applied to mining blocks.

Mining rich iron ores at the depth of over 1300 m at “Yubileynaya” underground mine with horizontal crowns in forming undercutting rooms is connected with problems of maintaining stable horizontal exposures.

At great depths, the growing rock pressure affects stability of horizontal compensating rooms. Possible failures of horizontal crowns result in deep holes’ integrity loss and, therefore, increased amount of oversize pieces of ore. The latter impairs rich ore drawing and extraction indices and increases mining costs.

In view of the above mentioned drawbacks and the current mining technology used at the mines, the authors recommend application of the vaulted crown.

A variant of the room-and-pillar system with formation of an vaulted crown of a stope when mining of blocks by stages recommended for the PrJSC “Sukha Balka” underground mines is given in Fig. 1.

As compared to horizontal exposures, the recommended vaulted crown of a stope is of increased stability and decreases the probability of casual failures.

Formation of a vaulted crown is performed through charging some of the deep blast holes within the contour of the future stope.

Simultaneously charged and blasted holes corresponds in number and layout to the stage of forming a vaulted crown of a stope.

In forming vaulted crowns in stopes, principal stress values are distributed according to the classical law of stress distribution in rocks.

The results of the Ansys 16.2.-based stress field calculations for various shapes of vaulted crowns are given below. The finite element size is 1 m, stress values on isolines are given in MPa.

The calculation results and isolines of principal stresses σ_1 in rocks during the second stage of forming a vaulted crown in the room-and-pillar variant recommended for the PrJSC “Sukha Balka” underground mines.

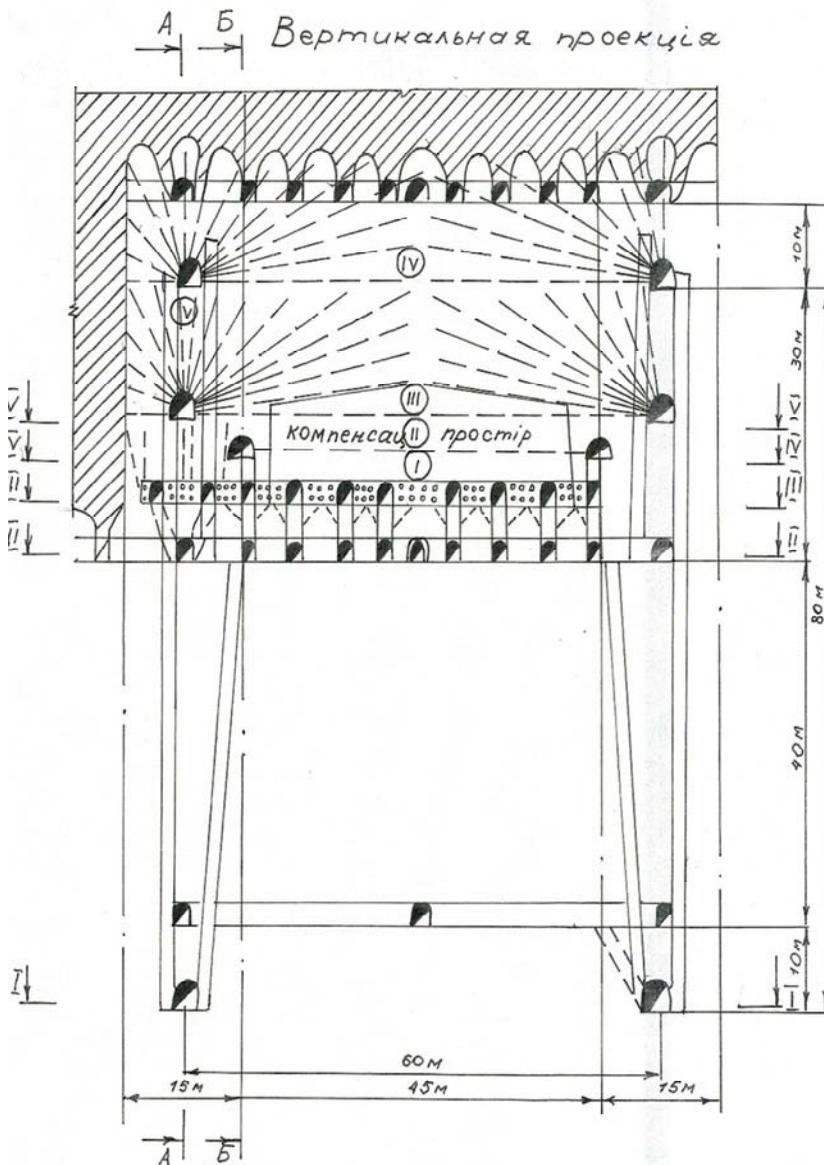


Fig. 1. A variant of the room-and-pillar system with formation of a vaulted crown:
I, II, III, IV - stages of forming an vaulted crown of a stope

Calculations principal stresses σ_1 of rocks when forming a vaulted crown demonstrate that the value of stresses in the crowns vary depending on the rock pressure value and the curvature radius of the vaulted crown. The most stable are the vaulted crowns with the curvature radius equal to a half of the stope width.

Multifactor experiments enable determination of the universal adaptive indices of a value of maximum stresses in vaulted crowns of stopes σ_1 depending on the mining depth H_p , a relative crown curvature radius R_o and differentiated iron ore hardness f , MPa:

$$\sigma_1 = (0,2880 \ln(H_p) - 1,0405) \cdot (6,636 \ln(R_o) + 9,4) \times \\ \times (-5,83 \cdot 10^{-3} f^2 + 0,1325 f + 0,4582).$$

The calculations show that vaulted crowns of stopes experience tension stresses. The largest values of stresses are observed in the vault springs; however, they are far from being destructive.

In contrast to horizontal crowns, the central part of vaulted crowns sees almost no tension stresses that are considered the most dangerous for stope exposures.

Due to absence of tensile stresses in vaulted crowns of stopes, they possess increased stability, other factors being equal.

Compensating rooms and stopes with stable vaulted crowns will experience the decreased number of destroyed holes over the stope zone caused by possible partial or complete failure of crowns.

Undestroyed deep holes will enhance massif breaking indices due to increased quality of muck crushing and corresponding decrease of oversize yield. Increased quality of rock massif breaking will enable muck ore indices and broken rich ore extraction from stopes.

Thus, considerable rock pressure growth at great depths places special requirements on stability of exposed stopes as well as stability of crown exposures of stopes and compensating rooms when designing stopes.

Due to it, when mining rich iron ores by sublevel caving systems, wider application of vertical and inclined compensating rooms is recommended for Krivbass underground mines.

Tent and vaulted crowns are advisable when mining rich iron ores by the room-and-pillar systems at depths of over 1300 m to provide maximum stability under high rock pressure.

Use of vaulted crowns in the PrJSC “Sukha Balka” underground mines will enable decrease of losses of deep holes in crowns from 18-21% to 11-15%.

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ADAPTIVE CONTROL OF THE ORE CRUSHING PROCESS IN CONE CRUSHERS BASED ON NONLINEAR PREDICTIVE MODEL

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Abstract

The paper deals with the development adaptive control system of the ore crushing process based on nonlinear block-oriented dynamic models. It is define that the best dynamic approximation quality according to the minimum coefficient of variation of the root-mean-square error and identification time are provided by applying a hybrid structure that combines the Wiener model, the Hammerstein-Wiener model and the Laguerre orthonormal functions. Using the recursive least squares algorithm for parametric identification enables to adapt the disturbances caused by changes in the mining mass characteristics. A nonlinear model predictive control system of the ore crushing process is also developed. A method of control formation is proposed and based on the block-oriented model static nonlinearities inversion. The obtained system demonstrated high dynamics quality and low computational load on the digital controller.

Keywords: ORE CRUSHING PROCESS, ADAPTIVE CONTROL, NONLINEAR MODEL PREDICTIVE CONTROL, BLOCK-ORIENTED MODELS, IDENTIFICATION, SIMULATION.

Introduction. The problem of the effective use of natural and energy resources in the context of constantly rising prices for fuel and electricity is a leading place in the state policy of Ukraine. In the production field, resource and energy capacity is inherent in the

technological processes of ore-enrichment plants, which include the ore preparation for enrichment by means of multi-stage crushing with subsequent grinding in the mills. In the structure of the ore processing cost the crushing operation share is the biggest, which is due to high energy costs.

The main factors influencing the energy consumption of a mill include particle-size distribution of the raw materials. Taking into account that grinding is preceded by crushing operations, the increase in the efficiency of ore preparation in general can be achieved by obtaining as much as possible fine and smooth ore at this processing stage. Thus, energy costs are transferred to less energy intensive process. This can be achieved either by the complete re-equipment of technological lines or by optimizing the operating modes of an existing technological installation by developing new and improving existing methods and algorithms for controlling it. From the economic point of view, the advantage is given to the last solution.

Existing methods and systems of automated control of the ore crushing process do not allow to control the particle-size distribution of the finished product effectively enough, therefore the development of adaptive control systems, which will allow to provide high characteristics of particle-size in conditions of fluctuations of ore properties, changes in the characteristic of technological equipment and the presence of obstacles in data transmission channels is an actual scientific task. For qualitative adaptive control formation, an adequate mathematical description of the plant must be known. Taking into account the nonlinearity of the ore crushing process, in predicting its behavior, it is advisable to use block-oriented models (BOM) because of a clear separation of linear and nonlinear parts. This feature allows to use a wide range of linear dynamic models and static nonlinear functions in plant modeling.

Thus, providing a given particle-size distribution of the finished product with crushing ore in cone crushers by applying adaptive control with a predictive block-oriented structure to parametrize the trajectory of control actions, which will allow with introduction into production to increase profits from the functioning of existing equipment by reducing the cost of the ore preparation product is currently the topical issue.

Analysis of literary data and problem statement. Today, in the theory of identification the following directions related to the synthesis of nonlinear system models are widely searched: models based on Volterra series use, nonlinear input-output models (NARX, NARMAX, NOE, etc.), block-oriented models (Hammerstein, Wiener and Hammerstein-Wiener) and fuzzy-neural network models.

For a nonlinear Volterra system, the relationship between input and output can be represented in the form of an infinite series [1]. The disadvantage of the practical use of this type of model lies in the estimation of a large number of unknown characteristics with truncation of infinite series [2, 3]. Also, parametric identification can be performed using a sufficiently deep sampling [3]. However, due to the lack of feedback on the output Volterra model provides guaranteed stability.

In contrast to Volterra nonlinear system, nonlinear input-output models have an output feedback that allows us to obtain a mathematical description of the dynamic process in a more compact form [3].

Depending on the type of operator, the following models are distinguished: NARX (nonlinear ARX), NARMAX (nonlinear ARMAX), NOE (nonlinear OE). As in the Volterra system, the regressors number of these models increase with rising polynomial orders and sampling depths. This disadvantage is especially critical for the NARMAX model. NARX, NARMAX and NOE models are sensitive to external disturbances and noise in data transmission channels. Also the modeling accuracy of the plant dynamics depends on the sampling period [4].

Block-oriented nonlinear models (BOM) are conventionally divided into nonlinear static and linear dynamic blocks. In the Hammerstein model, a nonlinear static block is located in front of a linear dynamic part. In the Wiener model, on the contrary, the linear block precedes the nonlinear one. In the Hammerstein-Wiener model, the dynamic part of the model is between two nonlinear blocks.

The complexity of the parametric identification of block-oriented models lies in the need to consider the interrelations between structural elements. Many methods and algorithms for evaluating the characteristics of the considered functions and models have been

developed and investigated [5]. However, if there is information about the nature of static nonlinearities, the process of developing a model is significantly reduced. In this case, the identification is a subject to an exclusively linear dynamic model. In the tasks of automated precise control, it is expedient to use the block-oriented models (BOM) in conjunction with the systems of Laguerre orthonormal functions (OBF) through a clear separation of linear and nonlinear parts [6].

To determine the structure of a nonlinear model that will allow to provide acceptable prediction of the plant reaction to change the controls, a comparative analysis of the approximation quality should be performed using three typical unified BOMs: Hammerstein, Wiener and Hammerstein-Wiener. In connection with the use of the Laguerre model to represent the linear part of these models, it is necessary to adapt their traditional mathematical description in the space of states by analogy with [7].

The model predictive control (MPC) method has demonstrated high efficiency in technological processes control. The principle of control is to predict the system behavior at a certain interval and ensure that it is best approximated to the output of the plant to the reference signal [8-10] by solving the optimization problem. In this case, the most common form of the objective function is the quadratic criterion.

Taking into account the nonlinearity of the ore crushing process, it is expedient to consider possible ways of solving the predictive control problem, provided that the predictive model is also nonlinear (NMPC). A simpler method is linearization of the nonlinear model around the operating point [11] and the application of linear predictive control (MPC) methods. However, the qualitative characteristics of such controller significantly deteriorate at significant deviations from the nominal operation conditions, which is explained by the inability of the linearized model to describe the global behavior of the nonlinear system.

NMPC control is a form of nonlinear programming problem, so methods of sequential quadratic programming or internal point [12] can be used to find the optimal trajectory of control actions. However, as a result, we see the increase of the computing load on the controller and the system's performance at computing in real

time. This is due to the more complex iterative procedure of finding a solution to the problem of nonlinear programming, in comparison with the linear one.

The third method is based on the application of the static nonlinearity inversion method [11], which is convenient to use with block-oriented systems because of the independence of linear and nonlinear block models. At the same time, formulating the task of predictive control, the replacement of the source coordinates, control actions and signals of the task by intermediate reciprocal variables is carried out. Then, to determine the control vector, algorithms of linear or quadratic programming are used, depending on the absence or presence of restrictions on the input-output variables. However, the use of nonlinear compensators can lead to quasi-optimal and often sub-optimal solutions of the modified predictive control task.

The key task in model predictive control is to determine the future control steps, that is, the sequence of amplitudes of control action or its increments. Given the high sampling frequency and the long horizons of prediction, the number of elements of the vector of controls that need to be determined may be sufficiently significant, which reduces the time of finding the optimal solution, due to the large computational load [13].

The research [14] is focused on reducing the time of settlement operations of MPC control by reducing the controls sequence number degrees of freedom within the prediction horizon. In research [15], the computational efficiency of determining the trajectory of control increases due to its approximate representation of wavelet functions. Similarly, in researches [13, 16], the approximation of the control trajectory is carried out by a system of orthonormal basis Laguerre functions. This allows to use a unified descriptor of the controls sequence. As a result, after determining the structure of the model (its order), the number of parameters to be identified is significantly reduced and limited by parameters of a set of orthonormal functions.

The advantage of using MPC in the conditions of the ore mining processes control is determined by the possibility of taking into account the physical and technological constraints of the process by imposing restrictions on the amplitude, increment of control and output coordinates and confirmed by researches [17, 18]. The

second, more general advantage is the definition of control effects in real time. In this case, the speed is limited only by the frequency of tacting modern hardware controllers and the speed of optimization algorithms convergence.

The purpose of the work. The purpose of the study is to develop principles and structure of ore crushing process adaptive control system based on the predictive model, which provide the formation and maintenance of the specified homogeneity of the crushed product and the control size class output under the influence of uncontrolled disturbances caused by fluctuations of the ore raw material characteristics, changes in the parameters of technological equipment and errors in data transmission channels.

Results of the development of nonlinear models of the ore crushing process in cone crushers and studying the quality of their structural and parametric identification approximating the process dynamics. The dynamics of the control plant can be described by a model constructed on the basis of orthonormal Laguerre functions, which is represented as follows in the state-space discrete form [6, 16]:

$$\begin{aligned} L[k+1] &= \Phi L[k] + \Gamma u[k], \\ y[k] &= C^T L[k], \end{aligned} \quad (1)$$

where p – the order of the model; $L[k] = [l_1[k] \ l_2[k] \ \dots \ l_p[k]]^T$ – state vector consisting of Laguerre functions; Φ – the lower triangular matrix of size $(p \times p)$; Γ – a vector-column of size $(p \times 1)$:

$$\Phi = \begin{bmatrix} \psi & 0 & 0 & \dots & 0 \\ \varrho & \psi & 0 & \ddots & 0 \\ -\psi\varrho & \varrho & \psi & \ddots & \vdots \\ \vdots & \vdots & & \ddots & 0 \\ (-\psi)^{p-2}\varrho & (-\psi)^{p-3}\varrho & \dots & \varrho & \psi \end{bmatrix}, \quad (2)$$

$$\Gamma = \sqrt{\varrho} \begin{bmatrix} 1 \\ -\psi \\ \psi^2 \\ \vdots \\ (-\psi)^{p-1} \end{bmatrix}; \quad C = \begin{bmatrix} c_1 \\ c_2 \\ c_3 \\ \vdots \\ c_p \end{bmatrix}, \quad (3)$$

where $g = (1 - \psi^2)$; ψ – a scale factor that must be within $0 \leq \psi < 1$ to ensure system stability.

The task of the Laguerre OBF model (1) parametric identification is reduced to the definition of coefficients vector C (3) components and the scale factor ψ .

The modified Hammerstein model containing the static nonlinearity in the part describing the effect of input actions on the state of the state vector can be represented as follows:

$$\begin{aligned} L[k+1] &= \Phi L[k] + \Gamma \Xi(u[k]), \\ y[k] &= C^T L[k], \end{aligned} \quad (4)$$

where $\Xi[k] = [g_1(u[k]) \ g_1(u[k]) \ \dots \ g_m(u[k])]^T$ – $g_i: \mathbb{R}^m \rightarrow \mathbb{R}$ – static nonlinear functions; $L[k] \in \mathbb{R}^n$ – Laguerre model state vector; $u[k] \in \mathbb{R}^m$ – input vector.

Similarly, the Wiener model is formulated:

$$\begin{aligned} L[k+1] &= \Phi L[k] + \Gamma u[k], \\ y[k] &= \Psi(C^T L[k]), \end{aligned} \quad (5)$$

where $\Psi(\cdot): \mathbb{R}^n \rightarrow \mathbb{R}$ – static nonlinear function, which connects the Laguerre model state vector with the block-oriented model output.

By combining the two previous models (4) and (5) we obtain the Hammerstein-Wiener model in the state-space:

$$\begin{aligned} L[k+1] &= \Phi L[k] + \Gamma \Xi(u[k]), \\ y[k] &= \Psi(C^T L[k]). \end{aligned} \quad (6)$$

To perform the series of computational experiments as an plant an improved [22] analytical model of the ore crushing process [21, 20] was used along the channels "rotation speed – homogeneity of the crushed product" and "rotational speed – control size class output".

The calculations were carried out in the MATLAB software package on a PC with the following configuration: Intel Core i3-3120M 2.5 GHz 4GB RAM Win7 x64.

To assess the accuracy of the approximation of the crushing process characteristics and the model predictive control quality, the coefficient of variation of the root-mean-square error between the test and model data is used:

$$CV(RMSE) = \frac{\sqrt{\frac{1}{n} \sum_{i=1}^n y_i - \hat{y}_i}}{\sum_{i=1}^n \hat{y}_i / n}, \quad (7)$$

where y_i – test value; \hat{y}_i – model value; n – the number of measurements.

The ore crushing process model identification was carried out according to the structural scheme (see Fig. 1). As already mentioned above, the multi-zone model of a cone crusher was adopted as an identification plant, the adequacy of which is confirmed by studies [20-22]. The inputs $u_\omega[k]$ i $u_\theta[k]$ were subjected to test samples corresponding to the laws of changing the rotation speed and closed side setting of the technological unit. Outputs were taken off the original values that characterize the qualitative parameters of the crushing process, namely the control size class output in crushed ore $y_\gamma[k]$ and the general index of its homogeneity $y_{CV}[k]$.

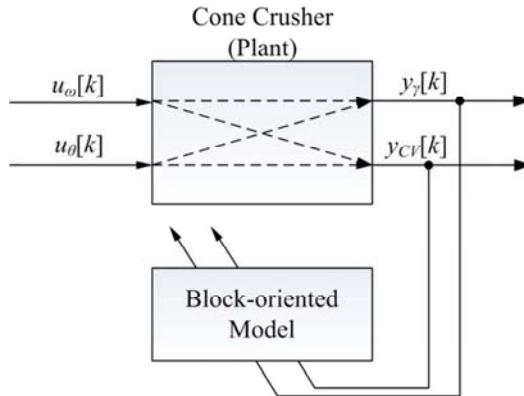


Fig. 1. Block-diagram of the ore crushing process model structural-parametric identification

Carrying out the research, the parametric identification of the Laguerre model was initially carried out using adaptive algorithms: least-mean squares algorithm (LMS), normalized least-mean squares algorithm (NLMS), recursive least squares algorithm (RLS) [12-14]. After obtaining a mathematical description of the linear part, an

iterative evaluation of the parameters of static nonlinearities was performed. Stationary nonlinearities of BOM are approximated by piecewise linear functions with iterative estimation of their parameters by nonlinear optimization algorithms (Gauss-Newton, Levenberg-Marquardt and the fastest descent) at each step of the calculations, followed by a choice of values that minimize the accuracy of the modeling CV(RMSE). The results of computational experiments for models of regime parameters of the crushing process are summarized in table 1.

The obtained data demonstrate that the best approximation of the process characteristics by control size class output is achieved using the Wiener model and the recursive algorithm for estimating the Laguerre OBF parameters. The accuracy of this model is 1.9% higher than that of nonlinear systems of the same structure, but with other parametric identification algorithms and 26.5 times higher than the linear model with the RLS algorithm [22].

Table 1

The accuracy of block-oriented models identification with Laguerre OBF and adaptive parameter estimation algorithms

Model	Identification algorithms	CV(RMSE), %	
		Control size class output, %	Coefficient of variations of size density function
Hammerstein with Laguerre OBF	LMS	20,65	11,14
	NLMS	21,03	4,19
	RLS	21,75	3,85
Wiener with Laguerre OBF	LMS	2,029	1,69
	NLMS	2,028	1,85
	RLS	1,99	3,13
Hammerstein-Wiener with Laguerre OBF	LMS	11,67	0,62
	NLMS	19,22	0,52
	RLS	12,5	0,51

Fig. 2b shows the time series of the plant and block-oriented models outputs that provide the minimum CV(RMSE_y).

As you can see from the graphs, the Hammerstein-Wiener model with the LMS algorithm does not adequately describe the inertia of

the ore crushing process and has the highest error of the steady-state of $e_{cm\gamma} = 3,9\%$ compared to other models that are given. In contrast, all Wiener models are acceptable to simulate the dynamics of a plant. The best accuracy in steady-state for the Wiener structures is achieved with a RLS algorithm of parametric identification $e_{cm\gamma} = 0,53\%$, and worst – with the NLMS algorithm $e_{cm\gamma} = 0,64\%$.

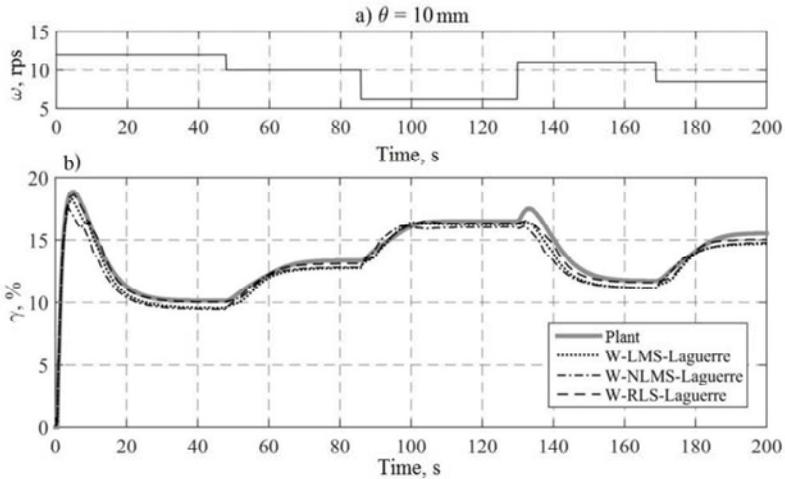


Fig. 2. Time series of the plant and Wiener models outputs

On the other hand, for the coefficient of variations of size density function, the Hammerstein-Wiener model is more accurate than the Wiener structure (table 1, fig. 3b).

The maximum accuracy of simulation of the mode parameter is a model with a recursive algorithm for determining the linear dynamic part $CV(RMSE_{CV}) = 0,51\%$. The dynamics of the ore crushing process by all three Hammerstein-Wiener models is described adequately. The average error in steady-state is $e_{cm CV} = 0,24\%$, $e_{cm CV} = 0,6\%$ and $e_{cm CV} = 0,16\%$ adapting the models using the algorithms of the usual and normalized least squares, and the recursive least squares algorithm, respectively.

Consequently, in this case, the best accuracy have a model with a recursive algorithm of linear block parameters estimation. This adaptive algorithm (so one source program in controller) can be used

to estimate the model parameters on other channels. This will reduce the controller memory load.

Using the Wiener nonlinear structure and the LMS algorithm identifying the Laguerre system parameters, there is oscillation in dynamics and the error in steady-state is $e_{cm CV}=0,59\%$.

We also note that for both parameters the worst identification accuracy was demonstrated by Hammerstein's models, regardless of the adaptive parameter estimation algorithm. In this case, the use of the LMS algorithm leads to deterioration in the modeling of the process behavior by the coefficient of variation of size density function 17.86%.

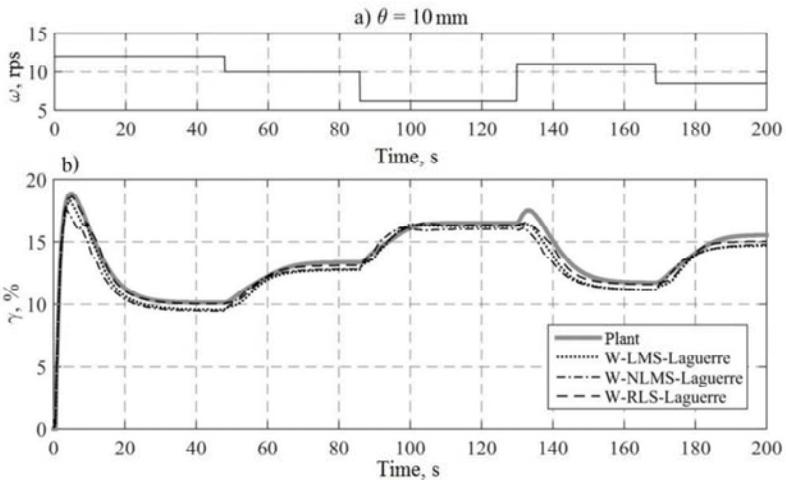


Fig. 3. Time series of the plant and Hammerstein-Wiener models outputs

We perform a comparison of the received block-oriented models that provide the minimum value of the approximation quality index, with the typical nonlinear structures used in the theory of system identification. Third-order Volterra, Wiener, and Hammerstein-Wiener models have a linear part of the "output-error" structure (W-OE, H-W-OE), as well as the nonlinear autoregressive model (NARX).

Simulation of the influence of external disturbances and noise in data transmission channels was carried out by applying to the output of the plant additive interference, which is represented by a sequence of random variables with normal distribution. Computational

experiments were performed at various values of the distribution standard deviation σ . With each change of standard deviation, the structural identification of the models W-OE, H-W-OE and NARX was carried out by direct sampling of the parameters n_a, n_b, n_f, n_k . In order to determine the stability of the parametric identification process, for each value of σ and defined sets of model parameters, calculations were made 30 times. The averaged CV(RMSE) values are summarized in table 2, 3.

According to the obtained data (fig. 4b, table 2), the Hammerstein-Wiener with Laguerre (HW-RLS-Laguerre) models and the "output-error" structure provides the best approximation of the crushing process reaction by the coefficient of variation of size density function on the cone rotational speed changes.

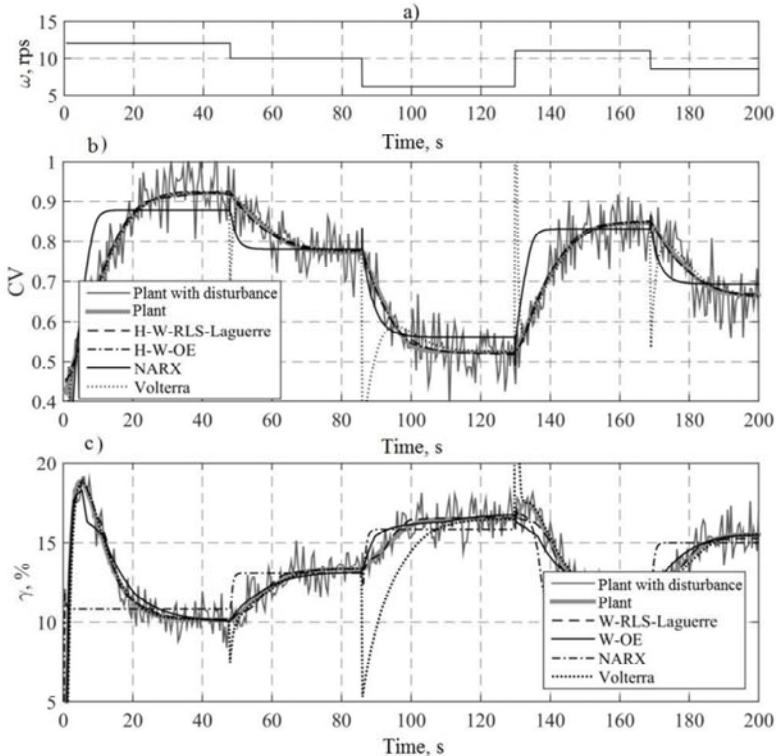


Fig. 4. Time series of the plant and nonlinear models outputs

In this case, the structure with the Laguerre model has 9.8% greater accuracy with $\sigma = 0,005$ and at 2.1%, with $\sigma = 0.1$ in relative terms. It should also be noted that the error of the stable mode of the proposed block-oriented system is only $e_{cm CV} = 0,18 \%$. For comparison, in W-OE and H-W-OE models with "output-error" structure, this value is $e_{cm CV} = 2,17 \%$ and $e_{cm CV} = 2,26 \%$, respectively.

The worst quality of simulation was shown by NARX. The spread of values of the variation coefficient of the root-mean-square error with a consistent increase σ shows the instability of the process parameters estimating. The polynomial structure inadequately reflects the inertia of the ore crushing process and, in general, the behavior of the plant in steady-state.

Table 2

The accuracy of nonlinear models identification of the process of fragmentation by the coefficient of variation of the grain size characteristic

Standard deviation σ , %	CV(RMSE), %				
	H-W-RLS-Laguerre	W-OE	H-W-OE	NARX	Volterra
0,005	1,02	2,17	1,12	9,75	9,23
0,01	1,27	2,39	1,54	8,73	9,29
0,025	2,97	3,64	3,15	14,4	9,72
0,05	5,89	6,34	6,11	17,37	11,07
0,075	8,85	9,18	9,25	22,5	12,94
0,1	11,84	12,1	12,09	20,05	15,22

The average deviation of the output of this model in a steady0state relative to the output of the plant without additive disturbance is $e_{cm CV} = 17,9 \%$, that is the worst among considered nonlinear structures. Unlike NARX, the 3rd order Volterra system has a better accuracy of steady-state description. The error of the steady-state is, at the same time interval $e_{cm CV} = 0,013 \%$. However, in the simulation of the transition process, such system loses stability. As a result, for its, the coefficient of variation of the root-mean-square error is quite high. At the same time, it changes by only 5.99% in the range of standard deviations of additive disturbance

$\{\sigma \in \mathbb{R} \mid 0,005 \leq \sigma \leq 0,1\}$. For comparison, with the increase of σ from 0.005 to 0.1, the accuracy of the Hammerstein-Wiener model with Laguerre OBF drops by 10.82%. At the same time, all the BOMs, regardless of the type of linear model, are quite stable.

A comparative analysis of the modeling results of the behavior of the ore crushing process by the control size class output (table 3, fig. 4c) shows that the best overall accuracy of the Wiener structure with Laguerre OBF (W-RLS-Laguerre). Its accuracy is higher by 3,02% at $\sigma = 0,05$ and 11,98% at $\sigma = 1$ than in the W-OE structure, which also showed a fairly high quality of approximation, especially with slight disturbances.

The Wiener model with Laguerre OBF best describes the steady-state process $e_{cm\gamma} = 0,61\%$ of the three block-oriented structures. For comparison, the steady-state errors of the W-OE and HW-OE systems make up $e_{cm\gamma} = 1,39\%$ and $e_{cm\gamma} = 1,64\%$, respectively. It should be noted that the W-OE system is unstable. At the same time, CV(RMSE) increases by a maximum of 2.67%, so loss of stability can be considered as not significant.

Table 3
The accuracy for identification of nonlinear models of the splitting process for the partial execution of the size class -9.1+6.7 mm

Standard deviation σ , %	CV(RMSE), %				
	W-RLS-Laguerre	W-OE	H-W-OE	NARX	Volterra
0,05	2,07	5,09	10,17	13,2	15,1
0,075	2,12	5,29	12,84	14,9	15,15
0,1	2,19	5,51	10,07	39,14	15,17
0,25	2,93	5,46	10,95	28,18	15,28
0,5	4,7	19,73	21,8	23,9	15,81
0,75	6,72	18,53	21,1	30,25	16,58
1	8,81	20,79	22,91	32,5	17,64

As with the simulation of the variation coefficient of the size density function, the impossibility of practical application of the NARX model proves low quality of identification (accuracy and stability). The 3rd order Volterra system demonstrated the best modeling accuracy of the ore crushing process in steady-state. The

steady-state error is only $e_{cm\gamma} = 0,012\%$. However, as before, it remains unstable in dynamics. The coefficient of variation of the root-mean-square error of the Volterra system increases by only 2.54% in the range of mean-square deviations of the additive disturbances $\{\sigma \in \mathbb{R} \mid 0,05 \leq \sigma \leq 1\}$. Under the same conditions, the general accuracy of the Wiener modeling with the Laguerre linear model increases by 6.74%.

Thus, as a result of the research, it was found that the best quality of the ore crushing process dynamics modeling has a hybrid block-oriented structure consisting of Hammerstein-Wiener and Wiener models, in which an orthonormal basis Laguerre functions system is used as a linear block (see fig. 5).

In addition to analyzing the quality of the ore crushing process simulation an investigation of the identification speed was additionally performed. Due to the high convergence rate of the adaptive RLS algorithm for identifying the Laguerre model, the total time of evaluating the characteristics of block-oriented structures on its basis was significantly lower than in standard nonlinear systems.

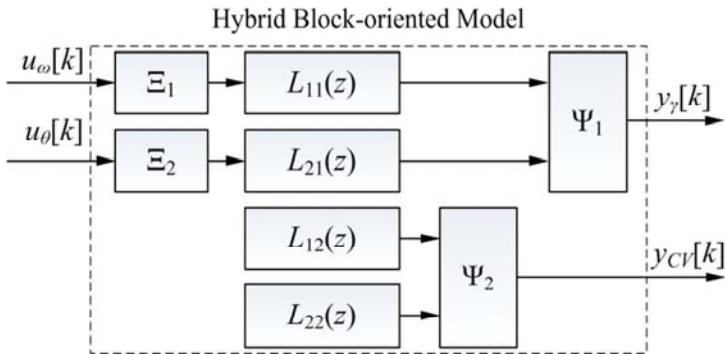


Fig. 5. Hybrid block-oriented model of ore crushing process

However, due to the determination of the parameters of static nonlinearities through the sequential use of three algorithms for nonlinear programming at one step, the total time of identification of the hybrid model was 153 milliseconds, which exceeds the system sampling period. Therefore, to reduce the identification time only the

Gauss-Newton algorithm was used, which reduced the time to 42.7 milliseconds without significant loss of accuracy.

Task formalization of multidimensional predictive control of ore crushing process based on hybrid block-oriented model

Given the nonlinearity of the hybrid model in the formation of controls, it is necessary to use methods and algorithms for nonlinear programming (NMPC), which, firstly, are complex in implementation and require significant memory resources of controller, and, secondly, have low performance through its iterative character [24]. Therefore, it is advisable to explore the possibility of using an alternative model predictive control method based on finding the block-oriented models static nonlinearities inverse functions.

In general, the solution of the model prediction control problem is to determine the trajectory of the controls, which minimizes the quadratic criterion of the form:

$$J = (R - \hat{Y})^T Q (R - \hat{Y}) + \Delta \hat{U}^T S \Delta \hat{U}, \quad (8)$$

Subject to:

$$\begin{aligned} & \left\{ \hat{U} \in \mathbf{R}^n \mid \hat{U}_{\min} \leq \hat{U} \leq \hat{U}_{\max} \right\} \forall \{k \in \mathbf{N} \mid 1 \leq k \leq N_c - 1\}; \\ & \left\{ \hat{Y} \in \mathbf{R}^n \mid \hat{Y}_{\min} \leq \hat{Y} \leq \hat{Y}_{\max} \right\} \forall \{k \in \mathbf{N} \mid 1 \leq k \leq N_p\}, \end{aligned} \quad (9)$$

Applying a block-oriented model:

$$\hat{U} = \Xi(U); \hat{Y} = \Upsilon(C^T \Lambda),$$

where N_p, N_c – prediction and control horizons; Q, S – matrix of input-output weighing coefficients; $\Xi(\cdot), \Upsilon(\cdot)$ – nonlinear input-output functions; U – vector of input actions $U = [u[k] \ u[k+1] \ \dots \ u[k+N_c-1]]^T$; $\Delta \hat{U}$ – control trajectory $\Delta \hat{U} = [\Delta \hat{u}[k] \ \Delta \hat{u}[k+1] \ \dots \ \Delta \hat{u}[k+N_c-1]]^T$; \hat{Y} – vector of predicted output values $\hat{Y} = [\hat{y}[k+1|k] \ \hat{y}[k+2|k] \ \dots \ \hat{y}[k+N_p|k]]^T$; R – reference signal $R = [r[k+1|k] \ r[k+1|k] \ \dots \ r[k+N_p|k]]^T$; $\hat{Y}_{\min}, \hat{Y}_{\max}, \hat{U}_{\min}, \hat{U}_{\max}$ – constraints on output and control amplitudes.

Inversion of the hybrid model static nonlinear input-output functions allows us to reduce the problem (8) and (9) to the quadratic programming problem in the following form [11]:

$$\begin{aligned} J &= \left(R^* - \hat{Y}^*\right)^T Q^* \left(R^* - \hat{Y}^*\right) + \Delta \hat{U}^{*T} S^* \Delta \hat{U}^*, \\ R^* &= \Upsilon^{-1}(R); \hat{Y}^* = \Upsilon^{-1}(\hat{Y}); \hat{U}^* = \Xi^{-1}(\hat{U}), \end{aligned} \quad (10)$$

Subject to:

$$\begin{aligned} \left\{ \hat{U}^* \in \mathbf{R}^n \mid \hat{U}_{\min}^* \leq \hat{U}^* \leq \hat{U}_{\max}^* \right\} \forall \{k \in \mathbf{N} \mid 1 \leq k \leq N_c - 1\}; \\ \left\{ \hat{Y}^* \in \mathbf{R}^n \mid \hat{Y}_{\min}^* \leq \hat{Y}^* \leq \hat{Y}_{\max}^* \right\} \forall \{k \in \mathbf{N} \mid 1 \leq k \leq N_p\}, \end{aligned} \quad (11)$$

where Q^* , S^* – matrix of input-output factors weight; $\Xi^{-1}(\cdot)$, $\Upsilon^{-1}(\cdot)$ – inverse nonlinearities of the input-output; $\Delta \hat{U}^*$ – inverted control trajectory $\Delta \hat{U}^* = [\Delta \hat{u}^*[k] \quad \Delta \hat{u}^*[k+1] \quad \cdots \quad \Delta \hat{u}^*[k+N_c-1]]^T$; \hat{Y}^* – inverted vector of predicted output values of the block-oriented model $\hat{Y}^* = [\hat{y}^*[k+1|k] \quad \hat{y}^*[k+2|k] \quad \cdots \quad \hat{y}^*[k+N_p|k]]^T$; R^* – reference on prediction horizon $R^* = [r^*[k+1|k] \quad r^*[k+2|k] \quad \cdots \quad r^*[k+N_p|k]]^T$.

In order to find the inverse functions, it is proposed to use the ZEROIN algorithm tested in [25, 26].

It should also be noted that the computing load of a digital controller is influenced by the length of the prediction N_p and the control N_c horizons, which determine the size of the system matrices and the number of control trajectory elements to be evaluated. It is proposed to investigate the appropriateness of the Laguerre OBF control trajectory approximation to reduce the number of parameters to be estimated.

Taking into account the considerations discussed above, a block-diagram of the ore crushing process control system was compiled with the predictive model presented in fig. 6.

It consists of three main units: the control plant – a cone crusher, a hybrid predictive model and a controller. The structure of the predictive model includes two pairs of Wiener and Hammerstein-Wiener systems.

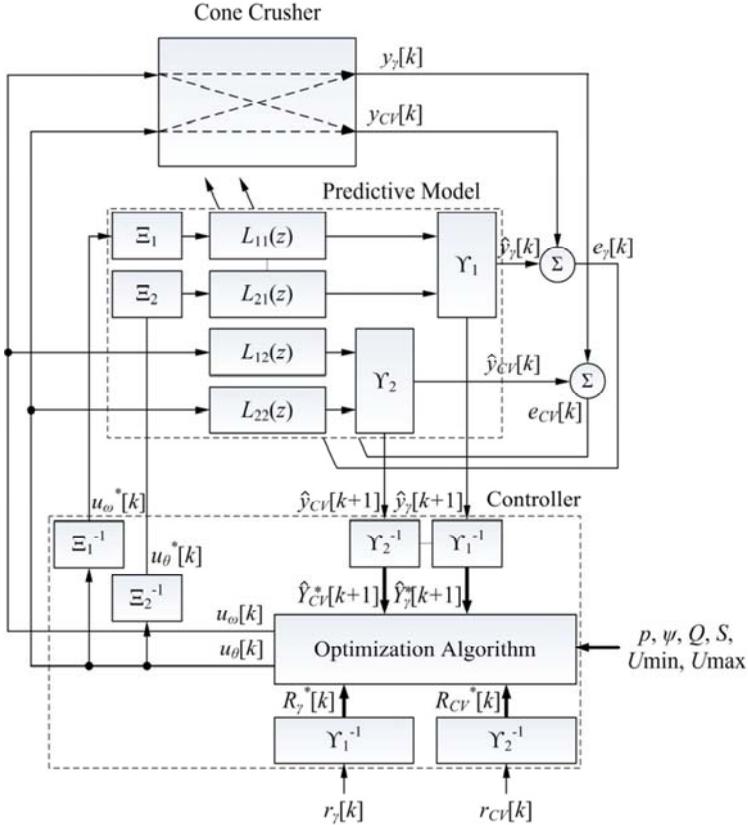


Fig. 6. Block-diagram of the crushing process model predictive control system

Note that the static nonlinearities at the output of the model Υ_1 and Υ_2 have a combined form. This is due to the fact that in the multidimensional system the output signals of the linear models $L_{11}(z)$, $L_{21}(z)$ i $L_{12}(z)$, $L_{22}(z)$ are added either before or after the nonlinear block and the number of values of the combination during the inverting is directed to infinity, that is, it can be argued that the inverse function does not exist. The features of the identification process of the combined nonlinearities of block-oriented models and their subsequent inverting are considered in [27, 28]. The controller consists of blocks of inverse nonlinear functions Ξ_1^{-1} , Ξ_2^{-1} , Υ_1^{-1} , Υ_2^{-1} ,

as well as a block that minimizes the criterion by one of the quadratic programming methods.

On the input of a closed system signals are given for control size class output and the coefficient of variation of the size density function $r_y[k]$, $r_{CV}[k]$, the matrix of weight coefficients Q , S , the constraints on the controls amplitude U_{\min} , U_{\max} , the order p and the scale factor ψ of Laguerre model, which approximates the control trajectory.

The symbol "*" denotes the scalar and vector inverse values of the corresponding actions. The totally thick line is indicated by vector signals.

Simulating the MIMO adaptive control system of the ore crushing process based on a hybrid block-oriented predictive model

To evaluate the efficiency of the ore crushing process model predictive control system which using inverse nonlinear functions and Laguerre OBF parameterizing the control vector (the iLMPC system), we perform a series of computational experiments. In order to carry out a comparative analysis of the dynamics quality and the computational load, we additionally perform the simulation of the predictive controller operation with the nonlinear sequential quadratic programming algorithm for identifying control trajectory elements (NMPC). The length of the prediction trajectory N_p , the number of controls N_c and the weight coefficients matrices values Q , S are assumed to be the same for both systems.

The simulation was performed for 1000 samples with sampling interval $\Delta t = 0.5$ c. So, the experimental time lapse was $\{t \in \mathbf{R} \mid 0 \leq t \leq 500\}$ seconds. The reference signals were submitted to the input of both systems, and change according to the same law. The reference signal by the variation coefficient of size density function r_{CV} in the beginning of the calculation increases to 0.7, and then in 90 seconds – to 1.2. Next, the value of r_{CV} is reduced to 0.8 on 190 seconds and to 0.4 on 290. The reference for a control size class output r_y after the start of the experiment steps from 0 to 9% and then to 15% and 18% at 140 and 240 seconds respectively. Then in 340 seconds signal is reduced to 11%.

The controllers are configured as follows. The prediction horizon for N_p is 20 counts. The matrixes of input-output weights were chosen arbitrarily with the following values: for the coefficient of variation $Q_{CV} = 600$, for the control size class output $Q_y = 300$, for the input of the cone rotational speed $S_\omega = 0,01$ and for entering the closed side setting $S_\theta = 0,01$. The NMPC system has a 10-point control horizon. The amplitudes of the control vectors first components are imposed by the constraints of $\{\omega \in \mathbf{R} \mid 6 \leq \omega \leq 12\}$ revolution per second and $\{\theta \in \mathbf{R} \mid 8 \leq \theta \leq 13\}$ mm.

In the real cone crusher the induction motor is used to rotate the cone. It is expedient to adjust the speed by influencing the supply voltage frequency. To simulate the dynamics of the executing mechanism along the channel "voltage frequency–rotation speed" we use the linearized model of the induction motor [30]:

$$\begin{cases} \frac{d\omega}{dt} = \frac{1}{J}(M - M_c); \\ T_e \frac{dM}{dt} = \beta(\omega - \omega_0); \\ \omega_0 = \frac{2\pi f}{p}, \end{cases} \quad (12)$$

where $\beta = 2M_{\max} / \omega_0 s_{\max}$ – mechanical stiffness module; $T_e = 1 / \omega_0 s_{\max}$ – the equivalent electromagnetic time constant of the stator and rotor circuits; f – voltage frequency; p – the number of poles pairs; M – engine torque; M_c – static torque; M_{\max} – critical torque; s_{\max} – critical slip; ω – motor speed; ω_0 – synchronous motor speed; J – moment of inertia.

The CH880 EEF cone crusher has motor with nominal parameters: $P = 600$ kW, $\omega_0 = 78.5$ rad/s, $\eta = 0.935$, $\cos\varphi = 0.85$, $J = 1600 \cdot 10^{-2}$ kg·m², $p = 4$, $\lambda_{\max} = 2$. According to the motor, the missing model parameters are calculated: $s_{\max} = 0.049$; $\beta = 8125$ H·m·s; $T_e = 0.262$ s.

The closed side setting dynamics can be approximated by the nonlinearity of the type "limiting the speed of the signal change". The analysis of the obtained experimental data shows that the level of speed restriction with signal growth is 0.1047 mm/s, with decrease –0.6059 mm/s.

Perform the research of controller qualitative characteristics exposing the plant of external uncontrolled disturbances. We will perform the models for two disturbances: high-frequency interference with low amplitude that is characteristic of channels for transmitting data from sensors to a control device and low frequency with high amplitude due to fluctuations in granulometric and physical-mechanical properties of the mountain mass. To model the first action we use a sequence of random numbers that change on each step in the normal distribution with the mean square deviations $\sigma_{CV1} = 0.01$ and $\sigma_{CV2} = 0.05$ for the coefficient of variation of size density function $\sigma_{\gamma1} = 0.1 \%$ and $\sigma_{\gamma2} = 0.5 \%$ for the control size class output $-9.1+6.7$ mm. The simulation of low frequency oscillations will also be accomplished by using a sequence of random numbers with mean square deviations $\sigma_{CV2} = 0.2$ and $\sigma_{\gamma2} = 2.2 \%$, varying at each 60th samples. The results of modeling the work of predictive controllers are presented in fig. 7 and in table 4.

Starting up in the considered systems the dynamics quality decreases with the control size class output in comparison with the system without interference. In the iLMPC system appears overshoot $\delta_{CV} = 12.7\%$, and the NMPC increases the settling time with less overshoot ($\delta_{\gamma} = 10.2\%$). On the other hand, while controlling the homogeneity, the qualitative characteristics of the iLMPC and NMPC systems do not significantly deteriorate. Overall, the CV(RMSE) for the systems under consideration was: iLMPC for CV12.1%, for γ 7.9%, NMPC for CV 16.5%, for γ 9.3%.

Table 4
Control errors during operation of various predictive controllers

Predictive controller	Control error, %		Computational time, ms	
	Disturbance $\sigma_{CV} = 0,01$ i $\sigma_{\gamma} = 0,1 \%$	Disturbance $\sigma_{CV} = 0,05$ i $\sigma_{\gamma} = 0,5 \%$	Disturbance $\sigma_{CV} = 0,01$ i $\sigma_{\gamma} = 0,1 \%$	Disturbance $\sigma_{CV} = 0,05$ i $\sigma_{\gamma} = 0,5 \%$
iLMPC	0,31	0,87	1,14	1,19
NMPC	2,53	3,48	168,34	195,16

We note that the speed of the iLMPC controller does not significantly change compared to the undisturbed system and averaged 1.19 milliseconds, while the computational speed of the NMPC system is reduced to 195.16 milliseconds that is on 15.9%.

Thus, it can be stated that the model predictive controller with the inversion of static nonlinearities and the approximation of the Laguerre OBF paths has the best dynamics qualitative characteristics, in particular, the speed, accuracy and computational load compared to the usual nonlinear model predictive controller. Taking into account the time of the adaptive parameters identification of the hybrid predictive model, the total calculation time is 43.84 milliseconds, which is much less than the sampling interval.

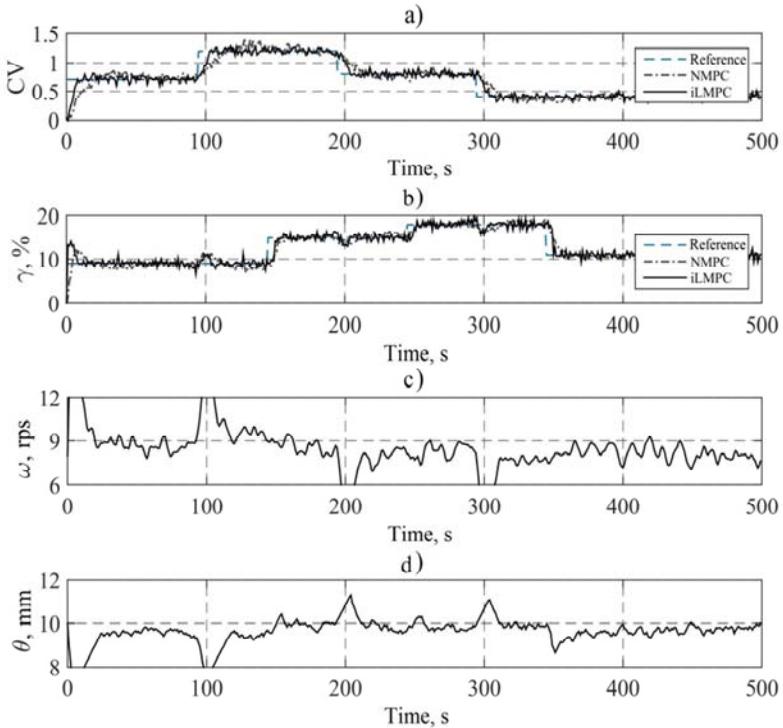


Fig. 7. Time series of the ore crushing process adaptive MPC control system with disturbances ($\sigma_{CV} = 0,05$ and $\sigma_{\gamma} = 0,5$ %)

This feature allows you to carry out the entire computer cycle in the interval between obtaining data on the current value of the operational process parameters of drainage from the corresponding sensors and ADC. Consequently, the proposed control system can be used in real conditions of ore preparation at ore mining and processing plants.

Conclusions. Thus, the conducted studies confirm the appropriateness of the use of block-oriented structures, which include Laguerre OBF, in the tasks of design and practical implementation of automated high-accuracy controllers. The high level of reconstruction of the technological object output signal allows the use of these nonlinear models in the real conditions of mining and concentrating production, where the plant is affected by uncontrolled disturbances caused by fluctuations of technological parameters, properties of iron ore raw materials, obstacles in data transmission channels and so on.

The method of model predictive control formation of the ore crushing process is proposed, which is based on inverting the static nonlinearities of the input-output of the block-oriented model and approximating the control trajectories by orthonormal basis Laguerre functions. This approach allows us to reduce the predictive control problem to the quadratic programming problem, and thus reduce the time of computational operations.

A comparative analysis of the control quality and performance of the system implementing the proposed method, with the system of nonlinear model predictive control, was conducted. It is established that with the same settings of comparable controllers and identical disturbances, the developed system allows to provide 12.5% and 11.9% less overshoot by the coefficient of variation of size density function and control size class output, at 6.3 and 14.7 seconds, the shorter settling time under the corresponding regime parameters and in 164 times (1.19 milliseconds) the lower time of control forming. It should be noted that the proposed control system has a lower steady-state error.

Further research will be devoted to the practical implementation of the proposed adaptive control system for the ore crushing process.

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EFFICIENCY OF USING MAGNETIC SEPARATION FOR THE PROCESSING OF METAL-CONTAINING BASALT RAW MATERIALS

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Abstract

The work has experimentally established a high efficiency of magnetic separation in the part of the output of the magnetic product from all three constituents of basalt raw materials - tuff, basalt and lavobrekchia. This indicates the expediency of including in the technological scheme a complex waste-free processing of basalt raw materials and a magnetic separation operation for separation of iron oxides (magnetite, titanomagnetite) from the pulverized mass. In general, the use of dry magnetic separation at the stage of ore preparation of basaltic raw materials (tuff, basalt, lavobrekchia) in order to extract copper into the non-magnetic fraction is very promising.

Introduction

The object of research was the rocks of the Rafalovsky basalt quarry. The purpose was to determine the manifestation of the magnetic properties of all three main constituents of basalt raw materials in the deposit (tuff, lavobrekchia and basalt) and to establish, on the basis of mineralogical and granulometric analysis of nutrition and magnetic separation products, the technological characteristics of the magnetic separation operation, in particular, the distribution of copper content and recovery in products of magnetic separation.

The research tasks included to consider how efficiently on the magnetic separators copper minerals can be separated into the tailings of magnetic separation and at what size of feed the extraction of copper into the tailings of magnetic separation will be the highest.

That is, the question of the application of magnetic separation for the purpose of concentrating copper minerals in the tailings of separation was investigated. The question of the application of magnetic separation for its intended purpose - for obtaining iron concentrate - was not considered, because for technology it is secondary.

Experimental studies were carried out on the basis of the raw material base of the Rafalovsky basalt quarry as one of the promising for complex processing of the rock mass with the subsequent extraction of native copper, its oxidized and sulphide formations, with the further use of rocks free of ore minerals. The unique properties of the rocks of the Rafalovsky quarry (the presence of native iron and copper species) suggest that their magnetic separation will be effective, and at a sufficiently high feed size, unlike conventional ferruginous quartzite, which require a high degree of grinding before magnetic separation (up to 95% of the class minus 0.044 mm).

To increase the reliability of the experimental results, samples for studies were taken in ten different sections of the blasted mining mass of the quarry, after which they were mixed, and for the separation experiments an averaged mixture was used for each of the three components of the deposit.

The preparation of the samples for the studies consisted of their preliminary crushing and grinding to a size class of less than 3 mm, in accordance with the recommendations for the dry magnetic separation of weakly magnetic ores [1-3]. The crushed rock mass was classified into four classes of coarse size, and in each of the classes a magnetic part (in two or three levels) and a non-magnetic part in a weight and percentage ratio to the sample weight were determined. The studies were carried out under laboratory conditions on a PBSU-0.5 / 0.2 drum magnetic separator in the course of dry magnetic separation. Mineralogical analysis was performed separately for the magnetic and non-magnetic parts of the sample. The content of native copper in each sample was estimated.

Thus, the performed studies of basalt, lavobrekchia and tuff of the Rafalovsky's field showed the expediency of further investigation of the magnetic separation operation, since all three most characteristic rocks give a high yield of magnetic concentrate: from the basalt, a magnetic product of 55% is obtained, 33% from lavobrekchia, 54 from tuff % [4-7].

1. Determination of the regularities of magnetic separation of the constituent basaltic raw materials

1.1. Dry magnetic separation of basalt

The primary experimental materials on the magnetic separation of the basalt sample studied and the calculated values of the dry magnetic separation of the sample are given in Table. 1.

From Table 1, it can be seen that the initial feed for separation was prepared as follows. The sample was crushed and screened into 4 small particle size classes, with the basalt crushing performed in such a way that the yield of each of the four classes of coarse size was approximately the same, about 20-30%.

The characteristics of magnetic separation of basalt are characterized by the fact that for the two upper large classes there is a high yield of the nonmagnetic fraction.

At the same time, the copper content in the non-magnetic fraction of large classes is very high: 13% and 15% (the last figure corresponds to the minimum condition for the finished copper concentrate).

Analysis of copper recovery confirms that copper is mainly extracted from large classes $-2.5 + 0.8$ mm. For them, the total recovery is: $48.4 + 34.8 = 83.2$ (%). The rest of the extraction, $11.5 + 5.3 = 16.8$ (%), accounted for the fraction of small classes $-0.8 \dots -0.25$ mm [3-5].

However, it is noteworthy that the largest upper class $-2.5 + 1.6$ mm is poorly separated: the yield of tailings is small (yield of a non-magnetic fraction is $\sim 5\%$ versus 16.62% in magnetic fraction), and copper extraction in both products is small (22% in magnetic and 26% in non-magnetic). That is, copper, contained in the upper large class, is divided, approximately, equally between magnetic and non-magnetic products.

Table 1

Evidences of dry magnetic separation of narrow basalt classes

Size class, mm	Product	The weight, 10^{-3} kg	Exit to class size, %	Exit from the original, %	Content Cu, %	Extraction Cu to class size, %	Extraction Cu from the original, %
-2,5+1,6	concentration 2	66	75,9	16,62	3,5	45,8	22,2
	non-magnetic	21	24,1	5,29	13,0	54,2	26,2
Total for the class		87	100,0	21,91	5,79	100,0	48,4
-1,6+0,8	concentration 1+2+3	87	79,8	21,91	0,37	9,0	3,1
	non-magnetic	22	20,2	5,54	15,0	91,0	31,7
Total for the class		109	100,0	27,46	3,33	100,0	34,8
-0,8+0,25	concentration 1+ 2	36	31,0	9,07	0,0014	0,04	0,0
	non-magnetic	80	69,0	20,15	1,5	99,96	11,5
Total for the class		116	100,0	29,22	1,03	100,0	11,5
-0,25	concentration 1+2	30	35,3	7,56	0	0	0
	non-magnetic	55	64,7	13,85	1,0	100,0	5,3
Total for the class		85	100,0	21,41	0,65	100,0	5,3
Total in the trial		397		100,0	2,624		100,0

In all smaller classes, -1.6 mm, the trend is different: the extraction of copper into a non-magnetic product is stably higher than in a magnetic product.

From this it follows that for the concentration of copper in the tails, it is advisable to maintain a feed size of not more than 1.6 mm.

The results of the dry magnetic separation of basalt are illustrated in Table 2, from which it can be seen that there is practically no copper in the magnetic product of small classes -1.6 mm. The magnetic fraction is obtained rich in copper only because of the presence in the feeding of separation of large classes of +1.6 mm.

Table 2

Summary data on the magnetic separation of basalt (according to Table 1)

Size class, mm	Exit from the original, %	Content C_u , %	Extraction C_u , %	magnetic, %			non-magnetic, %		
				Exit	Content	Extraction	Exit	Content	Extraction
-2,5+1,6	21,91	5,79	48,39	16,62	3,5	22,2	5,29	13,0	26,21
-1,6+0,8	27,46	3,33	34,80	21,91	0,37	3,1	5,54	15,0	31,68
-0,8+0,25	29,22	1,03	11,53	9,07	0,0014	0,0	20,15	1,5	11,52
-0,25	21,41	0,65	5,28	7,56	0,0	0,0	13,85	1,0	5,28
Total	100,0	2,62	100,0	55,16	1,20	25,3	44,84	4,37	74,70

The question arises, how can copper fall into the magnetic fraction (especially in large classes), if all copper minerals are nonmagnetic? There are two reasons for this.

The first is as follows. Despite the fact that mineralogy clearly shows that the deposit is rich in native and oxidized (not sulphide) copper, sulphide copper, that is, chalcopyrite, and, more importantly, usually the chalcopyrite minerals pyrite and pyrrhotite, are present in basalt raw materials. Although pyrite and chalcopyrite (copper pyrite, $CuFeS_2$) are nonmagnetic, but pyrrhotite FeS_2 or magnetic pyrite is a strongly magnetic mineral. It is extracted during separation, and together with it, in the form of splices, copper minerals are extracted.

The second reason for the extraction of copper into the magnetic concentrate of large classes is that copper splices with magnetic minerals are extracted: native iron, magnetite and titanomagnetite, as well as splices of iron and copper sulfides, for example, bornite Cu_3FeS_2 . Mineralogical analysis showed the presence of all these

minerals in the rocks of the Rafalovsky quarry. With them, in fact, a high yield of magnetic fraction is associated with separation.

The main conclusions on the magnetic separation of basalt are as follows:

- with a feed size of $-2.5 + 0$ mm, there is a high yield of the magnetic fraction -55 (16%);
- there is a concentration of copper in the tails of magnetic separation, namely, the amount of copper in the tails increases by 1.7 times compared with the content of copper in the original (from 2.6 % to 4.4 %).

However, both products are conditioned by the content of copper (Table 2), which indicates the insufficient opening of copper minerals in the diet and the need to reduce its size.

Magnetic separation of basalt showed the possibility of copper concentration in the tails. However, from the position of iron concentration, the high yield of the magnetic fraction obtained is more a disadvantage than dignity, since it is obtained on relatively rough nutrition, and naturally, together with a large mass of the magnetic product, many clusters and gangue are attracted. Due to this, the magnetic product will be poor in iron (less than 50% Fe in the experience of magnetic enrichment of iron ores). It requires refinement, i.e., grinding and repeated magnetic separation.

To increase the extraction of copper in the tails of magnetic separation, the achieved increase in copper in the tails (by a factor of 1.7) can be increased (up to a factor of 2 to 3 times), if the size of the food is reduced, at least remove the upper large class from the separation feed, that is, granulate basalt to a size of -1.6 mm. At the same time, more fine grinding is also achieved in improving the quality of the magnetic product in terms of iron content, since, according to the experience of iron ore Mining Concentrates, the opening of iron minerals is achieved with a very fine grinding - up to 95% of the class minus 0.05 mm.

1.2. Dry magnetic separation of lavobrekchia

The results of calculating the yield, content and extraction of copper in concentrate and tailings of dry magnetic separation of lavobrekchia are given in Tables 3 and 4 (summary).

Table 3

Indices of dry magnetic separation of narrow classes of lavobrekchia

Size class, mm	Product	The weight, 10^{-3} kg	Exit to class size, %	Exit from the original, %	Content C_{Cu} , %	Extraction C_{Cu} to class size %	Extraction C_{Cu} from the original %, %
-2,5+1,6	magnetic	40,0	40,0	12,5	1,16	34,1	10,7
	non-magnetic	60,0	60,0	18,8	1,5	65,9	20,7
Total for the class		100,0	100,0	31,3	1,37	100,0	31,3
-1,6+0,8	magnetic	32,0	40,5	10,0	0,33	8,2	2,4
	non-magnetic	47,0	59,5	14,7	2,5	91,8	27,0
Total for the class		79,0	100,0	24,7	1,62	100,0	29,4
-0,8+0,25	magnetic	34,0	37,8	10,6	0,16	24,3	1,2
	non-magnetic	56,0	62,2	17,5	0,3	75,7	3,9
Total for the class		90,0	100,0	28,1	0,25	100,0	5,1
-0,25	magnetic	16,0	31,4	5,0	2,75	29,5	10,1
	non-magnetic	35,0	68,6	10,9	3,0	70,5	24,1
Total for the class		51,0	100,0	15,9	2,92	100,0	34,2
Total in the trial		320,0		100,0	1,36		100,0

From Table 3 it can be seen that the yields of all classes differed slightly in the initial feed, namely: in order of decreasing the size, ~ 31 %, 25 %, 28 %, and for the smallest class -0.25 mm - 15.9 %.

Separation power is also characterized by the fact that the output of the smallest class -0.25 mm was less than all others (15.9 %), but it is characterized by the largest copper content - 2.92 %. This is twice as high as the copper content in the initial 1.36 %. This is connected with the fact that the extraction of copper in this class is maximum and equal to 34.2 %. Concerning the separation products obtained, we note the following.

When separating the smallest class -0.25 mm, the highest yield of tailings is 68.6 % and the highest copper content in the tails is 3%. However, the content of copper in the largest and in the smallest classes, that in the magnetic, that in a nonmagnetic product is approximately the same. This suggests that these classes contain a large number of splices of copper with ferruginous minerals.

The best copper recovery in a non-magnetic product is observed for two medium size classes of $-1.6 + 0.8$ mm and $-0.8 + 0.25$ mm, where they are 91.8 % and 75.7 %, respectively. Apparently, this size corresponds to the size of the native formations of copper.

For lavobrekchia, it is characteristic that for any class of coarseness the yields of tails are naturally higher than the yield of the magnetic product. With a decrease in particle size, the yield of tailings is markedly reduced, and the content of copper in the tail increases, the extraction of copper into tailings is reduced insignificantly (Figure 1).

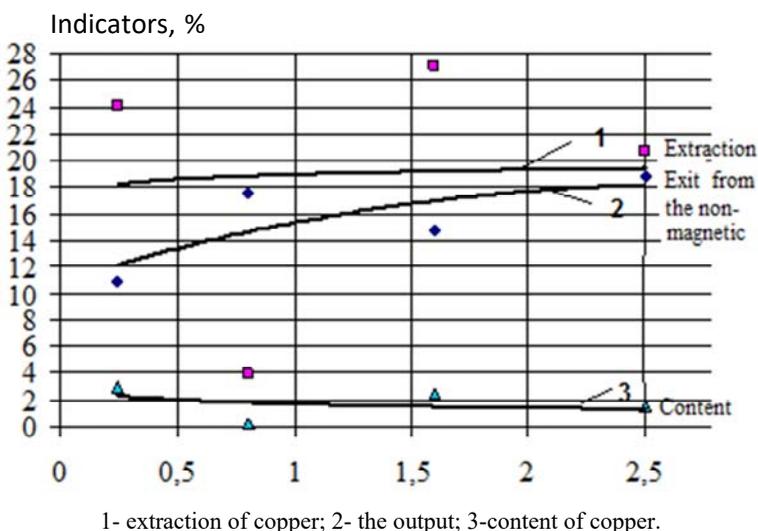


Fig. 1. Distribution of indicators of tails of magnetic separation lavobrekchia by size

All the dependences of Fig. 1 - weak (the square of the correlation coefficient is not high enough) and therefore are of an illustrative nature. The important thing is that for 3 depending tendency increasing copper content in the tailings to reduce their size. This suggests that the minerals of raw materials are better exposed in small classes and therefore are better separated during separation. That is, it is advisable for lavobrekchia to reduce the size of the supply of magnetic separation.

For clarity of further analysis according to Table 3 was compiled summary Table 4.

Table 4

Summary table of the results of dry magnetic separation of lavobrekchia

Size class, mm	Exit from the original, %	Content Cu, %	Extraction Cu, %	magnetic, %			non-magnetic, %		
				Exit	Content	Extraction	Exit	Content	Extraction
-2,5+1,6	31,25	1,37	31,3	12,50	1,16	10,67	18,75	1,50	20,66
-1,6+0,25	52,81	0,89	34,5	20,63	0,24	3,65	32,19	1,30	30,8
-0,25	15,94	2,92	34,2	5,00	2,75	10,10	10,94	3,00	24,10
Total	100,0	1,36	100	38,13	0,87	24,42	61,88	1,66	75,58

According to Table 4, the following conclusions can be drawn:

1) The yield of magnetic and nonmagnetic fractions from the lavobreakia test was 38.13 % and 61.88 %, respectively. Moreover, for all narrow classes of size, the amount of the non-magnetic fraction is stably higher than that of the magnetic product.

2) For this sample, the content of copper in the tails is not much higher than in the feedstock (1.66 % vs. 1.36 %). Both fractions, both magnetic and nonmagnetic, proved to be sufficiently rich in copper. They contain 0.87% and 1.66% copper, respectively, that is, both require a clean-up.

This situation is a consequence of the insufficient extraction of copper in the tails of the largest and of the smallest classes, namely, for classes larger than 1.6 mm and finer than 0.25 mm, the content of copper in the magnetic and non-magnetic fractions was approximately equal.

3) Copper extraction into the nonmagnetic fraction as a whole over the lavobrekchia sample is higher than in the magnetic fraction: 75.6 % versus 24.4 %.

Relatively high copper recovery into concentrate (24.2 %) occurs equally because of the copper-rich coarse (classes +1.6), and because

of the rich fines (classes -0.25). Most likely, both these classes contain a lot of splices of copper and iron minerals, with which copper enters a magnetic product. The lowest copper recovery in the magnetic concentrate occurs for a narrow class of fineness $-1.6 + 0.25$ mm [8].

Based on the above described for the process of magnetic separation of lavobrekchia, the following recommendations can be made.

First, it is advisable to get rid of the upper class of size, that is, to crush the whole material to a size of less than 1.6 mm.

Then there are two possible variants of the technology:

1) Select (sow) a narrow class of fineness $-1.6 + 0.25$ mm and send it to a magnetic separator. Then the magnetic product of the separator can be withdrawn from the beneficiation by copper, since, according to Table. 4.8, the content of copper in it is unconsolidated - 0.24%. The tails of the magnetic separator together with the lower class of crushed ore -0.25 mm should be sent for electrical separation.

2) The whole product should be mined to -1.6 mm and subjected to electrical separation. By its results, decide which of the products (or both) to subject to magnetic separation and further enrich them separately.

If the whole product is crushed to a size of -1.6 mm, then, naturally, the yield of small classes will increase -0.25 mm. As can be seen from Table 5, about 2/3 of these classes are released into the tail, but the copper content in the tail and concentrate for these classes is approximately the same. Most likely, the reason for the presence of clusters of ferruginous and copper minerals in classes is - 0.25 mm. Deeper grinding of lavobrekchia, up to 0.05 mm, will allow to get rid of such splices and increase the magnetic separation in the part of copper release into tails.

1.3. Dry magnetic separation of tuff

Studies of tuffs conducted by the Institute of Geotechnical Mechanics. N.S. Polyakov Institute of National Academy of Sciences of Ukraine together with the National University of Water and Environmental Engineering (Rivne), showed that the content of

native copper is up to 0.45 ÷ 0.7 %, and an unexpectedly high content of magnetic susceptible material in tuffs (by weight up to 54 %). As a result of the spectral analysis, the presence of up to 50 % iron and up to 4.0 % titanium in tuff is established [9-11].

The calculation of the technological indexes of magnetic separation of tuff in the raw materials (hereinafter referred to as sample 1) is presented in Table. 5.

Table 5

Indicators of dry magnetic separation of tuff, sample 1

Size class, mm	Exit from the original, %	Content Cu, %	Extraction Cu, %	magnetic, %			non-magnetic, %		
				Exit	Content	Extraction	Exit	Content	Extraction
-2,5+1,6	17,2	0,09	52,1	9,9	0,2	33,5	7,3	0,007	4,4
-1,6+0,25	63,8	0,00	0,0	35,3	0,0	0,0	28,5	0,0	0,0
-0,25	19,0	0,07	47,9	9,4	0,1	21,0	9,6	0,05	41,0
Total	100,0	0,03	100,0	54,6	0,04	54,6	45,4	0,01	45,4

The test tufa sample has a very low copper content of 0.03 %. According to accepted norms such raw materials are not subject to industrial processing with the purpose of copper production. Therefore, we do not analyze the extraction of copper into separation products. However, according to Table. 5, the following features of the fragmentation of the sample and its magnetic separation can be noted:

- with the chosen method of crushing, more than half the mass of tufa (63.8%) goes to the average size class -1.6 + 0.25 mm, the rest is divided approximately equally between the upper -2.5 + 1.6 mm and the lower -0 , 25 mm classes. This distribution is a consequence of the fact that tuff has a low density (1.3 kg / m³) and is easily destroyed. For the sample examined, copper is absent in the middle class, copper traces are contained only in the upper coarse and in the lower small classes;

- tuff in all narrow classes of size is well separated by magnetic separation. Thus, for two large classes $-2.5 + 0.25$ mm, the yield of the magnetic fraction is greater than that of the nonmagnetic fraction: slightly more than half the mass of the feedstock falls into the magnetic product. The fine classes -0.25 mm are divided approximately equally into the magnetic and nonmagnetic fractions.

It was shown in [104-108] that zeolite-smectite tuffs provide a yield of a magnetic product in an amount of 49 % of the sample mass. Tuff contains (in intergrowths) 35-40% of iron and 2.5-4.0% of titanium. The remaining silicate part contains 0.4-0.7% copper, which indicates the advisability of complex processing of tuff.

This conclusion is confirmed by the data in Tables 6 and 7, which show the results of dry magnetic separation for another tuff sample, designated below as sample 2 (calculations were made by experimental data [9-11]).

Table 6

Output of products of magnetic separation of tuff, sample 2

Size class, mm	Total in the trial		Exit from small size class %				Exit from the original, %	
			magnetic		non-magnetic		magnetic	non-magnetic
	g	%	g	%	g	%	%	%
$-2,5+0,63$	331,4	36,1	275	83,0	56,4	17,0	29,94	6,14
$-0,63+0,1$	208,9	22,7	175,4	84,0	33,5	16,0	19,09	3,65
$-0,1$	378,3	41,2	0	0	378,3	100	0	41,18
Total	918,6	100	450,4		468,2		49,03	50,97

Table 7

Technological characteristics of magnetic separation of tuff, sample 2

Size class, mm	Exit from the original, %	Content C_u , %	Extraction C_u , %	magnetic, %			non-magnetic, %		
				Exit	Content	Extraction	Exit	Content	Extraction
$-2,5+0,63$	36,1	0,31	21,01	29,94	0,29	16,38	6,14	0,40	4,63
$-0,63+0,1$	22,7	0,45	19,26	19,09	0,42	15,13	3,65	0,60	4,13
$-0,1$	41,2	0,77	59,83	0,00	0,00	0,00	41,18	0,77	59,8
Total	100	0,53	100	49,03	0,34	31,5	50,97	0,71	68,6

As can be seen from Table 6, the classes +0.1 mm more than 80% entered the magnetic product, and the thin classes -0.1 mm turned out to be nonmagnetic and completely fell into the tails. The distribution of the yields of the products from the initial in sample 2 was 49 % magnetic and 51 % non-magnetic.

According to Table 7 for sample 2, the following conclusions can be drawn:

- The magnetic fraction is represented only in large classes -2.5 + 0.1 mm;

- small classes of tufa -0.1 mm have the highest copper content - 0.77 %;

- extraction of copper into a non-magnetic product was 68.6 %, which is lower than for basalt (74.7 %) and lavobrekchia (75.58 %), with the same feed grain size of the magnetic separator -2.5 + 0 mm [10, 11];

- the content of copper in the tails is slightly higher than the content of copper in the initial feed - 0.71% compared to 0.53%.

Let us compare the separation indices for two tested tuff samples.

1) The ratio of yields of products (magnetic to non-magnetic) for sample No. 1 was 54.6 % to 45.4 %, for sample No. 2 49 % to 51 %, respectively. On average, for the two samples studied, when feeding the separator in classes of -2.5 mm, the yield of the magnetic fraction was $(54.6 + 49) / 2 = 51.8$ %.

The copper content in the first sample was substandard - 0.03 %, in the second sample - technologically acceptable 0.53 %. From this we can conclude that, regardless of the copper content in magnetic separation, the mass of the tufa sample is reduced, approximately, by a factor of two.

2) The highest copper recovery in a non-magnetic product is characteristic for shallow -0.25 mm and fine grades -0.1 mm tuff. Particles of small classes -0.25 mm are divided approximately equally into magnetic and nonmagnetic products (sample 1), and the fine classes -0.1 mm almost completely fall into tails (sample 2). The non-magnetic product of classes -0.1 mm has the highest copper content.

The performed studies have shown that tuff not only easily breaks down during crushing, but, regardless of the copper content, it

is well separated on a magnetic separator. The highest copper recovery in a non-magnetic product is typical for shallow -0.25 mm and thin -0.1 mm feed classes.

Given the variation in the copper content in the original, during the processing of tuff, constant monitoring of the copper content in the initial one should be carried out: it should not be less than 0.35%. At non-standard copper content tuff is a valuable raw material for production, iron - titanomagnetite concentrate (recoverable minerals: magnetite, titanomagnetite, pyrrhotite), tails can be used for agriculture.

Let's sum up the magnetic separation of basalt, lavobrekchia and tuff with the grain size of the separator less than 2.5 mm.

1) For basalt, the yield of the magnetic fraction was ~ 55%, the extraction of copper into the non-magnetic product (tails) was 74.7%. The content of copper in the initial - 2,62%, in the tails - 4,4%, in the magnetic product -1,2%. Thus, there is an increase in the content of copper in the tails compared with the initial supply from 2.6% to 4.4%, i.e., 1.7 times. However, both magnetic and non-magnetic products are conditioned by the copper content, which indicates a lack of disclosure of copper minerals and the need to reduce the feed size, at least from -2.5 mm to -1.6 mm.

2) For lavobrekchia, the yield of the magnetic fraction was ~ 38 %, copper extraction in the tailings was 75.58 %. The copper content in the initial - 1.36 %, in the tails -1.66%, in the magnetic product - 0.89 %.

The content of copper in the tailings has increased little compared to the initial, both separation products (and concentrate, and tails) have a high copper content, which also indicates insufficient opening of the rock minerals, and both products are further enriched in copper.

The highest copper extraction in tailings is achieved for a narrow size of lavobrekchia - classes -1.6 + 0.25 mm. Here, the recovery into the non-magnetic product is 30 %, versus 3 % - in the magnetic (from the initial supply). However, this does not mean that it is necessary to isolate and separate this narrow class. On the contrary, it is easier to reduce the size of the entire feedstock from -2.5 mm to -1.6 mm, and then either to separate the product as a whole or to separate the coarse fraction by sending small classes -0.25 mm into a

separate (slime) chain of enrichment. It is possible that due to this reduction in the size, it will be possible to obtain a magnetic product that is poor in copper and to remove it from the copper redistribution, and also to achieve a higher concentration of copper in the tails. If not, then you need to further reduce the size of the supply of magnetic separation.

3) For tuff, the yield of the magnetic fraction on the average for two samples was $(54.6 + 49) / 2 = 51.8$ %, copper extraction into tails - 68.6 %. The content of copper in the initial - 0.53 %, in the tails - 0.71 %, in the magnetic product - 0.34 %.

From this it can be seen that of the three rocks studied, tuff has the lowest copper content and extraction into tails. Unlike other rocks, the magnetic product of tufa separation cannot be further enriched in copper and can be removed from copper processing.

It is established that the extraction of copper into tails is accounted, in the main, of small tuff classes. Thus, for the sample 1, the copper content in the initial sample was sub-standard, the copper content analysis was not carried out, however, it was found that the copper recovery in the tailings was 45.4 %, of which 41 % accounted for the fraction of small classes -0.25 mm. For sample 2 it is confirmed that copper is mainly concentrated in small classes. In the case under study - in classes -0.1 mm. These classes completely separated into tails with magnetic separation. They are the richest in copper. Copper recovery for these classes was 59.8 % with a total copper recovery of 68.6 % in the non-magnetic product.

Thus, the crushed tuff is well separated by magnetic separation and at a feed size of $-2.5 + 0.25$ mm, slightly more than half the mass enters the magnetic product, at a smaller size of -0.25 mm the raw material is divided approximately equally into the magnetic and non-magnetic parts. The larger tuff classes in the magnetic product contain up to 39-39 % iron (in the intergrowths).

The obtained results made it possible to determine the type of equipment: at a feed size of $-2.5 + 0.1$ mm, it is expedient to use conventional magnetic separators, for example, of the PBS type. For smaller feed size, most likely, electromagnetic separators with a higher field strength, for example of the EMU type, will be required. To clarify the choice of the separator type, the effect of

magnetic field induction on the separation of tuff, basalt, and lavobrekchia was further investigated.

2. Determination of the effect of magnetic field induction on the separation indices of the components of basalt raw materials in magnetic separation

To fully describe the process of magnetic separation and generalization of the results obtained, the effect of magnetic field induction on the yield of magnetic concentrate was investigated. Two classes of fineness $-2.5 + 0.63$ mm and $-0.63 + 0.1$ mm were used as feed for the separation, which were separated from the crushed product by a vibration by a crust of -2.5 mm in size. The induction of the magnetic field varied in the range $0.08 \div 1.3$ T.

Analytical calculations of the yield of magnetic concentrate are given in Table 8.

Table 8

The yield of the magnetic separation concentrate (%) from the rocks of the basalt quarry with different induction of the magnetic field

Induction , Tl	Tuff				Basalt				Lavobrekchia			
	-2,5+0,63 mm		-0,63+0,1 mm		-2,5+0,63 mm		-0,63+0,1 mm		-2,5+0,63 mm		-0,63+0,1 mm	
	%	Σ, %										
0,08	19,1	19,1	14,6	14,6	19,3	19,3	14,5	14,5	14,1	14,1	11,8	11,8
0,16	17,9	37,0	17,7	32,3	20,0	39,3	17,8	32,3	16,6	30,7	14,7	26,5
0,30	15,6	52,6	18,1	50,4	17,8	57,1	20,3	52,5	17,7	48,4	17,8	44,3
0,44	15,0	67,6	15,3	65,7	16,6	73,7	19,3	71,8	18,8	67,2	20,4	64,7
0,58	13,5	81,1	15,6	81,2	15,8	89,5	15,2	87,0	17,3	84,5	16,0	80,7
1,30	1,9	83,0	2,8	84,0	2,8	92,3	3,8	90,8	5,5	90,1	7,2	88,0
non-magnetic	17,0	100	16,0	100	7,7	100	9,3	100	9,9	100	12,0	100
Total	100		100		100		100		100		100	

With a reliability of 0.95, the mean quadratic error in determining the yield was in the confidence interval 0.5-1.2 %.

The standard method of pair correlations was used to determine the analytical dependences of the yield of the magnetic concentrate on the field induction. At the same time, using the capabilities of the Microsoft Office Excel program, out of 6 possible approximating curves one was chosen, proceeding from the conditions for the presence of a physical sense for this dependence and the maximum of the reliability coefficient of approximation R^2 . The size of a private narrow class was given as the arithmetic mean of the boundary values, which is the most widely used approach.

The dependences of the yield distribution of the concentrate on the induction of the magnetic field for two feed sizes are shown in Fig. 2, 3.

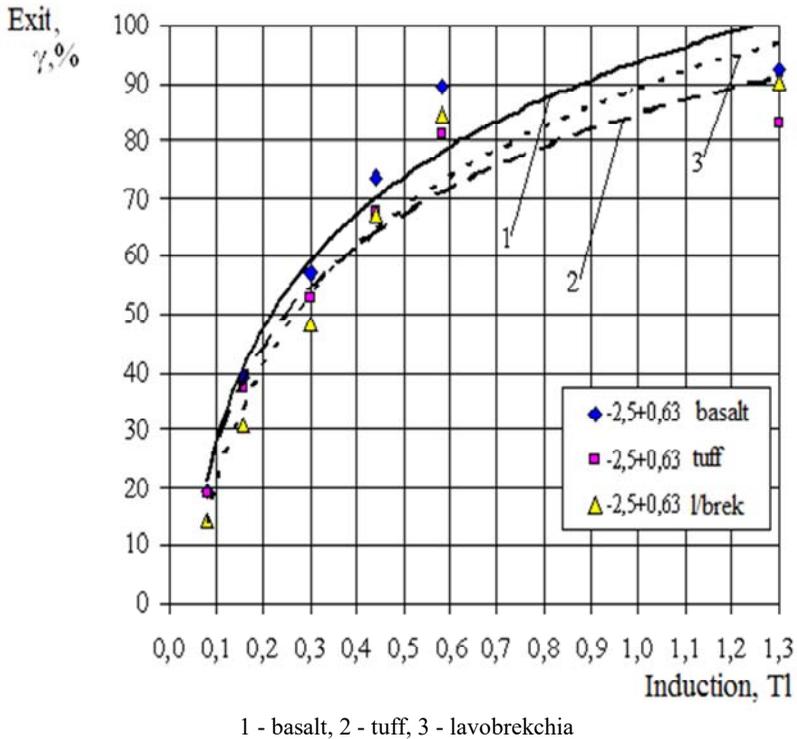
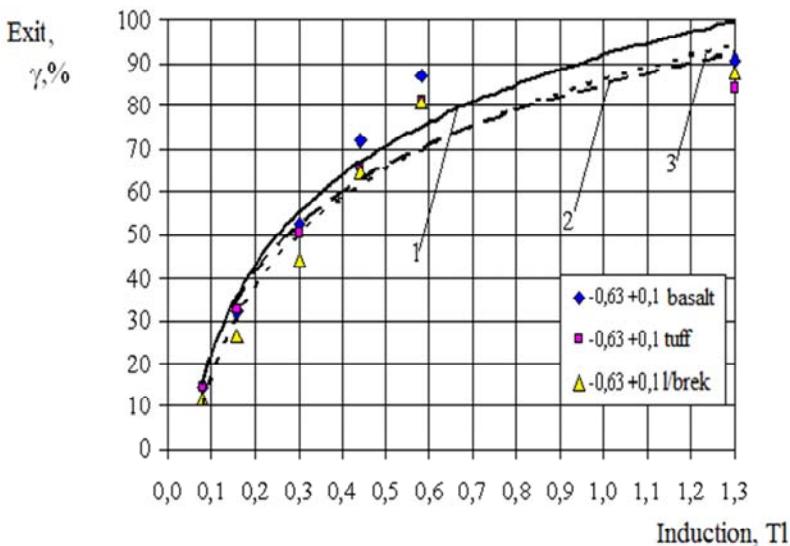


Fig. 2. Feed size $-2.5 + 0.63$ mm. Dependences of magnetic concentrate yield on field induction



1 - basalt, 2 - tuff, 3 - lavobrekchia

Fig. 3. Feed size -0.63 + 0.1 mm. Dependences of magnetic concentrate yield on field induction

Note that when choosing an approximating relationship, it was found that the second-degree polynomial gives the highest value of R^2 , but this dependence has no physical meaning, since at a certain site (0.7-1.2 T) the calculated yield exceeds 100 %. Therefore, in Fig. 2, 3, and also below, for similar dependences of the output from induction, we chose the approximating function in the form of logarithmic dependence.

The following correlation equations are obtained:

- for a relatively large class -2.5 + 0.63 mm (Fig. 2):

1 - basalt: $\gamma = 28.6 \ln(B) + 93.63, R^2 = 0.94;$

2 - tuff: $\gamma = 25.08 \ln(B) + 84.59, R^2 = 0.94;$

3 - lavobrekchia: $\gamma = 29.95 \ln(B) + 89.1, R^2 = 0.95;$

- for a smaller class -0.63 + 0.1 mm (Fig. 3):

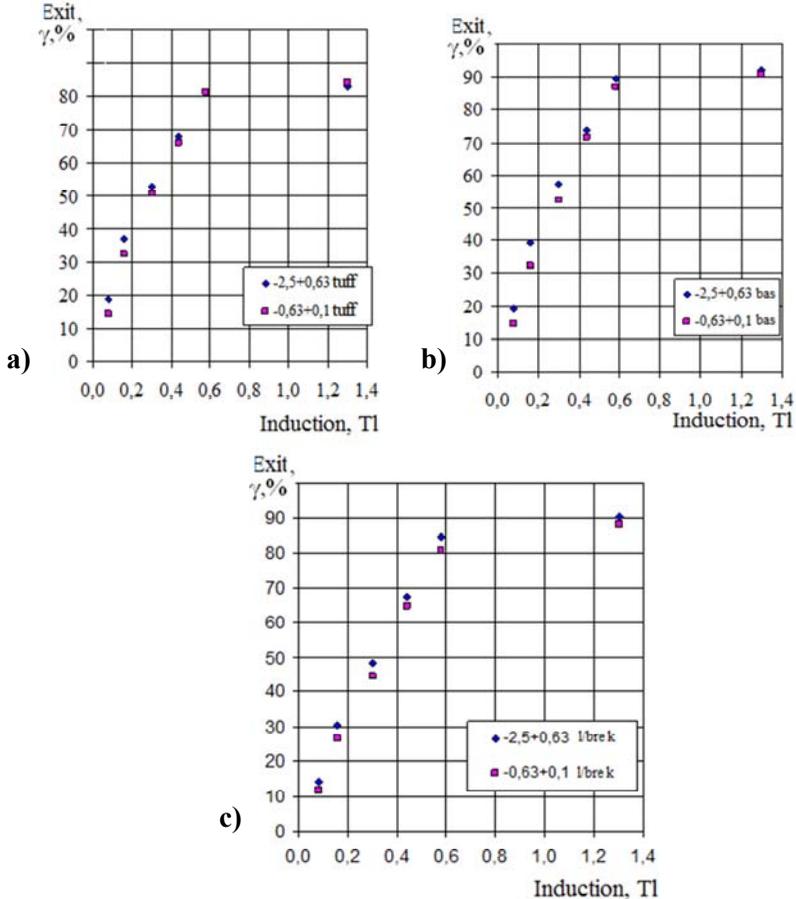
1 - basalt: $\gamma = 30.28 \ln(B) + 91.75, R^2 = 0.94;$

2-tuff: $\gamma = 27.32 \ln(B) + 85.05, R^2 = 0.94;$

3 - lavobrekchia: $\gamma = 30.15 \ln(B) + 86.16, R^2 = 0.95.$

From Fig. 2 that for the larger class these three rocks have a noticeable spread of dependencies, which increases with increasing field induction. For a smaller class (Fig. 3), this spread of dependencies decreases.

In this connection, the dependencies for each of the rocks for its two narrow classes of food size were separately constructed (Fig. 4).



a) tuff, b) basalt, c) lavobrekchia

Fig. 4. Characteristics of the yield of magnetic concentrate from the rocks of the basaltic quarry depending on the induction of the field for two narrow classes of size: $-2.5 + 0.63$ and $-0.63 + 0.1$ mm

Comparing the data in Fig. 4 it can be seen that for each of the breeds the difference in concentrate yields for the two narrow classes is insignificant, that is reasonably to analyze the broader, cumulative, nutrient-size class of $-2.5 + 0.1$ mm.

For this, according to Table. 8 for each of the rocks, the yield of the concentrate from a wider class of fineness $-2.5 + 0.1$ mm was calculated (Table 9). Then, for this class, the corresponding dependence of the output on the induction of the field was constructed (Fig. 5).

Table 9

The yield of magnetic concentrate from the cl. $-2.5 + 0.1$ mm for the three rocks of the basalt quarry with different induction fields

Induction, Tl	Tuff			Basalt			Lavobrekchia		
	g	%	Σ,%	g	%	Σ,%	g	%	Σ,%
0,08	93,7	17,3	17,3	102,6	17,6	17,6	72,3	13,3	13,3
0,16	96,5	17,9	35,2	111,9	19,2	36,9	86,7	16,0	29,3
0,30	89,4	16,5	51,7	108,5	18,6	55,5	96,0	17,7	47,0
0,44	81,6	15,1	66,9	102,0	17,5	73,0	104,9	19,3	66,4
0,58	77,3	14,3	81,2	90,9	15,6	88,6	91,6	16,9	83,3
1,30	12,0	2,2	83,4	18,2	3,1	91,8	33,1	6,1	89,4
non- magnetic	89,8	16,6	100	47,9	8,2	100	57,6	10,6	100,0
Total	540,3	100		582,0	100		542,2	100,0	

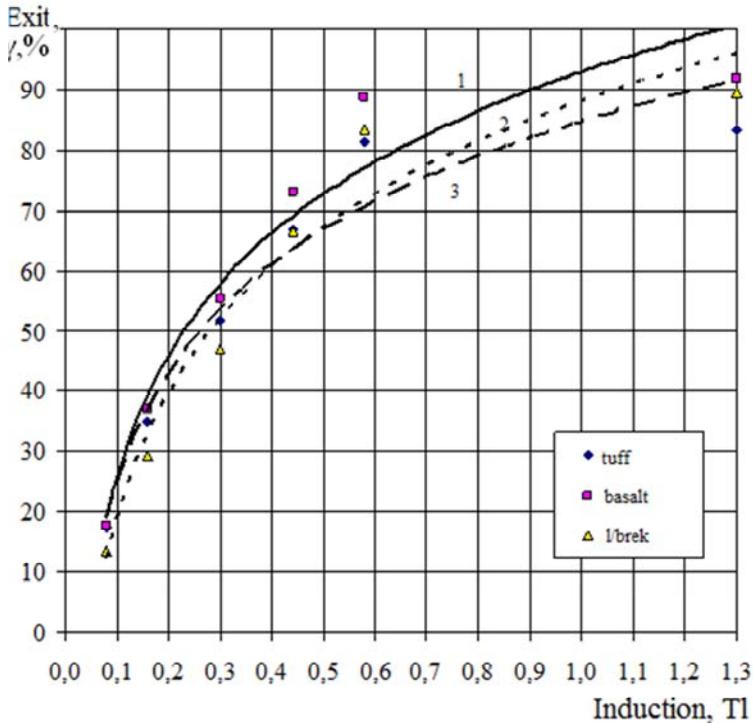
The correlation dependences of the yield of the concentrate from the generalized class of fineness $-2.5 + 0.1$ mm on the induction of the field (Fig. 5) have the following form:

1 - basalt: $\gamma = 29.18 \ln (B) + 92.99, R^2 = 0.94;$

2 - tuff: $\gamma = 25.94 \ln (B) + 84.77, R^2 = 0.94;$

3 - lavobrekchia: $\gamma = 30.02 \ln (B) + 88.13, R^2 = 0.95.$

From Fig. 5 it can be seen that tuff, basalt and lavobrekchia are characterized by fairly close dependences of the yield from induction.



1 - basalt, 2-tuff, 3-lavobrekchia.

Fig. 5. Dependence of concentrate output from cl. $-2.5 + 0.1$ mm from the field induction

This is also confirmed by a simple analysis of the data in Table 9. Thus, at 0.08 T the maximum yield is at basalt, the minimum at lavobrekchia, the difference between them is greatest (among all similar differences) and is $17.6 - 13.3 = 4.3$ % or 30.3 grams, which is within the error of weighing. It follows that one can obtain a generalized model of the distribution function of concentrate output from induction, which is common for all three rocks.

To construct a generalized model, we determine the total yield as the arithmetic mean of the yields of the three rocks (Table 10) and then construct the corresponding generalized dependence characterizing the change in the yield of the magnetic concentrate with increasing magnetic field induction (Fig. 6).

Table 10

Average yield of magnetic concentrate for three rocks with different field induction

Induction , Tl	Tuff	Basalt	Lavobrekchia	Average
	$\gamma \Sigma, \%$	$\gamma \Sigma, \%$	$\gamma \Sigma, \%$	$\gamma \Sigma, \%$
0,08	17,3	17,6	13,3	16,1
0,16	35,2	36,9	29,3	33,8
0,30	51,7	55,5	47,0	51,4
0,44	66,9	73,0	66,4	68,8
0,58	81,2	88,6	83,3	84,4
1,30	83,4	91,8	89,4	88,2

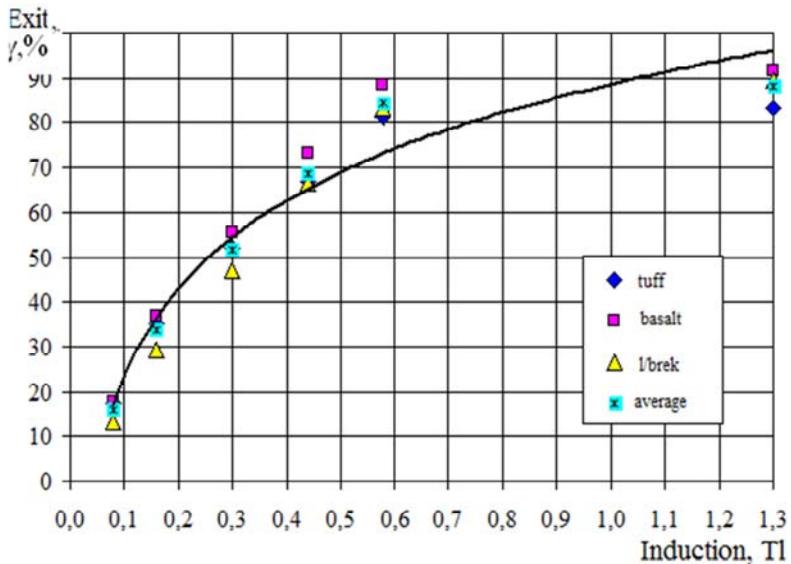


Fig. 6. Generalized dependence of the yield of the magnetic separation concentrate for tuff, basalt, and lavobrekchia on field induction

With a high reliability of approximation, the generalized model of the output of magnetic concentrate from field induction for all three rocks has the form:

$$\gamma = 28,38 \text{ Lg}(B) + 88,629, \quad R^2=0,947 \quad (1)$$

where γ - concentrate yield,%; B - field induction (T).

The validity of this generalized model can be explained by the fact that the magnetic properties of all three rocks of basalt raw materials are approximately the same, and the distribution of the yields of the magnetic product depend little on the size class in the investigated range of nutritional size.

If all three rocks are subjected to magnetic separation co-locally, then the yield of the concentrate with a grain size of $-2.5 + 0.1$ mm can be estimated from the equation of the generalized model (1).

Conclusions

Thus, in the process of the conducted studies, the high efficiency of magnetic separation in the part of the output of the magnetic product from all three constituents of the basaltic raw materials - tuff, basalt and lavobrekchia - has been experimentally established. This indicates the expediency of including in the technological scheme a complex waste-free processing of basalt raw materials and a magnetic separation operation for separation of iron oxides (magnetite, titanomagnetite) from the pulverized mass.

Technological indices of dry magnetic separation of classes $- 2,5 + 0$ mm basalt, lavobrekchia and tuff were determined for the first time in order to reveal the possibility and peculiarities of copper concentration in the tails, which is important for the development of copper recovery technology at the ore preparation stage.

It is obtained that for the basalt there is an increase in the content of copper in the tails compared with the initial supply by 1.7 times. For lavobrekchia the degree of concentration of copper in the tail is insufficient. For both rocks, the magnetic product has a copper grade (above 0.35%) and cannot be removed from the copper processing process. In order to increase the degree of copper concentration in the tailings and to reduce the copper content in the magnetic product, it is advisable to either reduce the size of the feedstock to at least -1.6 mm, or re-micrate and re-separate the magnetic product. This will provide copper-enriched tails and remove a non-copper magnetic product for further extraction of iron (titanomagnetite). In contrast to these rocks, with a dry magnetic separation of tufa with a content of

0, 53 % Cu, copper is almost completely concentrated in the tailings, the extraction of copper into tailings is mainly due to the fraction of small classes -0.1 mm, the richest in copper content. The classes of tuff that enter the magnetic product contain 36-39 % of iron and 2.5-4 % of titanium.

In general, the use of dry magnetic separation at the stage of ore preparation of basaltic raw materials (tuff, basalt, lavobrekchia) in order to extract copper into the non-magnetic fraction is very promising.

For the first time, regression dependences of the yield of the magnetic separation concentrate from tuff, basalt and lavobrekchia on the induction of the magnetic field of the separator were established. Regression models for the change in yield from induction at a feed size of the separator $-2.5 + 0.63$ and $-0.63 + 0.1$ mm have been established, as well as a generalized model with a separator feed size of $-2.5 + 0.1$ mm. For a generalized model, the yield of a magnetic product with magnetic separation of basalt, crust, and tuff at the ore-preparation conditioning fraction $-2.5 + 0.1$ mm statistically significantly changes according to a linear function, the argument of which is the logarithm of the magnetic field induction.

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DEMAGNETIZATION OF FINE FERROMAGNETIC MATERIALS

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Abstract

Analysis of the works on the magnetic separation of minerals in the preparation of ores shows that this process has proven to be successful in the extraction of valuable minerals from waste. As for the enriched product, the primary separation (with the initial content of the valuable mineral $\alpha < 50\%$) gives a significant quality increase. Since the liberated valuable mineral is not removed from further processing, the initial content of the valuable mineral keeps increasing through the stages. On the other hand, due to flocculation, the capture of the non-metallic phase also increases. The probability of capture is proportional to the product of valuable (P_M) and non-valuable ($1-P_M$) minerals content. Hereafter, without the use of special methods, it is not possible to extract these particles from the mass of the concentrate, since the probability of the removal of liberated non-metallic particles asymptotically tends to zero. Thus, it is theoretically impossible to obtain pure magnetite concentrates by magnetic methods that are currently used at iron ore preparation plants.

Production of pure concentrates requires reducing of all particles to zero residual magnetization, i.e. they need to be demagnetized before further non-magnetic separation.

The problems of demagnetization of fine ferromagnetic particles have been studied as long as their magnetic separation [6]. At the moment, in the preparation processes demagnetizing devices with an alternating magnetic field of an industrial frequency of 50 Hz are used [7]. In an alternating field of such a frequency, the floccules of magnetic particles in the form of strands rotate, which reduces their

dimensions. The effect of such a field is judged on the improvement of the selectivity indices compared to control samples that were not subjected to an impact. The measurement of sizes of particle aggregates, or floccules, is carried out either via photo recording or is assessed visually in a transparent bath of a separator with a water flow and a sufficiently small number of particles, which allows observing directly the behavior of these aggregates.

The main condition for the demagnetization of a ferromagnetic body is the stabilization of its position in space with regard to the changing vector of the external magnetic field [9]. It is not possible to fix the position of all individual particles in suspension by some mechanical method. The material body, no matter how small it is, has a mechanical inertia. Mechanical inertia is proportional to the mass of the particle; it is considerably larger than the magnetic one and shows itself already at frequencies of an external alternating magnetic field of several dozens of thousands of Hz.

Thus, if the vector of the external magnetic field outstrips the position of the axis of easy magnetization of the particle, the prerequisites for its partial demagnetization are developed. Moreover, if such an advance is more than 90° , then demagnetization can be carried out to the full.

The purpose of the paper is to study the process of demagnetization of ferromagnetic particles in the suspension flow.

Scientific novelty is in the fact that the dependence of the lag angle of the turn of magnetic particles in the aqueous medium on the angle of the external magnetic field vector is found. This makes it possible to determine the conditions for achieving the demagnetization effect of ferromagnetic particles in the suspension flow.

Research methodology: numerical mathematical modeling and recording of the results of experimental studies by microscopic observations.

Inertia of a mineral particle in an aqueous medium. Suppose that a particle of a spherical shape moves along with the flow of suspension and instantly falls within a magnetic field superimposed on the flow. It starts moving toward the highest gradient. According to Newton's law, (forces are normalized to specific, related to the mass of the particle):

$$\frac{dU_P}{dt} = F_M - F_g - F_\mu, \left[\frac{m}{s^2} \right];$$

where the force of the magnetic action on a particle in an isodynamic field is:

$$F_M = \mu_0 \cdot \chi_P \cdot H_0^2 \cdot c;$$

the force of viscosity of medium:

$$F_\mu = \frac{18 \cdot \mu \cdot U_P}{\delta_P \cdot d_P^2};$$

the force of gravity:

$$F_g = g \cdot \frac{\delta_P - \delta_W}{\delta_P}.$$

Under the action of these forces, the particle moves relatively to the medium and experiences an additional damping action of the viscosity of the medium.

One of the constituent forces depends on the velocity of the particle, the rest are constants; therefore, the equation of the law of a particle motion can be written as follows:

$$\frac{dU_P}{dt} = -\frac{18 \cdot \mu}{\delta_P \cdot d_P^2} \cdot U_P + (\mu_0 \cdot \chi_P \cdot H_0^2 \cdot c) - g \cdot \left(\frac{\delta_P - \delta_W}{\delta_P} \right),$$

where: μ – coefficient of dynamic viscosity of water, [N·s/m²]; $\mu_0 = 1.26 \times 10^{-6}$ – absolute magnetic constant, [N/A²]; d_P – particle size, [m]; δ_P, δ_W – particle and water densities, [kg/m³]; χ_P – specific magnetic susceptibility of a particle, [kg⁻¹]; c – external magnetic field variation factor, [m⁻¹]; H – magnetic field intensity, [A/m].

The solution of this equation is trivial; it shows that the time of the transient process of particle motion is 0.006 s.

The steady-state particle velocity is about 0.097 m/s. During this period, the particle passes a distance equal to its diameter. On this basis, we can conclude that particles in an aqueous medium in case of slowly changing conditions react instantly to a change in the ratio of the forces effecting on them; in terms of demagnetization, the

frequency of the change in the magnetic field should be greater than the value being reciprocal to the period of the transient process.

Methods and devices currently used to demagnetize ferromagnetic particles in a suspension. Historically, the first and most obvious way to study the behavior of ferromagnetic particles in a magnetic field is a visual one. To do this, transparent baths with the application of a magnetic field were created; a pure water flow, containing an insignificant amount of ferromagnetic particles of known physical parameters was supplied. Based on visual observations, the frequencies of rotation of the floccules were recorded as well as their length and thickness. To increase the objectivity of the measurement indicators, video and photo recording as well as scaling of the actual size of the floccules were applied. It was found that at a frequency of about 450 Hz magnetic particles moved in the form of a cloud. The static state of the individual particles was not detected.

Fig. 1 shows the results of measurements of the length of magnetic strands in an alternating magnetic field [5]. As follows from the graphs, the length of the floccules increases with the decrease in a particle size, since it is accompanied by their coercive force growth. An increase in the frequency of the magnetic field reduces the size of the strands. Destruction of strands to the size of individual particles was not obtained.

Paper [7] represents the results of studies of the alternating magnetic field effect on the flow of a ferromagnetic suspension, coming to the magnetic recovery of magnetite concentrate. The authors call the operation of an alternating magnetic field effect on the suspension flow as *demagnetization*. The effect of the impact was evaluated according to the results of improving the quality of the concentrate obtained both without its treatment with an alternating magnetic field, and with such a treatment. Naturally, with the influence of the alternating field, the selectivity indices are improved, since a partial destruction of the floccules occurs. Consequently, additional liberation of constrained released non-metallic particles, which are removed to a depleted product, takes place. The selectivity indices improve with an increase in the intensity and frequency of the magnetic field. Naturally, an increase in the amount of released fraction in the free state in the suspension

reduces the content of a valuable mineral in a depleted product. The greater the suspension density is, the more visible the effect of magnetic treatment by an alternating field is, since the capture of non-metallic particles is proportional to the solid content in the suspension. A significant increase in suspension density leads to a decrease in the mobility of particles in it; thus, starting with a content close to 50%, the effect of magnetic processing again decreases [7].

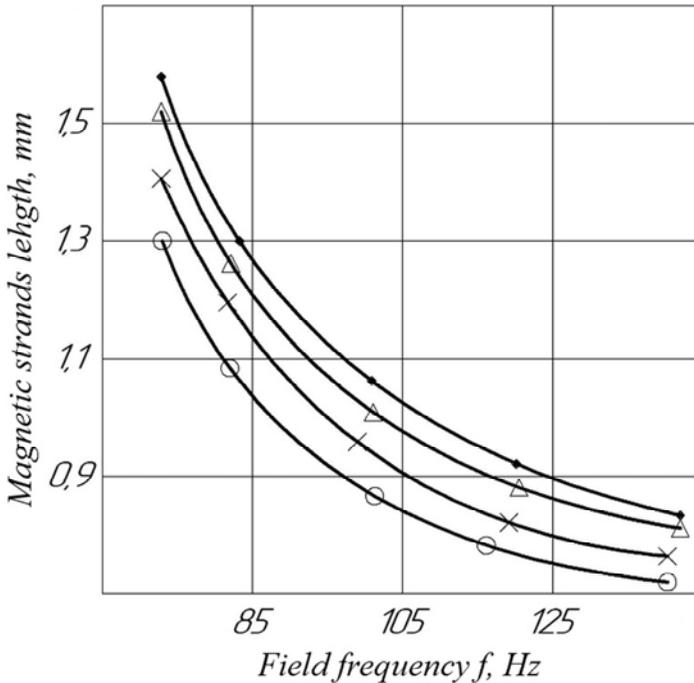


Fig. 1. Dependence of the average length of magnetic strands on the frequency of the traveling magnetic field during the dry separation of magnetite ore of various sizes on a top-feed drum separator: 1 – fraction –53 μm ; 2 – fraction –74 μm ; 3 – fraction –104 μm ; 4 – fraction –147 μm .

Thus, experimental studies, known in the scientific literature, show the effectiveness of preliminary treatment of the suspension flow by an alternating magnetic field. However, there is no valid information about the demagnetization degree of ferromagnetic particles.

At present, demagnetization is carried out in devices that represent a non-magnetic pipeline through which a suspension flows and around which there are mounted electric coils with a reduction in the number of turns. As a result, the intensity of the magnetic field gradually decreases from the saturation intensity to zero. Thus, one demagnetization condition is fulfilled. Thus, the condition for the motionlessness of the particles is not met.

The design and practical use of the applied demagnetizing devices suggest that the demagnetization of particles of the heavy fraction can be carried out in an alternating magnetic field without much influence on the position of the particles. Thus, it is believed that, after leaving the demagnetizing apparatus, the particles of the heavy fraction are demagnetized.

Processing of a suspension by an alternating magnetic field before hydrocycloning [7] has demonstrated that the classification efficiency increases up to 20% with an increase in the demagnetization degree up to 90%. The recovery of iron ore concentrate with the application of an alternating magnetic field gives an increase in the iron content by about 1%.

The condition for the demagnetization of particles is that during the period of the change in the polarity of the magnetic field, the particle does not change significantly its position in relation to the magnetic field vector. It is possible to meet this condition in two cases:

- when the particle is mechanically rigidly fixed in space;
- when the particle does not have time to follow the change in the vector of the external demagnetizing magnetic field.

Consider how an instantaneous fixation of the position of a particle in space is possible.

The change in the intensity of the field, created by a multipolar magnetic system, can be described in the simplest way by functions of the form:

$$H_x = H_0 \cdot \exp(-c \cdot X) \cdot \cos(c \cdot Y);$$

$$H_y = H_0 \cdot \exp(-c \cdot X) \cdot \sin(c \cdot Y);$$

where H_0 – field intensity on the surface of poles, [A/m];
 $c = \pi/S_{POL}$ – magnetic field variation factor, [m⁻¹]; S_{POL} – polar

pitch, [m]; X – coordinate directed normally to the line of the vertices of the poles, [m]; Y – coordinate directed normally to the axes of the poles, [m].

Consider the case of an alternating magnetic field of a stationary magnetic system. Assume that the frequency of the field change is ω , while the length of the strand (floccule) on the drum surface is a . Then, under favorable conditions, the speed of the strand moving on the drum surface will be:

$$U_S = 2 \cdot \pi \cdot \omega \cdot a.$$

The gradient of the intensity along the axis Y from the expression for H_Y is:

$$\frac{dH_Y}{dY} = H_0 \cdot c \cdot \exp(-c \cdot X) \cdot \cos(c \cdot Y).$$

Magnetic force F_{M1} pulling the floccule to the pole is:

$$F_{M1} = \mu_0 \cdot \chi_F \cdot H_Y \cdot \frac{dH_Y}{dY},$$

where χ_F – specific magnetic susceptibility of floccule, [kg^{-1}].

Floccule is forced against a drum surface by a magnetic force:

$$F_{M2} = \mu_0 \cdot \chi_F \cdot H_X \cdot \frac{dH_X}{dX}.$$

The equilibrium of the floccule on the drum surface will rise under the condition:

$$F_{M1} = \frac{F_{M2}}{k_F},$$

where k_F – coefficient of friction.

In case of a multipolar system, the floccule, located between the poles, experiences the action of two oppositely directed horizontal forces:

$$F_{MY}^1 = \mu_0 \cdot \chi_F \cdot H_0^2 \cdot c \cdot \sin((S_{POL} - a) \cdot c);$$

$$F_{MY}^{11} = \mu_0 \cdot \chi_F \cdot H_0^2 \cdot c \cdot \sin((S_{POL} + a) \cdot c).$$

Vertical component is:

$$F_{MX} = \mu_0 \cdot \chi_F \cdot H_0^2 \cdot c \cdot \cos(a \cdot c).$$

The equilibrium condition for floccule is:

$$F_{MX} k_F = F_{MY}^1 - F_{MY}^{11}.$$

Solving the last equation for a , we obtain:

$$\sin \frac{\pi \cdot a}{S_{POL}} = \frac{k_F}{2} \cdot \cos \frac{\pi \cdot a}{S_{POL}},$$

whence:

$$a = \frac{S_{POL}}{\pi} \cdot \arccos k_F^{-0.5}.$$

According to experimental data [5], along with frequency increasing, strands begin to decrease; at a frequency of 400 Hz they become indistinguishable for the naked eye.

The analysis of the data in Fig. 1 makes it possible to draw following conclusions. Since the magnetic susceptibility decreases with reduction in size, the strands become shorter, therefore their speed declines. However, as the frequency increases, their velocities no longer depend on the size, since the resistance to movement of the floccule does not depend on an individual particle, but rather has an average value, which depends on the size and speed of the floccule. Determine at what frequency floccules and particles are broken away from the drum surface when the sign of the field strength changes.

Consider the effect of an alternating magnetic field on magnetic particles. The latter usually have an elongated shape, close to the ellipsoid of rotation with the ratio of the major axis to the small being $\Lambda = 2$. The particle is oriented by a long axis (axis of easy magnetization) along the external magnetic field vector. In the case of an alternating magnetic field, the particle follows the change in this vector. The magnetic force that orients the particle is [10]:

$$F_{M1} = \frac{1}{\delta \cdot d} \cdot \mu_0 \cdot H^2 \cdot (k_a - k_b) \cdot \sin \alpha,$$

where k_a, k_b – volumetric magnetic susceptibilities of the particle along the long and short axes, [m^{-3}]; α – angle between vector H and axis a of the particle.

The values of k_a, k_b depend on the magnetic susceptibility of the substance k_M and on the relation $a/b = \Lambda$, that is, from the demagnetizing factor N along each of the axes [9].

The force of the viscosity prevents the rotation of the particle. As the particle rotates, then $U = 2 \cdot \pi \cdot \omega \cdot a$, where ω is the frequency of external magnetic field change.

Equation of a particle motion is $F_{M1} = F_\mu$, since forces are specific. Taking into account that $\omega = \frac{d\alpha}{dt}$, we have:

$$\frac{36 \cdot \pi \cdot \mu \cdot d\alpha}{\delta \cdot d \cdot dt} = \frac{\mu_0 \cdot H^2 \cdot (k_a - k_b)}{\delta \cdot d} \cdot \sin \alpha.$$

Denoting $B = \mu_0 \cdot H^2 \cdot (k_a - k_b)$ and $36 \cdot \pi \cdot \mu = A$, we obtain the equation:

$$\frac{d\alpha}{\sin \alpha} = \frac{B}{A} dt.$$

The solution of this equation has the form:

$$\ln(\csc \alpha - ctg \alpha) = -\frac{B}{A} \cdot t + \ln C.$$

Select the initial conditions from such considerations. We assume that the field is changing instantly from $-H1$ to $+H1$. It is natural that a particle cannot follow the field immediately, hence $\alpha_0 = \pi/2$. Further, the field remains at the level $+H1$ and with $t \rightarrow \infty$ $\alpha \rightarrow 0$. So, $t = 0$, then:

$$\ln\left(\csc \frac{\pi}{2} - ctg \frac{\pi}{2}\right) = \ln C,$$

that is: $\csc \frac{\pi}{2} = 1$, $ctg \frac{\pi}{2} = 0$, then $\ln C = 0$, $C = 1$.

In this case, the final solution of the equation of motion takes the form:

$$\csc \alpha - ctg \alpha = \exp\left(-\frac{B}{A} \cdot t\right).$$

Simplifying the last expression, we obtain the function of the angle of lag from the frequency:

$$\alpha = 2 \cdot \arctg \exp\left(-\frac{B}{2 \cdot \omega \cdot A}\right).$$

The magnitude of this angle varies depending on the variation frequency of the magnetic poles under the concentrated mass on the drum surface. At a frequency of 500 Hz, the angle of lag is already considerable and the particles make oscillatory motions, not having time to make a rotation. At a frequency of 1 kHz, the angle of lag is greater than 45°. Demagnetization will be observed in the case when the angle of lag is more than 90°. In this case, the opposite direction of the particle magnetization and the vector of the external magnetic field is observed. When the angle of lag reaches 180°, the conditions for demagnetizing the particles will be ideal.

Demagnetization of ferromagnetic particles in the flow of suspension. Demagnetization of the ferromagnetic sample is carried out as follows. The sample is fixed rigidly in space where a magnetic field will be induced. The orientation of the sample is such that the easy-magnetization axis is parallel to the external magnetic field vector.

The field strength corresponding to the saturation of the sample is established.

Multiple changes of the magnetic field intensity vector direction are made. As a result, the sample will have a residual magnetization, which is located on the curve of initial magnetization.

The amplitude of magnetic field intensity is decreased, and the procedure is repeated. This is done until the magnetic field intensity is reduced to zero. As a result, the sample will be completely demagnetized. The main condition for demagnetization is its rigid position in a variable magnetic field. When the sample is not fixed, it tends to change its position in accordance with the change in the direction of the external magnetic field vector, being oriented by the axis of easy magnetization along the vector of the external magnetic field. The speed of the sample following the rate of change of the external magnetic field depends on the magnetization intensity of the sample, on the environmental parameters, and on the conditions for

sample fixing in space. The magnetization of a non-fixed sample does not change.

In order to achieve reversal magnetization of particles in the suspension, where, as it is known, ferromagnetic particles can orientate freely along the vector of the external magnetic field, it must be altered with a rate that exceeds the rate of mechanical movement of the sample in this field. The angle of lag between the angle of particle rotation and the magnetic field vector should be more than 90° , for a particle to have time to follow the magnetic field, which at that time has a unipolar direction. Determine what the frequency of the field change should be in this case.

Suppose that the suspension stream contains particles of ferrosilicon as the solid phase, which have the form of ellipsoids of rotation with a major axis a and a minor axis b . Assume also that the magnetic field is uniform.

Upon entering a magnetic field, each particle of such type is magnetized, and "magnetic masses" are formed at its ends. These masses, interacting with an external magnetic field, cause the rotation of a particle with respect to a minor axis, i.e. are oriented by a large axis along the external magnetic field vector. This leads to a change in the demagnetizing factor of the suspension region and, as a consequence, to a change in the magnetic susceptibility of this region. This is so-called magnetic structuring. Prior to entering the magnetic field, the particles are oriented by a long axis arbitrarily in space, and the distribution of the orientation angles is uniform. Moreover, after getting into the field, the orientation of all the particles is the same.

The moment of rotation effecting the particle is [8]:

$$M_{MECH} = f \cdot a \cdot \sin \alpha_1,$$

where $f = m \cdot H$ – force effecting the ends of a particle; α_1 – angle between the major axis of the particle and the external magnetic field vector; m – magnetic mass.

Considering that $m = \Phi$, i.e. the magnetic flux passing through the cross section of the particle S , we obtain $f = \Phi \cdot H$. It is also known that $\Phi = B \cdot H$ and $B = \mu_0 \cdot \bar{\mu} \cdot H$, where $\bar{\mu}$ –

magnetic permeability of a particle; $\mu_0 = 1.26 \times 10^{-6}$ – absolute magnetic constant; B – magnetic induction in the particle region. Then:

$$M_{MECH} = \mu_0 \cdot \bar{\mu} \cdot S \cdot H^2 \cdot a \cdot \sin \alpha_1 = K_1 \cdot \sin \alpha_1.$$

The rotation of particles is prevented by the moment due to the reaction forces, which depend on the viscosity of the medium. The viscosity of the medium is [5]:

$$F_\mu = \frac{18 \cdot \mu \cdot U_p \cdot m_p}{a^2 \cdot \delta_p},$$

where μ – coefficient of dynamic viscosity of the medium, [N·s/m²]; U_p – linear speed of rotation of the ends of the particle, [m/s]; m_p – particle weight, [kg]; δ_p – particle density, [kg/m³]. In such a manner:

$$M_\mu = F_\mu \cdot a.$$

The linear velocity of the ends of the particle is $U = a \cdot \frac{d\alpha_1}{dt}$. The general expression for the static reaction moment is:

$$M_\mu = \frac{18 \cdot \mu \cdot m_p}{a^2 \cdot \delta_p} \cdot \frac{d\alpha_1}{dt} = K \cdot \frac{d\alpha_1}{dt}.$$

The angle of rotation of the particle can be determined from equality:

$$M_\mu = M_{MECH} + J_1 \cdot \frac{d\omega}{dt},$$

where J_1 – moment of inertia of a particle.

With an error not exceeding 5%, assume that $\sin \alpha_1 = \alpha_1$. Then the equation of dynamic equilibrium of the particle takes the form:

$$J_1 \cdot \frac{d^2\alpha_1}{dt^2} - K \cdot \frac{d\alpha_1}{dt} + K_1 \cdot \alpha_1 = 0.$$

The roots of the characteristic equation of this differential equation are:

$$p_{1,2} = \frac{K \pm \sqrt{K^2 - 4 \cdot J_1 \cdot K_1}}{2 \cdot J_1},$$

and the solution has the form:

$$\alpha_1(t) = \alpha_0 \cdot \left(1 - \frac{p_1}{p_1 - p_2} \cdot \exp(-p_1 \cdot t) + \frac{p_2}{p_1 - p_2} \cdot \exp(-p_2 \cdot t) \right).$$

Estimate the numerically obtained solution for such initial data:

$$a = 10^{-3} [m]; \quad \delta_p = 4000 \left[\frac{kg}{m^3} \right]; \quad \bar{\mu} = 2;$$

$$H = 50000 \left[\frac{A}{m} \right]; \quad \mu = 10^{-3} \left[\frac{N \cdot s}{m^2} \right].$$

The moment of inertia will be: $J_1 = 0.52 \times 10^{-12}$. Coefficients are: $K_1 = 5 \times 10^{-6}$, $K = 9.42 \times 10^{-9}$. The roots of the characteristic equation are equal to: $p_1 = 17.67 \times 10^3$, $p_2 = 0.59 \times 10^3$. Equation of rotation of a particle in numerical expression is:

$$\alpha_1(t) = \alpha_0 \left(1 - 1.03 \cdot \exp(-17.67 \cdot 10^3 \cdot t) + 0.03 \exp(-0.59 \cdot 10^3 \cdot t) \right).$$

Time can be expressed in terms of the velocity of the suspension flow in the area of action of the magnetic field. Alternatively, it is possible to set the condition that the angle of rotation of the particles should not be more than 5-10% of the initial value. Hence, we determine the time of action of a magnetic field of one polarity. To do that, confine ourselves to a member with the largest exponent and the largest coefficient, since the second term of the equation introduces a small fraction of the effect on the particle motion. Then:

$$\Delta t = \frac{\ln 0.95}{-17.67 \cdot 10^3} = 3 \times 10^{-6} s.$$

The frequency of the magnetic field will be:

$$f = \frac{1}{\Delta t} = 0.33 \times 10^6 Hz = 330 kHz.$$

The time of a field exposure with a frequency of 50 Hz corresponds to 0.01 s. As a result, the angle of lag of the particle orientation from the magnetic field vector is about 0.0003 of its initial position.

Thus, the effect on a ferromagnetic suspension by an alternating magnetic field with a frequency of 50 Hz cannot cause demagnetization of particles of a ferromagnetic material, since these particles have time to be oriented by the axis of easy magnetization along the external magnetic field vector. Hard fixing of the particle position in an alternating magnetic field is not observed. Some improvement in the rheological parameters of the ferromagnetic suspension, which passed through the demagnetizing device, can be explained by the mechanical destruction of flocs to the dimensions that will be determined by the value of the residual magnetization of the ferromagnetic particles.

The time of action of an alternating magnetic field of one polarity is equal to half a period: $t = (2 \cdot \omega)^{-1}$. On the basis of which obtain the function of the angle of lag from the frequency:

$$\alpha = 4 \cdot \arctg \exp\left(-\frac{B}{A} \cdot \omega\right).$$

Thus, the weighting agent particles at frequencies up to 100 Hz practically succeed in following the vector of the external magnetic field with lag of 1°. Over 1000 Hz, the lag is considerable and amounts to more than 90°. The reversal magnetization of particles begins. At a frequency of 100 kHz, there are sufficient conditions for remagnetization of the particles, since the angle of lag is 177°.

Table 1

The angle of lag of the particle turn to the external alternating magnetic field vector

Frequency, kHz	0.01	0.05	0.1	0.5	1	3	5	10	50	100
Angle, deg.	0	0	1	72	120	160	168	172	176	177

The angle of lag can be used as a measure of the demagnetization of particles: $K_{DEM} = \sin(\alpha - 90)$, since demagnetization starts, when the component of the external magnetic field vector begins to have an opposite direction to the magnetization vector of the particle.

Even if we assume that the magnetic particles are completely demagnetized, then, falling into the Earth's magnetic field, they acquire the magnetization corresponding to this field, and unite into aggregates. Thus, it is not possible to demagnetize the weighting agent particles completely.

Demagnetization experiments. To obtain high-quality magnetite concentrate at preparation plants, multi-stage magnetic separation is used. Since the ferromagnetic minerals of magnetite are characterized by a residual induction of magnetization, the particles of magnetite ore spontaneously form floccules, inside which the waste particles are trapped, which reduces the quality of the concentrate. In addition, magnetic flocculation reduces the efficiency of classification, which increases the circulating load on the mills.

Classification of the crushed product in the second and third stages of grinding is carried out in hydrocyclones. Since the floccules of particles of predefined grain-size class have a larger size, they go to the underflow and return to the mill for grinding, which increases the circulating load and reduces the grinding rate. Therefore, it is advisable to demagnetize the suspension, entering the hydrocyclones.

A suspension of magnetite of *Poltava mining and concentration complex* with a content of grain-size class of less than 50 μm of more than 96% and a solid phase concentration of 100 g/l was used for the studies.

Preliminary magnetization was carried out in a constant magnetic field with an induction of 0.25 T. Demagnetization was performed in pulsed mode in a laboratory device [1]. Demagnetization consists in placing the particle in an external alternating magnetic field, the induction of which decreases smoothly from the maximum value, which should be greater than the residual magnetization of magnetite particles, to zero. Herewith, the demagnetization occurs according to the hysteresis curves. In the coil of the solenoid, damped current oscillations are created, and the number of oscillations must be greater than 5. This is ensured by the corresponding Q-factor of the

oscillatory circuit. In this case, all particles inside the solenoid are subjected to demagnetization in places where the maximum induction of the magnetic field is greater than the residual magnetization of the particle.

In general, we can assume that the residual magnetization of magnetite plays a negative role in the separation of minerals. Thus, its reduction, or demagnetization, is an actual task.

It is energetically and constructively expedient to perform demagnetization in a pulsed mode.

Theoretical and quantitative relationships obtained for the degree of demagnetization of magnetite particles were verified by microscopic examination of the suspension, which was processed by a high-frequency magnetic field. The measurement standard is shown in Fig. 2.

Fig. 2 and 3 demonstrate photographs of particles of the magnetite suspension with magnification $\times 24$, subjected to magnetization (Fig. 2), and having passed through a field of decreasing intensity with a frequency of 20 kHz (Fig. 3). It is qualitatively possible to say for sure that the particle size decreases significantly. Both suspensions underwent intensive mechanical mixing, and the particle size in them was not changed. Thus, the mechanical effect does not have a significant impact on the size of the floccules.

Fig. 4 shows a photograph of particles of the suspension with magnification $\times 56$, which were magnetized by a constant magnetic field with an induction of 0.25 T. The division value of the measuring rule corresponds to 10 μm . As it can be seen, the length of the floccule reaches 300 μm (0.3 mm).

Having subjected the same suspension to the action of an alternating magnetic field with a frequency of 20 kHz and a decreasing intensity, we obtained floccules of smaller size (Fig. 5) – about 100 μm (0.1 mm), while the structure of these aggregates, consisting of chains of individual particles, is clearly observed. It can be concluded that the magnetite particles have been partially degaussed, but complete demagnetization was not achieved.

Having subjected the same suspension to the action of an alternating magnetic field with a frequency of 70 kHz (Fig. 6), completely demagnetized particles, separated from one another, were

obtained. It should be noted that the samples were screened from the Earth's magnetic field.

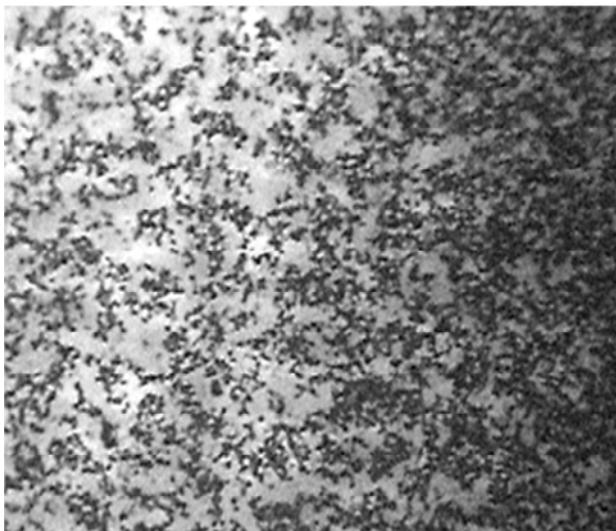


Fig. 2. Suspension of magnetized magnetite particles



Fig. 3. Suspension of demagnetized magnetite particles

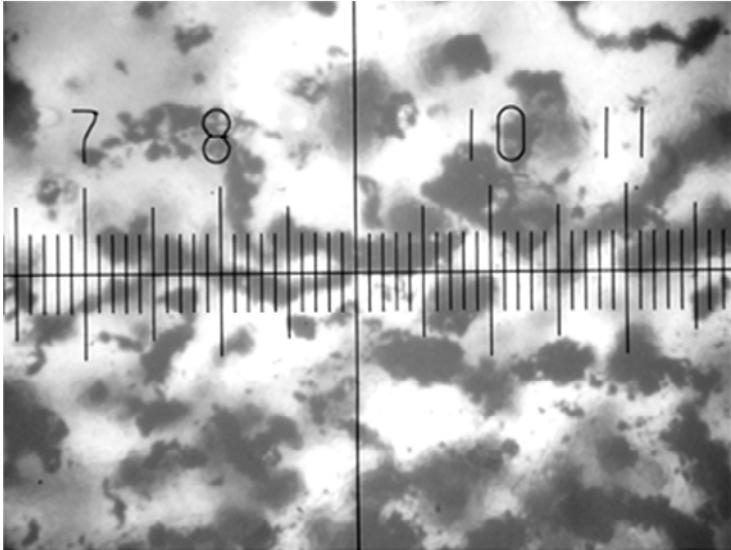


Fig. 4. Magnetized suspension

The study of the particle sizes in Fig. 6 allows concluding that practically all particles have a size of less than 20 μm . When classifying such particles, for example in a hydrocyclone, they will all go into an overflow product, in contrast to the magnetized particles, a large part of which in the form of floccules will fall into the underflow and will return to the mill. Consequently, the use of complete demagnetization of magnetite particles prior to their classification will reduce the value of the circulating load on the mill, which will increase the efficiency of its operation. It is possible to separate particles of this size in sensitive devices. These may be hydrocyclones with a diameter of cylindrical part of 50 mm or less, glazed in the area of a working part.

In addition, magnetite minerals in particles of such a size are almost completely released, which theoretically makes it possible to obtain ultrahigh-quality concentrates.

When the demagnetized particles enter the magnetic field of the existing magnetic separators, they are magnetized and form floccules, in which rock particles are trapped.

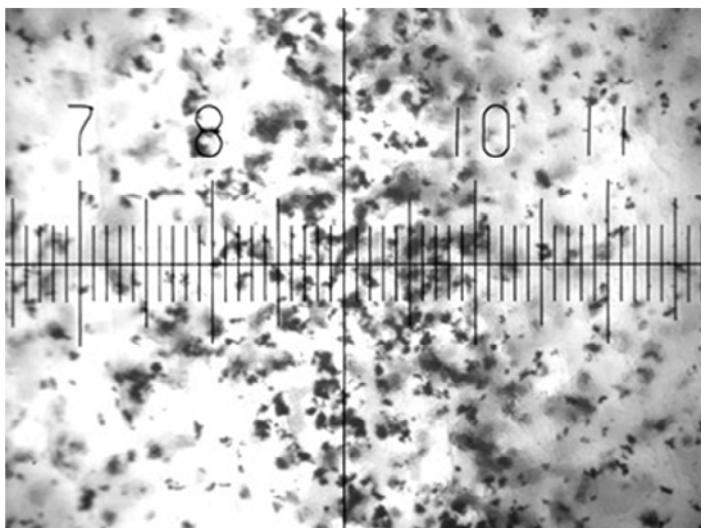


Fig. 5. Particles of suspension, demagnetized in the field of 20 kHz

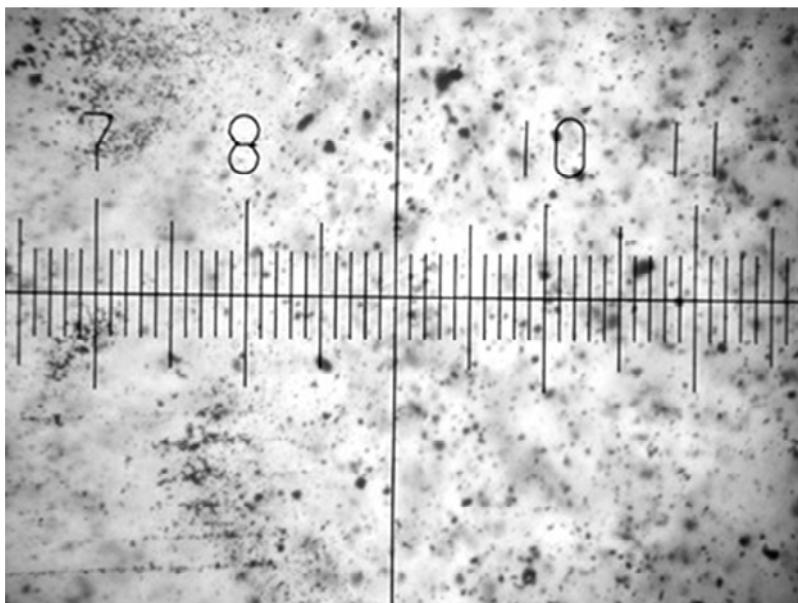


Fig. 6. Particles of suspension after treatment with a 70 kHz field

Accordingly, the quality of concentrate deteriorates. To prevent quality deterioration, it is necessary to separate demagnetized particles either by non-magnetic methods, e.g. by flotation, or in magnetic separators with an alternating high-frequency magnetic field.

Conclusions

- Until now, there are no devices for complete demagnetization of fine particles of the ferromagnetic material in the suspension flow.
- Demagnetization of such type can be carried out in high-frequency magnetic fields with induction amplitude greater than the residual magnetization of magnetite particles.
- Demagnetization of ferromagnetic particles before their further non-magnetic separation can provide indices of the concentrate quality, close to the theoretically possible ones.

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DEVELOPMENT OF METHODS FOR PREPARING FLY ASH FOR SEPARATION BY ACTIVATION

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Abstract

Purpose. Isolation of the aluminosilicate fraction from fly ash, study of physical and mechanical properties of binders obtained from TPP wastes.

Methods. Ash, carbon concentrate (underburn), ash concentrate and products of their treatment with reagents were tested by optical methods. The morphology of the particles of the subjects of inquiry was studied with the scanning electron microscope REM-100. The composition of the ash phases was investigated using the X-ray diffractometer DRON-2.

Findings. The technology of sorbents based on coal combustion products through a variety of methods was researched. It is shown that these sorbents are distinctive because their structure has non-localized π -electrons of the graphite-like networks of crystallites of carbon. This circumstance determines not only the uniqueness of electro-physical properties of coal but also adsorption, redox, chemisorption processes on the border of coal-slurry. The listed circumstances allow you to use the original methods of chemical and mechanochemical modification of the surface chemical and coal, due to introducing desired donor and acceptor atoms in carbon frame, which increases the absorption capacity and selectivity carbon sorbents.

Practical implications. The article presents the results of receipt of binders based on TPP ash. It has been shown that a component of the fly ash is aluminosilicate spheres that can be used in the production of lightweight concrete. It is proved that the result of mechanochemical activation mixture consisting of alumino-silicates, resulting lightweight concrete has high strength 7-8 MPa, which allows, while maintaining the technical characteristics save from 20 to 30 % of binder. Concrete obtained based on aluminosilicate spheres separated from fly ash may be used to prepare the

outer wall construction, small building blocks, as well as monolithic housing. In comparison with known compositions keramsit compositions comprising TPP waste.

Keywords: fly ash, aluminosilicate spheres, mechanical activation, vibro-impact mill

1. INTRODUCTION

Solid fuel, which is widely used in the production of electricity, produces a significant amount of ash and slag in the process of combustion.

Given the growing use of solid fuels, in particular low calorific fuel with high ash content, the effective use of TPP ash is in focus. The amount of ash and slag produced by TPPs is 500 thousand tons per 1 million kW of their power. The ash and slag are transported to special waste dumps. An average of 0.3% of the total amount of electricity produced by the TPP is used for the disposal of thermal waste from power plants to the dumps, that is, considerable material resources are spent for that. In addition, waste storage requires the alienation of land that could be otherwise used in agriculture. For example, today's TPP needs 1000 to 2000 hectares of area for its ash-dumps.

The level of ash and slag reuse is about 15% (for comparison: in the USA - 20%, in France - 62%, in Germany - 80%), and they are used as a rule in the production of building materials: as admixture to cement, as components of building concrete and mortar, in road construction, in the manufacture of bricks, and as raw materials for the production of Al_2O_3 , Fe_3O_4 , TiO_2 , K_2O , Na_2O , P_2O_5 , U_3O_8 , etc..

Some publications propose methods for the integrated use of TPP ash with the manufacture of conditioned products to be used in various industries as substitutes for natural raw materials.

Studies on the immobilization of heavy metal waste in the mixtures of portland cement and ground fly ash are now of importance. For the research, aqueous solutions containing zinc, cadmium and mercury were used. It was found that the degree of metal immobilization significantly increases in the presence of ash.

So attempts are being made to systematize the TPP ashes according to the specified indicators in order to predict the most rational use of both ash and its individual components obtained by

ash separation in various ways (mechanical, physic-chemical, chemical separation, etc.).

Ash from thermal power plants (TPPs) is produced as a result of coal combustion and is, on the one hand, a source of environmental pollution; but on the other hand, it is a resource of raw materials needed for industry.

In Ukraine, thousands of tons of ash containing a number of useful components have accumulated in the TPP dump. Moreover, more than 10 million tons of fly ash is added to the TPP dumps annually. Despite the relatively high content of phase constituents with high consumer properties in a number of ashes, it is only possible to attribute them to technogenic minerals by convention, since no economically viable methods of extracting the useful components are now available. Therefore, ash from thermal power plants is used to a limited extent.

In Ukraine, up to 75% of all electricity is supplied by thermal power plants that mainly use coal as a fuel; this leads to producing huge quantities of ash and slag waste. Every 10 years (according to statistical data), the amount of ash and slag produced in thermal power plants doubles.

Burned rock and coal waste are useful and widely used mineral raw materials for the production of building materials.

Ash is dust-like residuals formed by burning solid fuels, with particle size of 5 to 100 μm . Ash is classified according to the type of fuel being burned, the way it is prepared and burned, place where ash forms in the boiler, and methods of its disposal.

Fuel combustion is accompanied by the firing of barren rock, which results in the dehydration of clay substances and formation of low-base aluminates and calcium silicates. Depending on the amount of liquid phase in the ash, the content of glass, which usually has a gelenite-melilite composition, may vary widely.

Ash consists of organic and inorganic phases. The inorganic phase, in turn, includes aggregated and non-aggregated glass particles, as well as crystalline matter.

Coal ash may include, by volume, the following components: 1 – 22% unburned fuel particles, i.e., the organic phase; 25 – 85% aggregated vitreous particles; 8 – 57% non-aggregated vitreous particles, and 0.2 – 20% crystalline components [1].

Ash density and volume weight vary depending on the content of unburned fuel particles in it and its granulometric composition. The volume weight of ash is also affected by the fuel combustion temperature, ash humidity and degree of compaction whose values range from 1.75 to 2.4 kg / m³, and the volume weight in loose state, which may be 600 to 1300 kg / m³. The specific surface area of ash from different TPPs varies and ranges from 1100 to 3000 cm³ / g and more.

Fuel slags are vitreous grains of irregular shape up to 40 mm in size, and they are the main type of waste in lump coal combustion. In the case of pulverized coal combustion, the slag makes up 10–25% of the mass of the ash formed. Slags form either as a result of sintering of individual particles on the fire grate at a temperature above 1000° C, or as a result of cooling the mineral matter of coal, having been molten at a temperature of above 1300 °C.

2. PURPOSE

Isolation of the aluminosilicate fraction from fly ash, study of physical and mechanical properties of binders obtained from TPP wastes.

3. FORMULATING THE PROBLEM

To obtain a raw material for manufacturing heat-insulating materials, fly ash was subjected to the following operations: preliminary flotation performed to separate the inorganic and organic parts, followed by activation of the inorganic part by two methods, i.e. chemical and mechanochemical activation. The consumer properties of the raw materials were assessed by studying their phase composition, physico-chemical and physico-mechanical properties, and their compliance with the technical specifications for the further processing of the products obtained.

Ash, carbon concentrate (underburn), ash concentrate and products of their treatment with reagents were tested by optical methods. The morphology of the particles of the subjects of inquiry was studied with the scanning electron microscope REM–100. The composition of the ash phases was investigated using the X-ray diffractometer DRON–2.

4. RESULTS OF ANALYSIS

Coal ash is formed from all mineral impurities in the coal bands, rock layers of the roof and soil, which are entrapped in the commercial coal during its mining. The major impurities are clayed minerals and quartz, while carbonates, sulphides, iron oxides and other minerals are present in minor amounts. The phase composition of the ash is mainly determined by clayed minerals, which are transformed into other aluminosilicates (glass phases, possibly mullite, cristobalite) during high-temperature processing in boilers. It should be taken into account that this is a large group of minerals with compositions varying over a wide range [2].

In natural conditions, the group of mixed-layer minerals is the most common. A combination of layers of hydromica, montmorillonites, chlorites and vermiculites is of frequent occurrence. A common feature of all clayed minerals is their aluminosilicate composition.

The challenge is to create a technically rational and cost-effective process for separating the constituent components of TPP ash.

Ash is a complex polymineral system, characterized by varying composition. Depending on its reactivity and mineral composition, the content of components in the waste varies widely. Thus, for example, the carbon content of the waste can vary from 12 to 45% over a period of several hours.

A dispersed analysis of the TPP fly ash suggests a conclusion that it is advisable to separate particles of a narrow class of fineness, within the range of 40 to 150 μm , with an ash content of about 33%.

The product (larger than 40 μm with a yield of 25% of the total ash mass) can be used as a heat-insulating coating in steel casting (instead of ash-graphite mixtures) to produce ingots of killed steel, for example wheel steel.

In addition, the TPP ash may be used to obtain an ash product with an ash content of more than 90% (fineness less than 40 μm with a yield of 75% of the total ash mass) for the production of ceramic and other products.

The former product may be enriched by flotation. As a result, the coal concentrate may contain less than 20% ash with a sulfur content of about 0.4%. The sulfur content exceeds the European

requirements. Such a coal product may be used as a reducing agent in metallurgical processes, agglomeration, etc. The calorific value of the concentrate is about 6,000 kcal / kg (25,000 kJ / kg).

TPP ash with high carbon underburn should be subject to flotation to obtain a carbonaceous fraction (concentrate) and mineral fraction (waste).

The unburned particles of coal can be separated by direct flotation using apolar and heteropolar surfactants at a low pH of the pulp (pH 6 – 8 and pH 3– 6,5 at the first and the second stage, respectively). Several recleanings are needed to obtain products of the specified quality. A method has been developed for ash flotation with the use of MK reagent as a collector in the presence of a depressor. The method ensures obtaining conditioned products without recleaning, with 95-98% carbon yield to the concentrate.

A preliminary hydroclassification of ash into 40 μm class enhances the efficiency of its separation.

The effect of the raw material grinding on the process of ash flotation was studied. Due to the increased contrast range of the phases being separated, a concentrate with a high content of the combustible fraction was obtained with a slight change in the ash content of the waste. The inverse flotation of the TPP ash allows obtaining products with physico-chemical properties that are different from the properties of the products obtained by direct ash flotation. Hence, the carbon-bearing fraction of ash features a high specific surface area, which allows it to be used as an effective sorbent, in contrast to a direct flotation concentrate, which exhibits a slightly less effective specific surface area of the particles due to collector adsorption on their surface. The reverse flotation waste manifests clearly expressed hydrophobic properties due to the collector adsorption on the surface of the particles, which allows using it as a hydrophobic filler of polymer composite materials with no additional surface modification. The essence of this method is the use of a depressor of coal particles (carbon-alkaline reagent) and a MSTM (Soap of crude tallow oil) collector with adjusting the pH of the medium. The method allows the extraction of the ash mineral fraction into the froth, while the flotation tail will be a carbon-containing fraction of ash.

The most important phase components of ash are iron oxides (20%). Iron compounds are part of the spheroidized particles of the glass phase. The magnetic component is black melted balls consisting of magnetite and hematite. The use of dry and wet magnetic separation allows the separation of high-quality magnetite from fly ash. Magnetite, obtained in this way, can be used as a weighting material in heavy medium coal preparation. The nonmagnetic product is used for the manufacture of cement, as a filler for mining workings, etc. Also, dry ash can be separated by electrostatic separation in electrostatic corona drum separators with a sufficiently high carbon extraction into the concentrate.

A complex use of ash components involves extracting rare-earth elements, refractory and non-ferrous metals. The average content of metals in ash is much higher than in sedimentary rocks. Even with minimal content of rare-earth elements in ash, their extraction can be beneficial. Ash and slag waste can be an important raw material for alumina production. For this purpose, several flow processes have been developed, which can be divided into two groups, namely: acid and alkaline methods.

The chemical composition of the mineral matter of the ash enrichment products practically coincides with the chemical composition of the waste, with the exception of the carbon content.

An ash processing technique and a plant operating in conjugation with TPP will allow obtaining products of specified quality irrespective of the conditions of coal combustion at the TPP. This, on the one hand, excludes building new ash dumps at the TPP, and on the other hand, allows the production of sorbents of oil products and heavy metals, heat-insulating coatings for steel casting, low-sulfur reductants, fuels, and raw materials for building materials.

Carbonaceous concentrate is also an effective sorbent of apolar and surface-active substances (surfactants), including petroleum products.

The resulting ash cake (mineral matter of ash) may have an ash content of 90-95%. Decarburized aluminosilicates can find a variety of applications in the production of building materials, composites, in metallurgy and other industries.

The studies confirmed the effectiveness of the use of enriched ash in the preparation of clay-ash claydite and brick, high-quality ash

gravel, contact-condensation mixtures, as well as in the construction of road bases and pavements.

Electron microscopic studies and X-ray diffraction analysis of the morphology of the ash phases and products of its enrichment revealed the features of their surface microstructures. Figure 1 shows the general view of TPP ash particles.

Figure 1 shows that ash consists mainly of two components: alumina-silicate particles of spherical shape and unburned coal particles (carbon underburn).

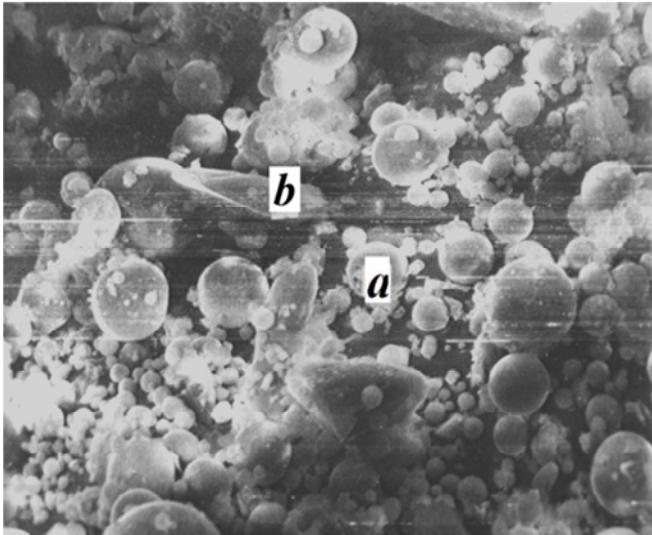


Fig. 1. General view of TPP ash particles (initial), magnification 517x:
a - spheres; b - coal particles

Silicate particles of irregular shape are few, these are mainly fragments of spherical particles; particles of slag and other minerals occur occasionally. Aluminosilicate particles have different degrees of hollowness: from balls with a thin shell (microspheres), to practically dense particles (with relatively small inclusions of the rock inside). In all cases, their surface is smooth, sometimes characterized by the presence of small protrusions-nipples (Figure 2).

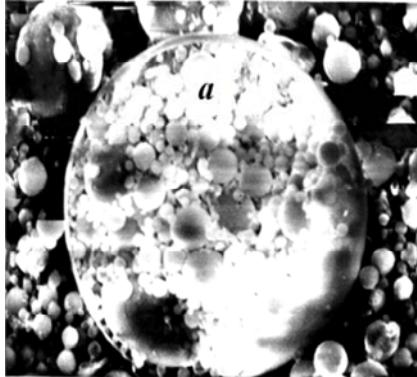


Fig. 2. General view of the TPP ash concentrate, magnification 929x:
a – spheres; b – coal particles

Through-pores occur on the surface, but as a rule, porosity in the form of small bubbles is only observed within the sphere shells. The size of spheres varies in wide limits: from micron fractions, to hundredths of millimeters. Large spheres are single and are formed by smaller microspheres. Spheres with a size of 5 to 20 μm predominate.

Chemical activation was carried out by treating the ash with NaOH solution.

Figure 3 shows a particle of coal concentrate obtained from TPP ash and treated with a 20% solution of NaOH.

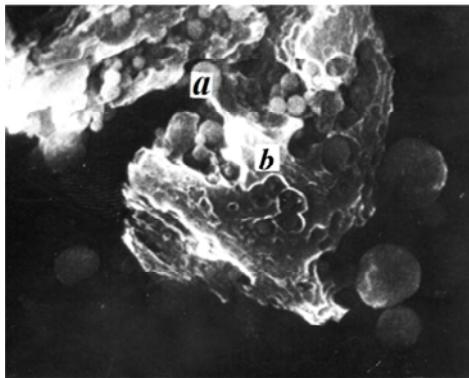


Fig. 3. General view of TPP ash particles (after treatment with 20% NaOH),
magnification 1305X: a – spheres; b – coal particles

Treating the samples with alkali results in a partial separation of the coal particles and shallowly embedded spheres from the concentrate surface, so that its surface acquires a cellular microstructure. Colonies of spheres located more deeply (in microcracks and microcavities) are not separated by this chemical activation, their separation is possible through mechanical action. After the treatment of the TPP ash concentrate with a 40% solution of NaOH, the sample material predominantly consists of disintegrated spheres (Figure 4).

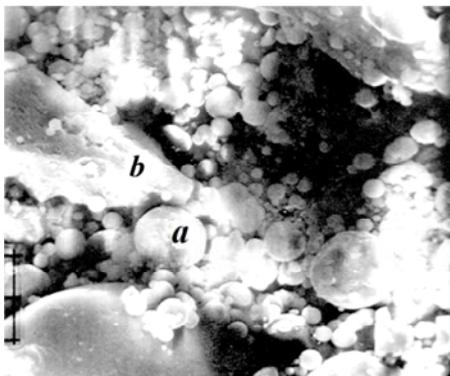


Fig. 4. Ash concentrate after treatment with 40% NaOH, magnification 842x:
a – spheres; b – coal particles

Figure 5 shows the general view of particles of the TPP carbon concentrate.

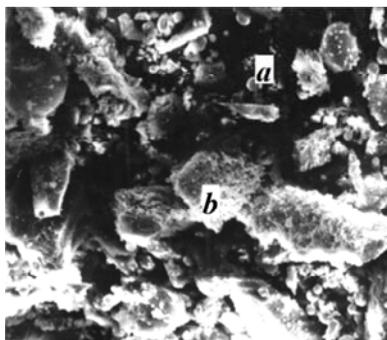


Fig. 5. General view of the carbon concentrate of TPP ashes, magnification 20x:
a – spheres; b – coal particles

The spheres are mainly vitreous, colorless of two types: transparent or translucent "white".

Some spheres are located in the depressions of coal particles formed during its uneven burning (gasification), therefore the carbon concentrate consists of a mechanical mixture and technogenic clusters of aluminosilicate spheres and coal particles.

There occur dark opaque spheres, which along with the silicate part contain dendritic magnetite. In addition, the X-ray phase analysis of samples of ash carbon concentrates revealed peaks of quartz, magnetite, hematite and the presence of pyrrhotite and maghemite.

The chemical composition of fly ash from DTEK Pridneprovskaya TES is presented in Table 1.

Table 1

Chemical composition of fly ash

Product	Content, % on dry substance										
	SiO ₂	Fe ₂ O ₃	CaO	MgO	SO ₃	Al ₂ O ₃	TiO ₂	P ₂ O ₅	K ₂ O	Na ₂ O	C
Fly-ash	49.7	12.3	2.1	1.3	0.4	12.8	0.5	0.9	3.1	0.6	12.0

Table 1 shows that, after the separation of aluminosilicate spheres (AS) from fly ash, they can be used to produce various grades of concrete [3]. The most common binder used in construction is portland cement. Depending on the purpose, portland cement of M 300, 400, 500 grades is used.

As an alternative raw material, the study used AS that feature latent (hidden) properties; to activate them, it is necessary to carry out special technological operations.

At the first stage, the concrete composition was selected by selecting a rational "cement-to-AS" ratio of their mixture.

The concrete mixtures and cement paste were produced in compliance with DSTU B V.2.7-92-99.

All the compositions feature an increased water consumption, which reduces the strength of the material. This can be explained by the high water demand of the AS, although the water absorption of the concrete remains within the permissible limits.

An analysis of the experimental data leads to a conclusion that compositions 4 – 5 are optimal. These are the compositions with a moderate cement consumption (190–210 kg) and they exhibit the necessary properties (compressive strength, thermal conductivity).

Table 2 shows the compositions of lightweight concrete and their basic physical and mechanical characteristics.

Table 2

Physical and mechanical properties of fly ash-based binders

Experiment No.	Consumption of materials per 1 m ³			Main characteristics and properties					
	Cement, kg	Aluminosilicate spheres (AS), kg	Water, l	Density, kg / m ³	Bending strength, MPa	Compressive strength, MPa	Conductivity coefficient, W / m · °C	Water absorption by volume, %	Frost resistance, cycle
1	300	350	360	750	1.9	7.5	0.16	5	25
2	250	360	360	696	1.7	6.5	0.15	6	25
3	210	380	370	650	1.3	4.8	0.14	6	25
4	190	380	365	620	9.0	4.0	0.14	6	25
5	160	370	330	600	7.0	3.3	0.14	6	25
6	120	390	340	540	6.0	2.6	0.12	7	25
7	100	396	340	530	4.5	1.3	0.12	7	25
8	80	400	360	510	3.5	0.9	0.12	8	25

A feature of the AS use is the creation of a porous (closed) structure throughout the volume of the material. The cement paste envelops the AS in the course of stirring and creates a strong frame after hardening [4]. The experimental results can be explained by the insufficient volume of the cement paste for enveloping the AS and the creation of a strong framework.

It is interesting to note that a decrease in the consumption of cement leads to a significant deterioration of its strength characteristics.

It is known that plasticizers are used to increase the strength of concrete. In this study, the problem was approached by pre-activation of a mixture of cement and AS isolated from fly ash, in a vertical vibratory mill. This approach allows to fully reveal the

hidden properties of some substances and use the potentialities of other substances (binders).

Mechanochemical activation was carried out in a vertical vibratory mill, designed and manufactured at the National Mining University. [5].

Table 3 shows the results of calculating the kinetic curves and internal energy U consumed in grinding the mixture in a laboratory vertical vibratory mill.

Table 3

Energy characteristics of aluminosilicate grinding process

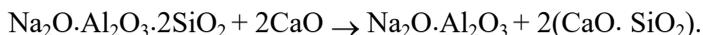
Grinding time, t, s	Internal energy, U, kJ	Specific surface area, S, m ² / kg	Dispersion constant, k, s ⁻¹
0	2500.00	165	0.6
1	4210.08	178.7	1.7·10 ⁻²
10	5363.14	13431	1.8·10 ⁻²
20	6044.00	2665.7	1.9·10 ⁻²
30	6031.84	3204.8	2.2·10 ⁻²
40	6009.73	3498.4	2.5·10 ⁻²
50	5687.38	3797.6	2.9·10 ⁻²
60	5658.30	3928.7	3.1·10 ⁻²
120	6056.67	3812.9	3.0·10 ⁻²
180	6092.47	3851.6	3.1·10 ⁻²
240	6082.09	3933.6	3.2·10 ⁻²
300	5992.00	4358.1	3.0·10 ⁻²

The data in Table 3 demonstrate that grinding predominantly lasts for 40 – 50 minutes, an energetically stable process associated with the formation of activated complexes starts in the 20th minute; the activated complexes should not be confused with aggregates that do not feature the improved energy characteristics.

Discussion. A mixture of cement and AS is not individual silicates, but a solid solution. In activating the isolated aluminosilicates in a vertical vibratory mill, the surface is partially blocked by phase contacts and by the formation of micropores inaccessible to water molecules [6]. In this case, the formation of the reinforcing core of the crystallization spatial structure is basically completed. Further grinding results in hardening the condensation-crystallization structure due to the intergrowth of new hydrated

formations and chemical condensation of the surfaces of Si-OH groups with the formation of Si-O-Si bonds; new coagulation contacts appear, which later turn into phase contacts.

Thus, the ability of the components to be set and hardened when mixed with water will depend on the only crystalline component β -2CaO·SiO₂ formed by the reaction:

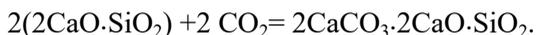


It further interacts with SiO₂ (from AS) by the reaction:



In order to confirm the changes that occur as a result of mixing the concrete components with water, spectroscopic studies were carried out after the vibro-impact processing. The infrared spectra of the activated samples show the main band shift 716, 878 cm⁻¹, which intensifies C₃S. The maximum of the broad vibration band $\nu_{\text{as}}\text{SiO}_4$ shifts toward larger wave numbers. The spectra of AS being hydrated indicate absorption, which is not characteristic of orthosilicates, in addition to the shift of the stretching band Si=O – vibrations to 1050 cm⁻¹. Also, bands of deformation vibrations were detected in the low-frequency region, insignificant absorption was observed in the interval of 600-800 cm⁻¹, which indicates the condensation of tetrahedra and formation of layered hydrosilicates.

Absorption in the region 1430 – 1480 cm⁻¹ is associated with the carbonization of samples, which can occur due to prolonged solidification in the open air. Thus, as a result of the dicalcium silicate interaction with atmospheric carbon dioxide, calcium carbonate crystallizes together with pseudo-wollastonite by the reaction:



Activation of the mixture containing aluminosilicates leads to the formation of particles of irregular shapes. When ground, an AS splits along planes located at an angle of 90°; as a result, its powder consists of block-like particles and fragments. This mineral is characterized by a relatively high Mohs hardness. As the particle sizes are reduced to 5 μm or less, their abrasiveness is reduced due to the reduction of the particle edge sharpness; consequently, the mixture is characterized by a smaller specific surface area due to

enlarged particle sizes and low content of fine fractions. A feature of the aluminosilicate activation in vertical vibratory mills is the production of homogeneous powders with an increased content of fine fractions [4].

The ground samples were also analyzed by X-ray fluorescence spectroscopy to determine the structure changes by a chemical analysis. Fig. 6 shows the obtained X-ray phase spectra.

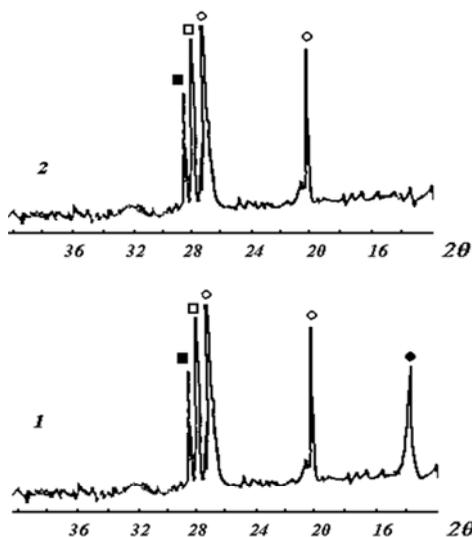


Fig. 6. Diffractograms of vibro-activated (1) and initial (2) ash

It was found that when the mixture is being ground, the structure of the AS changes, since breaking the crystal lattice as a rule destroys the weakest bonds-bridges between sodium, potassium and calcium oxides. Free ions formed on the surface of the particles interact with the environment; thus, in a moist medium they interact with hydrogen ions of water (chemisorption) to form free hydroxyls, which produces an alkaline medium that increases with decreasing the particle sizes.

Products of TPP ash separation may be used in various industries: metallurgy, power engineering, chemical industry, building material production.

In metallurgy, attempts are being made to replace the beneficiated char with the high-temperature char and anthracite in the process of iron ore concentrate agglomeration. Obviously, a high-carbon flotation concentrate can be used to partially replace the beneficiated char.

Table 4 presents the test results for the activated AS samples; the results demonstrate that the mechanochemical treatment enables obtaining high-strength concrete with a strength of 7-8 MPa.

Table 4

Physical and mechanical properties of fly ash-based binders after mechanochemical activation

Sample	Specific surface, m ² / g	Density of initial mixture, kg / m ³	Compressive strength, MPa	Water absorption, %
1	0.18	1700	6.2	2.5
2	0.18	1600	6.8	2.4
3	0.18	1700	7.1	2.2
4	0.18	1700	7.5	2.6

Thus, preliminary mechanoactivation of fly ash-based binders allows to improve the water absorption and enhance their strengths.

In the production of metallized pellets, the fuel additive (beneficiated char) makes up 14% (grain size of <100 microns - 70%, ash content - 18.35%, sulfur content - 0.65%). Thus, without lowering the quality of metallized pellets, the high-temperature char may be replaced by ash flotation concentrate.

As a component of pulverized fuel oil compositions for blast furnace processes, it is possible to use low-ash flotation concentrate. In addition, a method was developed for metal siphoning into a mold, which uses a concentrate with carbon content of 35 - 60% as a heat-insulating coating. It is important that the use of this material allows abandoning graphite for the same purposes.

In the energy sector, the concentrate froth can be used as a fuel admixture to boiler fuel. The use of concentrate can significantly reduce unburned carbon losses when burning the AS grade coal; in addition, the operating parameters of the boiler will not deteriorate under the condition of continuous dispensing of the concentrate.

The method for sorption of heavy metal cations has been developed, which allows capturing them from industrial effluents. In this regard, a concentrate froth of the cleaner flotation is the most effective due to its high carbon content. The cleaner flotation waste with ash content of 40 - 50% may be used as a sorbent of organic acids and various petroleum products. An ash-based sorbent was obtained through changes in the boiler temperatures and coarsening the ground raw materials; it is sufficiently effective for the purification of surfactant-containing sewage.

Waste of TPP ash flotation is more widely used in the production of building materials than the initial ash. This can be explained by the adjustable carbon content and granulometric compositions of the ash mineral fraction.

Flotation waste is beneficially used in the manufacture of various concretes and concrete mixtures in order to save cement without deterioration in the strength characteristics of the concrete. Moreover, they can be used to obtain special concretes, for example, foam concrete, light-weight and cellular concretes. By the addition of ash mineral fraction, the consumption of cement is significantly reduced. In the process of producing asphalt concrete mixtures, it is recommended to use the product of the TPP ash flotation as a mineral powder; the mixture thus obtained is cheaper than that with a limestone mineral powder. It is not inferior in terms of performance and strength parameters. In the manufacture of wall products, a modified ash of low-reactive coal added to clay may serve as raw material. The obtained burned bricks exhibit higher strength characteristics than the material with no modified ash admixtures.

The promising direction of the use of ash wastes is their processing into various fillings for building materials. Lightweight fillers are made by granulation of ash with the subsequent firing of the resulting pellets. Ash, together with loam and waste admixture from a lead mining and processing enterprise, is part of the mixture for agglomerated pyrite. A method for obtaining artificial filler has been proposed, which involves the separation of ash into two classes, with further processing the class that contains about 4.5% of unburned coal residuals.

Methods for the production of sorbents. Carbon sorbents may be obtained by high temperature coal processing and, in particular, by

combustion. The optimum carbon content of the sorbents can be achieved by controlling the coal combustion process or by separating the resulting products. The separation process can occur in air medium (electro- and pneumatic separation) and water medium (flotation, selective flocculation, etc.).

The major parameters that influence the carbon content in the sorbents derived from the coal combustion are the dust dispersity and excess air coefficient. Coarsening the coal grinding and a decrease in the excess air coefficient lead to an increase in the carry-over carbon content. Some methods for obtaining sorbents, which have been developed by our team, are listed below.

The technique refers to the ways of enrichment of dispersed materials having electrical properties. It can be used for enrichment of clayed carbon-containing materials that have undergone heat treatment, for example, ash from thermal power plants.

The technique ensures obtaining a carbon-containing fraction with a carbon content of up to 50% and high-ash waste with ash content of 98%.

The purpose can be reached in the following way: the fuel combustion is performed at a temperature of 900 – 1800°C at a fuel flow rate, the corresponding excess air coefficient of 1-2, and a voltage of 20-30 kV in the first field of high-resistivity electrostatic precipitators, while the ash enrichment occurs directly in the electrostatic precipitator.

Ash component of the ash is captured in the first field of high-resistivity electrostatic precipitators, while carbon-containing particles slip through this area and are caught in the second field, or directly in front of the smoke exhauster. At the same time, the highest possible voltage (close to the breakdown) is being maintained in the second field of the high-resistivity electrostatic precipitators. The temperature of the dust-air mixture is above 100°C.

The variations of the excess air ratio during heat treatment and the voltage on the electrodes forming corona in the process of separation have shown an unexpected result, namely, a product containing up to 78% of carbon is collected in the second field and after the electrostatic precipitators. The separation efficiency is enhanced, firstly, due to the reduction of adhesion properties of the system at a

temperature above the dew point; secondly, as a result of the individual movement of particles in the dust-loaded air stream under the action of the electric field, in contrast to conventional ball electrostatic separation in the electronic drum separator.

When the temperature of the dust-air mixture is below the dew point, the electrical resistance of the system dramatically decreases due to the condensation of water vapor, which leads to a deterioration of the product separation and capture.

The product has beneficial properties and can be used as a substitute for low-sulfur fuel in metallurgical processes, as a heat insulating filler, low-cost sorbent, etc.

The collection electrodes in the first field of the high-resistivity electrostatic precipitator collect a product which is an aluminosilicate fraction with an ash content of up to 98%. The fraction is a dry loose mass, which does not require additional preparation for use in construction, in the preparation of various suspensions, etc.

The separation properties of the high-resistivity electrostatic precipitator with a relatively weak electric field are based on the fact that particles with different resistivity are captured with different efficiency. Fly ash particles about 50 μm in size with a resistivity greater than $10^2 \text{ ohm}\cdot\text{m}$ are captured effectively, while unburned coal with the particle size of about 120 μm and a resistivity of less than $10^2 \text{ ohm}\cdot\text{m}$ easily lose the negative charge as they arrive at the collecting electrode. As a consequence, they acquire a positive charge and are repulsed by the collection electrode, re-entrained in the gas stream and carried out of the high-resistivity electrostatic precipitator.

There is an assumption that coal ash particles of an irregular shape may give off their charge, even without touching the electrode, as a result of a corona discharge that occurs between a grounded electrode and negatively charged particles.

In the second field of the high-resistivity electrostatic precipitators operating with relatively high voltage (close to a breakdown), coal particles show less tendency to be repulsed from the collecting electrode due to the electric field strength.

More than 50% of coal particles pass through the second field and settle on the collection electrode of this field. After leaving the high-resistivity electrostatic precipitator, the uncaptured carbon particles

are trapped by well-known methods, e.g. air cyclones, sleeve filters or other devices installed in front of the smoke exhaust.

A decrease in the excess air coefficient, and consequently, in the velocity of the gas stream, is accompanied by underburn in the boiler, which leads to an increase in the amount of carbon-containing particles in the stream. In case of low coal combustion, it is possible to obtain products with the value exceeding the value of unburned coal taking into account the cost of its handling and processing. The carbon content of the enriched product was obtained with an excess air coefficient $\alpha = 1 - 2$. At higher values, the effect of enrichment is sharply reduced and heat losses increase. With a reduced excess air coefficient (less than 1.0), the percentage of underburn can exceed 80%, which makes the process economically disadvantageous. In addition, the abrasive wear of metal surfaces rises. When excess air coefficient is raised ($\alpha = 2$), the concentration of carbon in the enriched carbonaceous product decreases dramatically, which is also ineffective. The tables show that the quality of the resulting products is determined by the excess air coefficient and the voltage applied to the electrodes. Voltage below 0.95 of breakdown results in a decrease in carbon content of the enriched product.

The proposed method of electric enrichment of a clayed carbon-containing product will provide the production of desired aluminosilicate and carbon-containing raw materials. The aluminosilicate product may be used as a main component in the manufacture of wall products and in the brick industry, and as an additive to various mixtures. The carbonaceous product may be used as a low-cost sorbent and a substitute for low-sulfur fuel in metallurgical processes, as a heat-insulating additive and for other purposes.

CONCLUSIONS

The analysis of the obtained results gives grounds for the following conclusions:

1. Concrete obtained on the basis of alumina-silicate spheres isolated from fly ash can be used for the production of lightweight concrete, external wall structures, small building blocks, and also in monolithic housing construction. In comparison with the existing

compositions of expanded clay lightweight concrete, the compositions containing the TPP waste allow to save 20 to 30% of the binder, the performance characteristics being maintained.

2. Preliminary vibro-impact activation allows increasing the reactivity of aluminosilicates due to the formation of new effective bonds, which improves the physico-mechanical properties of binders based on them.

3. The study complies with the high-priority strategic directions in mining industry and is devoted to the development and investigation of multipurpose sorbents to be used in environmentally friendly technologies for mining and processing minerals (coal, ores, ferrous and non-ferrous metals, etc.).

4. It has been discovered that in the case of high-temperature coal processing (within the range of 800-1300 ° C), sorbents of multipurpose use can be obtained.

5. It has been found that the most effective low-cost sorbents may be produced at TPPs with the controlled combustion of coal of certain grades.

6. The mechanism of action of carbon-containing sorbents was studied. It has been found that adsorption of electrolytes occurs due to donor-acceptor interaction, while apolar surface-active substances are adsorbed due to the formation of strong structures.

7. A new mechanism of sorption by certain components of carbon-containing ash (microspheres), which are spherical hollow particles, has been established.

8. Identification of sorbents was carried out; it has been found that the adsorption of organic acids by microspheres obeys the Traube series.

9. Analysis of the microporous structure revealed the presence of micro- and macropores in the sorbents.

10. It has been established that given their adsorption properties, the sorbents under study are amphoteric adsorbents, i.e., they are capable of adsorbing both acids and hydroxides. This implies they can adsorb complex electrolytes, though their anion exchange properties are only expressed in the case of high-temperature processing.

11. The parameters of metal cation adsorption by sorbents of various compositions have been evaluated. The kinetic

characteristics of the isotope exchange of cations in the oxidized coal were determined in the state of the adsorption equilibrium of the coal – solution system.

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OPTIMIZATION OF PREPAREDNESS FOR EXTRACTION OF BALANCE-INDUSTRIAL MINERAL RESERVES

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Abstract

Investigational provision of extractive power-shovels industrially-balance supplies with the different degree of preparedness to the booty and quarry overall depends on a time domain between loosening of array of ferrous quartzite's. Time domain between loosening of array of ferrous quartzite's in coalfaces and a time domain between the mass loosening can be anything. It is well-proven that effective work of quarry in the case when time domains between the mass loosening of array of balance-industrial supplies and loosening of array of ferrous quartzite's in coalfaces gather. Reasonably, that an optimal time domain between the mass loosening of balance-industrial supplies in iron-ore careers is in limits from two to three weeks.

Forming each of single streams of iron-ore mass it takes place under act of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass. Therefore, taking into account influence of losses of balance-industrial supplies and the obstruction of content of quality indexes of minerals in the stream of iron-ore mass execute calculations of signs of content of quality indexes of minerals in the single streams of iron-ore mass. Base the decision of these task expressions of balance of amount of iron-ore mass and amount of content of quality indexes of the iron related to magnetite at working off areas of deposits of hard minerals.

The offered methodology of setting of norms of the balance-industrial supplies prepared to the booty is approved on the careers of Krivbass and methodology of setting of norms of preparedness of the balance-industrial supplies prepared to the booty, developed for operating mining industry enterprises modernized and adjusted to the use on the stage of planning. Setting of norms of balance-industrial supplies of hard minerals consists in determination of optimal correlation of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in a stream iron-ore mass, characterize that a parameter for the case of working off contacts longitudinal lift.

In relation to work of one extractive unit in a *i-y* change have changeable indexes of booty, losses of balance-industrial supplies, obstruction of content of quality indexes of minerals and content of quality indexes of the iron related to magnetite, what deposits set for areas, used in a *i-y* change. Thus examine change abilities of indexes of technological process as casual functions during the fixed period (change, ten-day period, month, quarter and other), that mean that during work of separate mining unit and all ore mining enterprise) amount and content of quality indexes of the obtained and lost balance-industrial supplies, amount of breeds of obstruction and content of quality indexes for them averaging out useful component examine as casual processes and for their description the mathematical vehicle of theory of casual functions (is used cross-correlation functions).

Entry. Productive work of every extractive unit at the booty of ferrous quartzites arrive at an open method, if certain accordance sticks to between the different project technological types of mountain works. Planning of development of mountain works in the process of exploitation of balance-industrial supplies of areas of array of hard minerals of deposit is the important stage in the decision of questions of technology of mountain production that provides plenitude of mastering of balance supplies of bowels of the earth [1].

At the annual planning of development of mountain works go into detail and specify perspective plans, and also decide concrete technological questions: establishment of volumes of pre-production mining and threaded works taking into account norms on the degree of preparedness of the prepared and ready to the booty balance-industrial supplies exposed, and also task on the volume of commodity products; set experience works that is sent to the improvement of booty of balance-industrial supplies from the bowels of the earth; determine the rational amount of simultaneously working extractive units with the aim of providing of necessary amount and quality of commodity products; set the optimal loading, fold the calendar graphic arts of booty of balance-industrial supplies of ferrous quartzites on every extractive unit and determine terms their redemption.

Provision of every extractive power-shovel the industrially-balance supplies prepared to the booty with the different degree of preparedness to the booty, and quarry on the whole, depends on a time domain between loosening of array of ferrous quartzites. Effective work of quarry will be in that case, when time domains between the mass loosening of array of balance-industrial supplies and loosening of array of ferrous quartzites in coalfaces gather, then, when time of mass explosion coincides in all extractive coalfaces. An optimal time domain between the mass loosening of array of balance-industrial supplies in iron-ore careers is in limits from two to three weeks [2,3].

An aim of work is development of methodology of setting of norms of ready of the booty of balance-industrial supplies.

For the achievement of the aim such tasks are untied:

– it is an analysis of present methods of setting of norms of ready to boot of balance-industrial supplies;

- it is an improvement of existent methodologies of setting of norms of the balance-industrial supplies prepared to the booty;
- it is establishment of norms of the balance-industrial supplies prepared to the booty.

An idea of work is an analysis and determination of methods of calculation of optimal of the balance-industrial supplies prepared to the booty for development of economy of ore-mining enterprises and indexes of plenitude of the use of resources of bowels of the earth at present labour and material sources.

A research object is the balance-industrial supplies of ferrous quartzites prepared to the booty.

The article of research is methodology of setting of norms of ready to the booty of balance-industrial supplies.

Initial positions of optimization of preparedness are to the booty of balance-industrial supplies. Methodology of optimization preparedness to the booty of balance-industrial supplies and system of content of quality indexes of minerals in the stream of iron-ore mass at the open method of booty in comparing to underground, at general methodical approach, has substantial differences [4,5]. Firstly extractive unit at the underground method of booty keeps balance reserves that determine the parameters of the accepted system of development, and optimization of preparedness to the booty of balance-industrial supplies is optimization of number of extractive units. Secondly on careers ready to the booty, the prepared and exposed industrially-balance supplies change in wide limits at the same number of extractive power-shovels. There is a task to optimization of not only number of extractive power-shovels but also their provision of the industrially-balance supplies prepared to the booty and in addition, at an open method in the process of averaging out of content of quality indexes of minerals in the stream of iron-ore mass of value has direction of mining on extractive ledges.

If at the underground method of booty, optimizing the system characterize a presence two independent changeable are numbers of extractive units on an ore-mining enterprise and capacity of composition of averaging out of content of quality indexes of minerals in iron-ore mass, then at the open method of booty of balance-industrial supplies to these two add changeable two is a provision of the industrially-balance supplies prepared to the booty

and direction of mining. Optimization at four and mathematical vehicle let to untie such task. However for realization number of independent changeable expediently and it maybe to shorten. Create the industrially-balance changeable is difficult and bulky, but will be carried out, so as the worked out methodology supplies prepared to the booty on a career with the aim of providing of the productivity of extractive power-shovels. Determine necessary front of works of one extractive power-shovel under right technological planning of enterprises, that is why size balance-industrial supplies that provide one extractive power-shovel prepared to the booty, determine coming from his productivity and normative front of extractive works. If the examine the provision of quarry the industrial supplies prepared to the booty as dependent changeable, conditioned by the number of extractive units, in connection with it independent changeable during optimization will shorten to three: number of extractive units; direction of booty of balance-industrial supplies; a volume of composition of averaging out of content of quality indexes of minerals is in iron-ore mass.

Optimization of balance-industrial supplies after the degree of preparedness to the booty together with the system of averaging out content of quality indexes of minerals in the stream of iron-ore mass carry out separately for every category of balance-industrial supplies and for all categories simultaneously.

In first case, during separate optimization, for the every category of balance-industrial supplies estimate the economical consequences of their changeability: from one side, charges on realization of mountain works for creation of ready to the booty, prepared and exposed balance-industrial supplies, from other is a loss from changeability. To define an economic effect from changeability of the prepared and exposed balance-industrial supplies will execute through the estimation of their influence on the size of the balance-industrial supplies prepared to the booty, on the number of extractive units. Between industrial supplies ready to the booty, prepared and exposed on a career there are dependences that is set in relation to the terms of booty balance-industrial supplies from an iron-ore deposit, bed, ore body or areas of array of minerals. During separate optimization of every category of balance-industrial supplies together with the system of averaging out of content of quality

indexes of minerals in the stream of iron-ore mass will set the objective functions of optimization for ready to the booty of E_z , prepared E_n and exposed E_p of balance-industrial supplies. Thus in general case determine every objective function as a function of changeability three independent changeable: numbers of mining units of N ; straight to the booty of hard minerals φ ; to the volume of composition of averaging out of content of quality indexes of minerals in iron-ore mass of V .

$$E_r = F_r(N, \varphi, V) \quad (1)$$

$$E_n = F_n(N, \varphi, V) \quad (2)$$

$$E_p = F_p(N, \varphi, V). \quad (3)$$

Optimizations each of objective functions (1)–(3) specifies on existence of local optimum, that is answered by the optimal value of capacity of composition of averaging out of content of quality indexes of minerals in iron-ore mass and means that in this case (for one system of averaging out of content of quality indexes of minerals in iron-ore mass) it is impossible simply to define the volume of composition of averaging out of content of quality indexes of minerals in iron-ore mass [7,8]. On a career simultaneously all categories of balance-industrial supplies (ready to the booty, preparation and exposed) function from the methodical point of view expedient determination of function total economic effect.

$$E = E_z + E_n + E_p = F(N, \varphi, V) \quad (4)$$

At most the function of total economic effect allows simply setting optimal values of independent changeable:

a) Numbers of extractive units, straight to the booty of balance industrial supplies and volume of composition of averaging out of content of quality indexes of minerals in iron-ore mass;

б) Optimal values of dependent changeable – ready to the booty, prepared and exposed balance-industrial supplies.

Dependences are between the industrial supplies losses of balance-industrial supplies and obstruction of content of quality indexes of minerals prepared to the booty. In accordance with the worked out methodology of norm of balance-industrial supplies after the degree of preparedness to the booty, the norms of the industrial supplies prepared to the booty on the careers of Krivbass determine from the count losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass at a booty. An account shows that the losses of balance-industrial supplies result in an increase, and obstruction of content of quality indexes of minerals – to reduction of normative size of the balance-industrial supplies prepared to the booty. In this connection during optimization of quality indexes of booty of balance-industrial supplies from the bowels of the earth take into account their influence on normative preparedness to the booty of balance-industrial supplies, that is only one of consequences of connection losses of balance-industrial supplies, obstruction of content of quality indexes of minerals in the stream of iron-ore mass and preparedness to the booty of balance-industrial supplies of minerals [8,9]. For consideration of this question the conducted analysis of indexes of mining of balance-industrial supplies is from the bowels of the earth and state of preparedness to the booty of balance-industrial supplies of hard minerals. Statistical processing of current data of ore-mining enterprises of Krivbass, used for establishment of dependence of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass at a booty from the provision of the industrially-balance supplies prepared to the booty and obstruction quality indexes of minerals in the stream of iron-ore mass on the mines of Krivbass (fig. 1 and fig. 2).

By the nature of these dependences next conclusions are done:

a) dependence of obstruction of content of quality indexes of minerals in iron-ore mass from value of the balance-industrial supplies prepared to the booty near to hyperbolic and testifies that on the mines of Krivbass at the insufficient provision of the industrially-balance supplies of obstruction of content of quality indexes of minerals prepared to the booty at iron-ore mass grows;

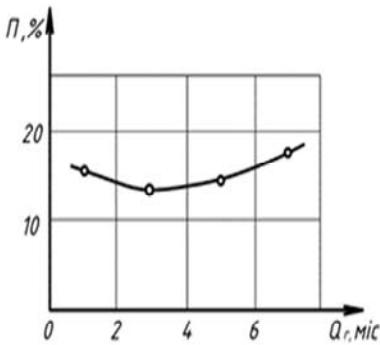


Fig. 1. Dependence of losses of balance-industrial supplies at a booty (Π) from the provision of the industrial supplies (Q_r) prepared to the booty on the mines of Krivbass

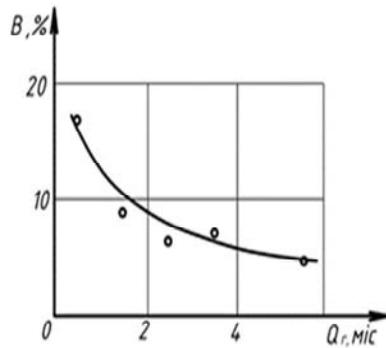


Fig. 2. Dependence of obstruction of content of quality indexes of minerals in iron-ore mass at a booty (B) from the provision of the industrial supplies prepared to the booty on the mines of Krivbass

б) at providing the industrially-balance supplies prepared to the booty in iron-ore mass changes near to the normative level of obstruction content of quality indexes of useful minerals and doesn't depend from the provision of mine the industrially-balance supplies prepared to the booty. Explain character of the set conformity to law that in the conditions of deficit of the balance-industrial supplies prepared to the booty of ore-mining enterprise, in order to avoid blowing off plan tasks from the booty of balance-industrial supplies enterprise, in order to avoid blowing off plan tasks from the booty of balance-industrial supplies, increases the amount of breeds of obstruction of content of quality indexes of minerals, bring over that to the booty of unstandardized supplies.

в) Dependence of losses of balance-industrial supplies on the industrial supplies prepared to the booty does not have specific character (a cross-correlation relation presents 0,4–0,5) clearly. In area of deficit of the industrial supplies prepared to the booty the average of losses of balance-industrial supplies grows, and at the surplus provision of the industrial supplies of hard minerals of loss of balance-industrial supplies prepared to the booty also have a tendency to the increase.

An analogical analysis is conducted from actual data of work of iron-ore quarries of Krivbass. The set dependences of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass from the provision of the industrial supplies prepared to the booty are brought around to fig. 3 and on fig. 4.

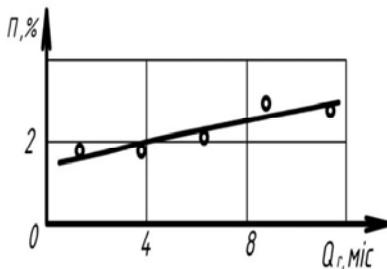


Fig. 3. Dependence of losses of balance-industrial supplies at a booty (π) from the provision of the industrial balance supplies (Q_r) prepared to the booty on the careers of Krivbass

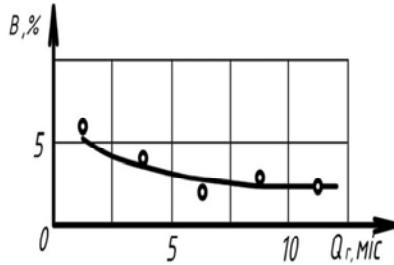


Fig. 4. Dependence of obstruction of content of quality indexes of minerals in iron-ore mass at booty (B) from the provision of the industrial balance supplies (Q_r) prepared to the booty on the careers of Krivbass.

Character of these dependences shows that they have the same conformities to law, as well as dependences that is set for ore-mining enterprises with the underground method of booty, namely the obstruction of content of quality indexes of minerals in iron-ore mass at the booty of balance-industrial supplies increases at the deficit of the industrial supplies loss of balance-industrial supplies prepared to the booty grow at the surplus provision of the industrial supplies prepared to the booty.

The set dependences specify on the existent ore, technological, organizational and other terms of mining from iron-ore deposits, beds, ore bodies or areas. These dependences take place, when ore-mining enterprises work in the conditions of surplus or deficit of the industrial supplies prepared to the booty. The set dependences show that passing of ore-mining enterprises to work with normative preparedness of balance-industrial supplies to the booty will allow on iron-ore careers to bring down the obstruction of content of quality

indexes of minerals in the stream of ore mass approximately on 1,5–2 and on mines to bring down the losses of balance-industrial supplies on 2.

Got results testify that setting of norms of balance-industrial supplies after the degree of preparedness to the booty and realization of norms in a production is an effective measure in area of guard of bowels of the earth, that will allow to decrease the losses of balance-industrial supplies and obstructions to content of quality indexes of minerals in iron-ore mass.

Influence of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals is in iron-ore mass on the process of averaging out. Development of ore body or areas of array of hard minerals of deposit is accompanied by the losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass by barren layers and containing breeds. Losses of balance-industrial supplies and obstructions of content of quality indexes of minerals in flow of iron-ore mass influences on homogeneity of content of qualities indexes of constituents of iron-ore mass. Under act of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass oscillation of content of quality indexes of useful components grows in the obtained iron-ore mass. The calculation of indexes of homogeneity of content of quality composition of iron-ore mass during exploitation and planning on a ore-mining enterprise takes into account influence of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals is on the process of averaging out of content of quality indexes of minerals. There is question, as homogeneity of content of quality indexes of useful shock-tench in the stream of iron-ore mass related to the losses of balance-industrial supplies and obstruction of content of quality indexes of minerals. Homogeneity of quality composition of total ore mining stream of iron-ore mass depends on homogeneity of single streams that come from the coalfaces of separate extractive units. In turn, forming each of single streams of iron-ore mass it takes place under act of losses of balance-industrial supplies and obstruction of content of quality indexes of useful minerals in the

stream of iron-ore mass. Therefore taking into account influence of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass will execute the calculations of signs of content of quality indexes of minerals is in the single streams of iron-ore mass. Base the decision of this task expressions of balance of amount of iron-ore mass and amount of content of quality indexes of the iron related to magnetite at working off the areas of deposit of hard minerals with the industrially-balance supplies of B [7,12]

$$D = B - \Pi + B, \quad (5)$$

$$Da = Bc - \Pi c_{\Pi} + Bb, \quad (6)$$

where D , Π is accordingly amount of the obtained and lost balance-industrial supplies, thousands of τ ; B is an amount of breeds of obstruction content of quality indexes of minerals, thousands of τ ; a , c , c_{Π} , b is content of quality indexes of useful component accordingly in obtained, in an array, in the lost balance-industrial supplies and in the breeds of obstruction of content of quality indexes of minerals.

Formulas (5) and (6) just for any area of minerals develop that, that is why will use them for the estimation of amount of content of quality indexes of the iron related to magnetite and to content of quality indexes of minerals in the stream of iron-ore mass, by obtained mining unit in a fixed period of time (hour, change, daytime). For example, in relation to work of one extractive unit in a i -y change expressions (5) and (6) have changeable indexes of booty, losses of balance-industrial supplies, obstruction of content of quality indexes of minerals and content of quality indexes of the iron related to magnetite, that is set for the areas of bed exhaust in a i -y change. Thus change abilities of indexes of D_i , Π_i , B_i , C_i et al examine as casual functions during a corresponding period (twenty-four hours, ten-day period, month, quarter and other), that mean, that during work of separate extractive unit or all ore-mining enterprises amount and content of quality indexes of obtained and the lost balance-industrial supplies, amount of breeds of obstruction and content of quality indexes for them averaging useful component examine as casual processes and for their description the mathematical vehicle of theory of casual functions is

used. In the quality indexes of descriptions of casual processes, losses of balance-industrial supplies and obstruction of content of quality indexes of minerals determine accordingly the cross-correlation functions of $r_{\mathcal{D}\mathcal{D}}, r_{\mathcal{D}\mathcal{I}}, r_{\mathcal{B}\mathcal{B}}$. If in a formula (6) to put the value of B , that certainly from equality (5), then expression is for balance of amount of content of quality indexes of the iron related to magnetite, at working off the area of bed will be

$$\mathcal{D}_i a_i = \mathcal{D}_i c_i + \mathcal{I}_i c_i + B_i b_i - B_i c_i - \mathcal{I}_i c_{\mathcal{I}i}, \quad (7)$$

For determination of correlative functions of right and left parts of equality (7) determine the presence of cross-correlation of accidental functions, that is included in expression of balance of amount of content of quality indexes of the iron related to magnetite. Task decides taking into account cross-correlation all accidental functions of $\mathcal{D}_i, \mathcal{I}_i, B_i, C_i$ and. Thus limited to consideration of characteristic case of booty of balance-industrial supplies from a deposit at that casual functions of $\mathcal{D}_i, \mathcal{I}_i, B_i$ mutually correlates, and all other casual functions, that is included in expression (7) – does not correlate. For these terms, taking into account, that X and X' values of casual function, that attributing to the different moments of time of t and t' (to the different crossing of casual function) we have

$$\begin{aligned} r_{\mathcal{D}\mathcal{D}} r_{aa'} &= \overline{\mathcal{D}\mathcal{D}}' r_{aa'} + \overline{aa'} r_{\mathcal{D}\mathcal{D}} = r_{\mathcal{D}\mathcal{D}} r_{cc'} + \overline{\mathcal{D}\mathcal{D}}' r_{cc'} + \overline{cc'} r_{\mathcal{D}\mathcal{D}} + r_{\mathcal{I}\mathcal{I}\mathcal{I}} r_{cc'} + \overline{\mathcal{I}\mathcal{I}\mathcal{I}}' r_{cc'} + \\ &+ \overline{cc'} r_{\mathcal{I}\mathcal{I}\mathcal{I}} + r_{\mathcal{B}\mathcal{B}\mathcal{B}} r_{cc'} + \overline{\mathcal{B}\mathcal{B}\mathcal{B}}' r_{cc'} + \overline{cc'} r_{\mathcal{B}\mathcal{B}} + r_{\mathcal{I}\mathcal{I}\mathcal{I}} r_{c_{\mathcal{I}i}c_{\mathcal{I}i}'} + \overline{\mathcal{I}\mathcal{I}\mathcal{I}}' r_{c_{\mathcal{I}i}c_{\mathcal{I}i}'} + \overline{c_{\mathcal{I}i}} \overline{c_{\mathcal{I}i}'} r_{\mathcal{I}\mathcal{I}\mathcal{I}} + \\ &+ r_{\mathcal{B}\mathcal{B}} r_{bb'} + \overline{\mathcal{B}\mathcal{B}}' r_{bb'} + \overline{bb'} r_{\mathcal{B}\mathcal{B}} + 2r_{\mathcal{D}c, \mathcal{I}c} - 2r_{\mathcal{D}c, \mathcal{B}c} - 2r_{\mathcal{D}c, \mathcal{I}c_{\mathcal{I}i}} + 2r_{\mathcal{D}c, \mathcal{B}b} - 2r_{\mathcal{I}c, \mathcal{B}c} - \\ &- 2r_{\mathcal{I}c, \mathcal{I}c_{\mathcal{I}i}} + 2r_{\mathcal{I}c, \mathcal{B}b} + 2r_{\mathcal{B}c, \mathcal{I}c_{\mathcal{I}i}} - 2r_{\mathcal{B}c, \mathcal{B}b} - 2r_{\mathcal{I}c_{\mathcal{I}i}, \mathcal{B}b}, \end{aligned} \quad (8)$$

where $r_{xy, zu}$ is a mutual covariance function of product of $X \times Y$ and $Z \times U$ casual functions of X, Y, Z, U ; $M[x] = \bar{x}$ it is a mathematical hope of casual function of X .

Determine mutual covariance functions that is included in expression (8) as

$$\begin{aligned} r_{\mathcal{D}c, \mathcal{I}c} &= M[\mathcal{D}c \mathcal{I}c] - M[\mathcal{D}c] M[\mathcal{I}c] = \\ &\overline{\mathcal{D}\mathcal{I}} \overline{c}^2 - \overline{\mathcal{D}c} \overline{\mathcal{I}c} = \overline{\mathcal{D}\mathcal{I}} \sigma_c^2 r_{\mathcal{D}\mathcal{I}} (\sigma_c^2 + c^2), \end{aligned} \quad (9)$$

where $r_{\mathcal{D}\mathcal{I}}$ is a mutual covariance function of casual functions of \mathcal{D}_i and \mathcal{I}_i .

$$\begin{aligned} \text{Determine like } r_{\mathcal{D}c,\mathcal{I}c} &= \overline{\mathcal{D}\mathcal{B}}\sigma_c^2 + (\sigma_c^2 + \bar{c}^2)r_{\mathcal{D}\mathcal{B}}; \quad r_{\mathcal{D}c,\mathcal{I}c_{\mathcal{I}}} = \bar{c}\bar{c}_{\mathcal{I}}r_{\mathcal{D}\mathcal{I}}; \\ r_{\mathcal{D}c,Bb} &= \bar{c}\bar{b}r_{\mathcal{D}\mathcal{B}}; \quad r_{\mathcal{I}c,Bc} = \overline{\mathcal{I}\mathcal{B}}\sigma_c^2 + (\sigma_c^2 + \bar{c}^2)r_{\mathcal{I}\mathcal{B}}; \quad r_{\mathcal{I}c,\mathcal{I}c_{\mathcal{I}}} = \bar{c}\bar{c}_{\mathcal{I}}\sigma_{\mathcal{I}}^2; \\ r_{\mathcal{I}c,Bb} &= \bar{c}\bar{b}r_{\mathcal{I}\mathcal{B}}; \quad r_{Bc,\mathcal{I}c_{\mathcal{I}}} = \bar{c}\bar{c}_{\mathcal{I}}r_{B\mathcal{I}}; \quad r_{Bc,Bb} = \bar{c}\bar{b}\sigma_B^2; \quad r_{\mathcal{I}c_{\mathcal{I}},Bb} = \bar{c}_{\mathcal{I}}\bar{b}r_{\mathcal{I}\mathcal{B}}. \end{aligned}$$

Putting the value of covariance functions in expression (8) and uniting him relatively $r_{aa'}$, determine

$$\begin{aligned} r_{aa'} &= \frac{1}{r_{\mathcal{D}\mathcal{D}'} + \overline{\mathcal{D}\mathcal{D}'}} [(r_{cc'} + \bar{c}\bar{c}') (r_{\mathcal{D}\mathcal{D}'} + r_{\mathcal{I}\mathcal{I}'} + r_{B\mathcal{B}'}) + r_{cc'} (\overline{\mathcal{D}\mathcal{D}'} + \overline{\mathcal{I}\mathcal{I}'} + \overline{B\mathcal{B}'}) + \\ &+ r_{c_{\mathcal{I}c_{\mathcal{I}}},c_{\mathcal{I}c_{\mathcal{I}}'}} (r_{\mathcal{I}\mathcal{I}'} + \overline{\mathcal{I}\mathcal{I}'}) + r_{bb'} (r_{B\mathcal{B}'} + \overline{B\mathcal{B}'}) + (\bar{c}\bar{c}' - 2\bar{c}\bar{c}_{\mathcal{I}}) r_{\mathcal{I}\mathcal{I}'} + (\bar{b}\bar{b}' - 2\bar{c}\bar{b}') r_{B\mathcal{B}'} - \\ &\overline{a\bar{a}'} r_{\mathcal{D}\mathcal{D}'} + 2r_{\mathcal{D}\mathcal{I}'} (\sigma_c^2 + \bar{c}^2 - \bar{c}\bar{c}_{\mathcal{I}}) - 2r_{\mathcal{D}\mathcal{B}} (\sigma_c^2 + \bar{c}^2 - \bar{c}\bar{b}) - \\ &- 2r_{\mathcal{I}\mathcal{B}} (\sigma_c^2 + \bar{c}^2 - \bar{c}\bar{b} - \bar{c}\bar{c}_{\mathcal{I}} + \bar{c}_{\mathcal{I}}\bar{b}) + 2\sigma_c^2 (\overline{\mathcal{D}\mathcal{I}\mathcal{I}} - \overline{\mathcal{D}\mathcal{B}} - \overline{\mathcal{I}\mathcal{B}})]. \end{aligned} \quad (10)$$

Expression (10) shows, copulas between quality signs, characterize the processes of averaging-out of content of quality indexes of minerals that in iron-ore mass, losses of balance-industrial supplies and obstructions of content of quality indexes of minerals during work of separate extractive unit. In case at $t=t'$ formula (10) will use for determination to dispersion of content of quality indexes of averaging out useful component in the obtained balance-industrial supplies.

$$\begin{aligned} \sigma_a^2 &= \frac{1}{\sigma_{\mathcal{D}}^2 + \overline{\mathcal{D}^2}} [(\sigma_c^2 + \bar{c}^2)(\sigma_{\mathcal{D}}^2 + \sigma_{\mathcal{I}}^2 + \sigma_B^2) + \sigma_c^2(\overline{\mathcal{D}^2} + \overline{\mathcal{I}^2} + \overline{B^2}) + \\ &+ \sigma_{c_{\mathcal{I}c_{\mathcal{I}}}}^2(\sigma_{\mathcal{I}}^2 + \overline{\mathcal{I}^2}) + \sigma_b^2(\sigma_B^2 + \overline{B^2}) + (\bar{c}^2 - 2\bar{c}\bar{c}_{\mathcal{I}})\sigma_{\mathcal{I}}^2 + (\bar{b}^2 - 2\bar{c}\bar{b}')\sigma_B^2 - \\ &\overline{a^2}\sigma_{\mathcal{D}}^2 + 2K_{\mathcal{D}\mathcal{I}} (\sigma_c^2 + \bar{c}^2 - \bar{c}\bar{c}_{\mathcal{I}}) - 2K_{\mathcal{D}\mathcal{B}} (\sigma_c^2 + \bar{c}^2 - \bar{c}\bar{b}) - \\ &- 2K_{\mathcal{I}\mathcal{B}} (\sigma_c^2 + \bar{c}^2 - \bar{c}\bar{b} - \bar{c}\bar{c}_{\mathcal{I}} + \bar{c}_{\mathcal{I}}\bar{b}) + 2\sigma_c^2 (\overline{\mathcal{D}\mathcal{I}\mathcal{I}} - \overline{\mathcal{D}\mathcal{B}} - \overline{\mathcal{I}\mathcal{B}})]. \end{aligned} \quad (11)$$

where $K_{\mathcal{D}\mathcal{I}}$, $K_{\mathcal{D}\mathcal{B}}$, $K_{\mathcal{I}\mathcal{B}}$ is cross-correlation moments of sizes of \mathcal{D} and \mathcal{I} , \mathcal{D} and B , \mathcal{I} and B . Will consider cases at the booty of balance-industrial supplies.

1. The obstruction of content of quality indexes of minerals passes gobs. For determination of f covariance function in a formula (10) assume that $\bar{b} = \bar{b}' = 0$, $r_{bb'} = 0$.

$$r_{aa'} = \frac{1}{r_{\mathcal{M}\mathcal{M}'} + \overline{\mathcal{M}\mathcal{M}'}} [(r_{cc'} + \overline{c\bar{c}'}) (r_{\mathcal{M}\mathcal{M}'} + r_{\mathcal{M}\mathcal{M}'} + r_{\mathcal{B}\mathcal{B}'}) + r_{cc'} (\overline{\mathcal{M}\mathcal{M}'} + \overline{\mathcal{M}\mathcal{M}'} + \overline{\mathcal{B}\mathcal{B}'}) + r_{c_n c'_n} (r_{\mathcal{M}\mathcal{M}'} + \overline{\mathcal{M}\mathcal{M}'}) + r_{\mathcal{M}\mathcal{M}'} (\overline{c\bar{c}'} - 2\overline{c\bar{c}'_n}) - \overline{a\bar{a}'} r_{\mathcal{M}\mathcal{M}'} + 2r_{\mathcal{M}\mathcal{M}'} (\sigma_c^2 + \overline{c^2} - \overline{c\bar{c}'_n}) - 2r_{\mathcal{M}\mathcal{B}} (\sigma_c^2 + \overline{c^2}) - 2r_{\mathcal{M}\mathcal{B}} (\sigma_c^2 + \overline{c^2} - \overline{c\bar{c}'_n}) + 2\sigma_c^2 (\overline{\mathcal{M}\mathcal{M}'} - \overline{\mathcal{M}\mathcal{B}} - \overline{\mathcal{M}\mathcal{B}})]. \quad (12)$$

For the calculations of dispersion of content of quality indexes of useful component at the obstruction of content of quality indexes of minerals assume gobs, that $\overline{b} = 0$ and $\sigma_b = 0$. Will have

$$\sigma_a^2 = \frac{1}{\sigma_{\mathcal{M}}^2 + \overline{\mathcal{M}^2}} [(\sigma_c^2 + \overline{c^2}) (\sigma_{\mathcal{M}}^2 + \sigma_{\mathcal{M}}^2 + \sigma_{\mathcal{B}}^2) + \sigma_c^2 (\overline{\mathcal{M}^2} + \overline{\mathcal{M}^2} + \overline{\mathcal{B}^2}) + \sigma_{c_n}^2 (\sigma_{\mathcal{M}}^2 + \overline{\mathcal{M}^2}) + \sigma_{\mathcal{M}}^2 (\overline{c^2} - 2\overline{c\bar{c}'_n}) - \overline{a^2} \sigma_{\mathcal{M}}^2 + 2K_{\mathcal{M}\mathcal{M}'} (\sigma_c^2 + \overline{c^2} - \overline{c\bar{c}'_n}) - 2K_{\mathcal{M}\mathcal{B}} (\sigma_c^2 + \overline{c^2}) - 2K_{\mathcal{M}\mathcal{B}} (\sigma_c^2 + \overline{c^2} - \overline{c\bar{c}'_n}) + 2\sigma_c^2 (\overline{\mathcal{M}\mathcal{M}'} - \overline{\mathcal{M}\mathcal{B}} - \overline{\mathcal{M}\mathcal{B}})]. \quad (13)$$

2. The obstruction of content of quality indexes of minerals is absent. For the calculation of covariance function of content of quality indexes of useful component at this case in a formula (12) will accept, that $\overline{B} = \overline{B'} = 0$, $r_{\mathcal{B}\mathcal{B}'} = 0$. Will get on such conditions

$$r_{aa'} = \frac{1}{r_{\mathcal{M}\mathcal{M}'} + \overline{\mathcal{M}\mathcal{M}'}} [(r_{cc'} + \overline{c\bar{c}'}) (r_{\mathcal{M}\mathcal{M}'} + r_{\mathcal{M}\mathcal{M}'} + r_{\mathcal{M}\mathcal{M}'}) + r_{cc'} (\overline{\mathcal{M}\mathcal{M}'} + \overline{\mathcal{M}\mathcal{M}'}) + r_{c_n c'_n} (r_{\mathcal{M}\mathcal{M}'} + \overline{\mathcal{M}\mathcal{M}'}) + r_{\mathcal{M}\mathcal{M}'} (\overline{c\bar{c}'} - 2\overline{c\bar{c}'_n}) - \overline{a\bar{a}'} r_{\mathcal{M}\mathcal{M}'} + 2r_{\mathcal{M}\mathcal{M}'} (\sigma_c^2 + \overline{c^2} - \overline{c\bar{c}'_n}) + 2\sigma_c^2 \overline{\mathcal{M}\mathcal{M}'}]. \quad (14)$$

For the calculation of dispersion in a formula (13) assume, that $\overline{B} = 0$, $\sigma_B = 0$, $K_{\mathcal{M}\mathcal{B}} = 0$, $K_{\mathcal{M}\mathcal{B}} = 0$. A formula (13) substantially will simplify and she will assume an air

$$\sigma_a^2 = \frac{1}{\sigma_{\mathcal{M}}^2 + \overline{\mathcal{M}^2}} [(\sigma_c^2 + \overline{c^2}) (\sigma_{\mathcal{M}}^2 + \sigma_{\mathcal{M}}^2) + \sigma_c^2 (\overline{\mathcal{M}^2} + \overline{\mathcal{M}^2}) + \sigma_{c_n}^2 (\sigma_{\mathcal{M}}^2 + \overline{\mathcal{M}^2}) + \sigma_{\mathcal{M}}^2 (\overline{c^2} - 2\overline{c\bar{c}'_n}) - \overline{a^2} \sigma_{\mathcal{M}}^2 + 2K_{\mathcal{M}\mathcal{M}'} (\sigma_c^2 + \overline{c^2} - \overline{c\bar{c}'_n}) + 2\sigma_c^2 \overline{\mathcal{M}\mathcal{M}'}]. \quad (15)$$

3. The losses of balance-industrial supplies at a booty are absent. It means that in the formula (10) of value of all indexes, balance-industrial supplies related to the losses from the array of hard minerals, id est $\overline{\mathcal{M}}$, $\overline{c_n}$, $r_{\mathcal{M}\mathcal{M}'}$, accept such that equal a zero. Have thus

$$r_{aa'} = \frac{1}{r_{\mathcal{D}\mathcal{D}'} + \overline{\mathcal{D}\mathcal{D}'}} [(r_{cc'} + \overline{c\bar{c}'}) (r_{\mathcal{D}\mathcal{D}'} + r_{BB'}) + r_{cc'} (\overline{\mathcal{D}\mathcal{D}'} + \overline{B\bar{B}'}) + r_{bb'} (r_{BB'} + \overline{B\bar{B}'}) + r_{BB'} (\overline{b\bar{b}'} - 2\overline{c\bar{c}'}) - \overline{aa'} r_{\mathcal{D}\mathcal{D}'} - 2r_{\mathcal{D}B} (\sigma_c^2 + \overline{c^2} - \overline{c\bar{c}'}) - 2\sigma_c^2 \overline{\mathcal{D}\mathcal{D}'}] \quad (16)$$

Dispersion for this case will get from expression (11), taking on all values of indexes of losses of balance-industrial supplies, that equal a zero.

$$\sigma_a^2 = \frac{1}{\sigma_{\mathcal{D}}^2 + \overline{\mathcal{D}^2}} [(\sigma_c^2 + \overline{c^2}) (\sigma_{\mathcal{D}}^2 + \sigma_B^2) + \sigma_c^2 (\overline{\mathcal{D}^2} + \overline{B^2}) + \sigma_B^2 (\sigma_B^2 + \overline{B^2}) + \sigma_B^2 (\overline{b^2} - 2\overline{c\bar{c}'}) - \overline{a^2} \sigma_{\mathcal{D}}^2 - 2K_{\mathcal{D}B} (\sigma_c^2 + \overline{c^2} - \overline{c\bar{c}'}) - 2\sigma_c^2 \overline{\mathcal{D}\mathcal{D}'}] \quad (17)$$

4. The losses of balance-industrial supplies are absent and the obstruction of content of quality indexes of minerals in the stream of iron-ore mass passes gobs. In this case in a formula (16) accept $\overline{b} = \overline{b}' = 0$, $b = b' = 0$, $r_{bb'} = 0$ and expression for the calculations of covariance function assumes

$$r_{aa'} = \frac{1}{r_{\mathcal{D}\mathcal{D}'} + \overline{\mathcal{D}^2}} [(r_{cc'} + \overline{c\bar{c}'}) (r_{\mathcal{D}\mathcal{D}'} + r_{BB'}) + r_{cc'} (\overline{\mathcal{D}\mathcal{D}'} + \overline{B\bar{B}'}) - \overline{aa'} r_{\mathcal{D}\mathcal{D}'} - 2r_{\mathcal{D}B} (\sigma_c^2 + \overline{c^2}) - 2\sigma_c^2 \overline{\mathcal{D}\mathcal{D}'}] \quad (18)$$

Like from a formula (17) will get expression for the calculations of dispersion of content of quality indexes of useful component

$$\sigma_a^2 = \frac{1}{\sigma_{\mathcal{D}}^2 + \overline{\mathcal{D}^2}} [(\sigma_c^2 + \overline{c^2}) (\sigma_{\mathcal{D}}^2 + \sigma_B^2) + \sigma_c^2 (\overline{\mathcal{D}^2} + \overline{B^2}) - \overline{a^2} \sigma_{\mathcal{D}}^2 - 2K_{\mathcal{D}B} (\sigma_c^2 + \overline{c^2})] \quad (19)$$

In default of losses of balance-industrial supplies and obstruction of content of quality indexes of useful minerals, as follows from formulas (18) and (19), a covariance function and dispersion of content of quality indexes of averaging-out useful component in the obtained balance-industrial supplies depend only on content of quality indexes of useful component in the bowels of the earth.

$$r_{aa'} = \frac{1}{r_{\mathcal{M}\mathcal{M}'} + \overline{\mathcal{M}}^2} (r_{cc'} r_{\mathcal{M}\mathcal{M}'} + \overline{c} \overline{c}' r_{\mathcal{M}\mathcal{M}'} + \overline{\mathcal{M}} \overline{\mathcal{M}}' r_{cc'} - \overline{a} \overline{a}' r_{\mathcal{M}\mathcal{M}'}), \quad (20)$$

$$\sigma_a^2 = \frac{1}{\sigma_{\mathcal{M}}^2 + \overline{\mathcal{M}}^2} (\sigma_c^2 \sigma_{\mathcal{M}}^2 + \sigma_{\mathcal{M}}^2 \overline{c}^2 + \sigma_c^2 \overline{\mathcal{M}}^2 - \overline{a}^2 \sigma_{\mathcal{M}}^2), \quad (21)$$

what confirms the justice of the set dependences.

Analyzing formulas (5)–(19) taking into account the terms of booty of balance-industrial supplies will execute the estimation of influence of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass on homogeneity of quality composition of single stream of iron-ore mass that comes from the coalface (or coalfaces) of separate extractive unit. Changeable in these formulas determine from geological survey data and materials of exploitation of deposit, bed, ore body or areas of array of hard minerals. Difficulties are presented by determination of functions of $r_{BB'}$, $r_{bb'}$, $r_{III'}$, $r_{c\Pi c'}$ at the booty of balance-industrial supplies underground method.

Determination of influence of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals on the processes of booty of balance-industrial supplies and averaging of content of quality indexes of minerals in the stream of iron-ore mass results in necessity expansion and deepening of scientific bases about essence of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals. Examine the obstruction of content of quality indexes of minerals as a process of bringing in the booty of gobs and unstandardized balanced supplies. Point description of process of obstruction of content of quality indexes of minerals with the use of mathematical vehicle to the theory of casual functions that allows in number to estimate influence of obstruction on the process of content of quality indexes of minerals in iron-ore mass.

Like will consider and will describe the losses of balance-industrial supplies as process of losses. In connection with it actual tasks of study of processes of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals are in the stream of iron-ore mass, exposure of conformities to law of their formation and flowing in time at the different systems and methods

of booty of balance-industrial supplies from the areas of array of hard minerals of deposit. The processes of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in general case are uneven, in identical time domains in a booty attract the different keel-bone of breeds of obstruction with different content of quality indexes of useful component, and also lose the different amount of balance-industrial supplies with different content of quality indexes, that influences on the process of averaging of content of quality indexes of minerals in the stream of iron ore mass, increasing oscillation of content of quality composition of averaging of the obtained balance-industrial supplies. For the increase of homogeneity of content of quality, composition of averaging of balance-industrial supplies it is necessary so to plan a booty that the processes of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass flowed in time evenly.

Estimating influence of obstruction of content of quality indexes of minerals in the stream of iron-ore mass on averaging of content of quality indexes of minerals in iron-ore mass assume, that between the average of obstruction averaging and standard deviation there is dependence determine that on results the analysis of current data of ore-mining enterprises of Krivbass during a few years. As evidently from a diagram, what is shown on fig. 5, approximate dependence a line.

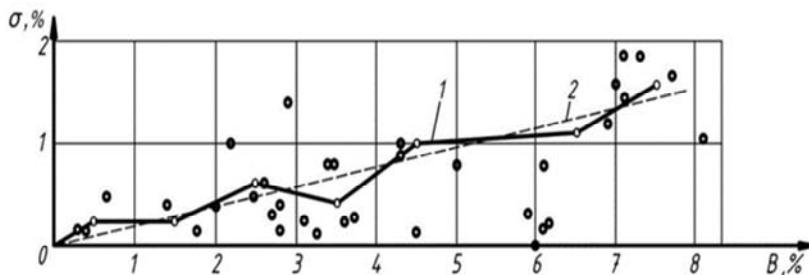


Fig. 5. Dependence of standard deviation of size of obstruction of content of quality indexes of minerals in the stream of iron-ore mass from her mean value: 1 is an empiric curve; 2 is equalization of regression

The results of cross-correlation analysis testify to the presence of connection between sizes: the coefficient of correlation equals 0,7; the error of coefficient of correlation presents 0,04 and testifies to authenticity of the got results and equalization of regression

$$\sigma_B = 0,2\bar{B}. \quad (22)$$

Dependence (22) simplifies the calculations of size and estimation of influence of obstruction of content of quality indexes of minerals on the process of average of content of quality dates of minerals in iron-ore mass, so as an exceptional necessity of realization of experimental works. Explain the presence of such conformity to law that an average is related to the scope of vibrations of obstruction of content of quality indexes of minerals in the stream of iron-ore mass. An increase or reduction of average of obstruction of content of quality dates of minerals in the stream of iron-ore mass is related to the increase or reduction to the range of changeability of obstruction of content of quality indexes of minerals in the stream of iron-ore mass with a size middle quadratic rejection.

Optimization of correlation of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals is in the stream of iron-ore mass. Setting of norms of balance-industrial supplies of hard minerals consists in determination of optimal correlation of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass, that for the case of working contacts longitudinal characterize a parameter [7–10]. On the set technology of mountain works and processing of content of quality indexes of the iron related to magnetite in the stream of iron-ore mass optimal estimation of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass, that certainly by the method of variants of working off contacts or analytical method by being of extremum of technical-economic model of prognostication of the system "a quarry is an ore mining and processing factory". Method of variants universal, but is labour intensive and does not allow exactly to expect optimal correlation, that is why will take advantage of analytical method of optimization singing-relation of losses of balance-industrial supplies and

obstruction content of quality indexes of minerals in the stream of iron-ore mass by determination of a maximum of mathematical technical-economic model of income. Will analyses influence of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass on an income that gets an ore mining and processing combine because of the activity, determine as a difference of cost of concentrate and complete charges on its production.

$$\Pi_P = Q_K y_K - 3_K. \quad (23)$$

The amount of the obtained balance-industrial supplies and concentrate withdrawn from them is bound by dependence [9]

$$Q_K = \mathbb{A} \gamma_K, \quad (24)$$

where γ_K is an exit of concentrate, part of units.

Coming from balance of balance-industrial supplies and content of quality indexes of the iron related to magnetite at enriching and taking into account the mechanical losses of concentrate determine the exit of concentrate from expression

$$\gamma_K = \frac{a - a_x}{(a - a_K)(1 - \pi_K)}, \quad (25)$$

where and is AV content of quality indexes of the iron related to magnetite in the obtained balance-industrial supplies in a plan period; a_K , a_x is AV content of the quality dates iron related to magnetite, accordingly in a concentrate and milltailings in a planning period; π_K are mechanical losses of concentrate in enumeration on a concentrate, parts of units.

Complete charges on the production of concentrate fold from charges on industrially-balance supplies and transporting of the obtained iron-ore mass, on the exception of content of quality indexes of the iron related to magnetite, and charges, on the booty of opening breeds. In addition, an enterprise must bring in paying for the bowels of the earth and make amends for injury to the state through industrially-balance supplies with the use of earth under a quarry (mountain taking), dumps of opening breeds, and industrial

ground. An ore-mining enterprise gets an additional income from realization of breeds of opening and milltailings of content of quality indexes of iron related to magnetite after processing. Thus possible cases:

a) Losses of balance-industrial supplies and obstructions of content of quality indexes of minerals in the stream of iron-ore mass influences on the size of additional income;

б) Losses of balance-industrial supplies and obstructions of content of quality indexes of minerals in the stream of iron-ore mass does not influence on the size of additional income.

Meet the second case more often at the robot of ore-mining enterprises that is part of the first and will consider him at development of technical-economic model of combine. The size of income that gets ore-mining enterprise depends on that, or lose industrially-balance supplies in an array or loosen and take out in a dump. At working off the high-dipping balance-industrial supplies of deposit, bed, ore body or areas of array of hard minerals lose only loosening balance-industrial supplies. At deposits, beds, ore bodies or areas lose balance-industrial supplies both in the loosening state (contacts of hanging side of bed) and in an array (contacts of laying side). For simplification of researches we are work out models for every type of balance-industrial supplies lose that. Taking into account previously mentioned and different character of influence of conditionally changeable and conditionally-permanent charges on the income of enterprise, for a case, when lose loosening balance-industrial supplies, expression (23) will write down as [9]

$$\begin{aligned}
 \Pi_P = & \mathcal{D}\gamma_K (u_K - z'_{TK}) - Q_B z'_B - z''_B - \mathcal{D}z'_{ДТР} - z''_{ДТР} - \mathcal{D}z'_{ПР} - \\
 & - z''_{ПР} - Bm_B - Bm_{ГО} - Q_B (1 - u_B)m_B - \mathcal{D}(1 - \gamma_K)(1 - u_X)m_X - \\
 & - T_{III} + Q_B u_B [\gamma_{TB} (u_{TB} - z'_{ТТБ}) - (z'_{ТТБ} - z'_{ТБ}) - z'_{ПБ} - \\
 & - z'_{ТХБ} (1 - \gamma_{TB}) - (1 - \gamma_{TB})m_{ХБ}] - z''_{ТБ} + \\
 & + \mathcal{D}u_X (1 - \gamma_K) [\gamma_{TX} (u_{TX} - z'_{ТТХ}) - (z'_{ТТХ} - z'_{ТХ}) - z'_{ПХ} - \\
 & - z'_{ТХХ} (1 - \gamma_{TX}) - (1 - \gamma_{TX})m_{ХХ}] - z''_{ТХ}, \quad (26)
 \end{aligned}$$

where $\gamma_K, \gamma_{TB}, \gamma_{TX}$ is an exit of concentrate, commodity products accordingly from the breeds of opening of balance-industrial supplies and milltailings of content of quality indexes of the iron related to magnetite, part of units; u_B, u_X is a coefficient of the use (processing) of

breeds of opening and tails of enrichment content of quality indexes of the iron related to magnetite, part of units; u_K, u_{TB}, u_{TX} is a cost of 1 τ concentrate, commodity products, that is got from the breeds of opening and milltailings of content of quality indexes of the iron, related to magnetite, hrn.; z'_{TK} are conditionally-changeable charges on to the transportation concentrate from an ore mining and processing factory to the point, a consumer, hrn./of τ , bears expenses from that; z' In are conditionally-changeable charges on a production z'_{DB} and transporting in mine dump z'_{TB} , taken on 1 τ of breeds of opening, hrn.; $z'_{ДП}$ and z'_{IP} are charges accordingly on the booty of balance-industrial supplies and transporting to the ore mining and processing factory and primary exception of content of quality indexes of the iron, related to magnetite in the stream of iron-ore mass and on a production and transporting in the dump of breeds of opening, thousand a hrn.; T_B, T_{TB} is paying accordingly for the bowels of the earth and earth of the mountain taking (on 1 τ of balance-industrial supplies), hrn.; T_B is paying for earth under the dumps of opening breeds, what relate to 1 τ of the geological opening, hrn.; T_{XX} is paying for earth under tailing dump, take that to 1 τ milltailings of content of quality indexes of the iron, related to magnetite, hrn.; T_{III} is paying for earth under industrial site, thousand a hrn.; z'_{TB}, z'_{TX} are conditionally-changeable charges on transporting of 1 τ of commodity products, that is got from the breeds of opening and tails of content of quality indexes of the iron related to magnetite, from the places of their processing to the railway, hrn.; z'_{TBI}, z'_{IB} are conditionally-changeable charges on transporting from a quarry to the place of processing and exception of 1 τ breeds of opening, hrn.; z'_{TII}, z'_{IX} are conditionally-changeable charges on transporting from an ore mining and processing factory to places of processing of milltailings and on the exception of 1 τ milltailings of content of quality indexes of the iron, related to magnetite, hrn.; z'_{TXB}, z'_{TXX} are conditionally-changeable charges on transporting of 1 τ tails (wastes) of processing of breeds of opening and milltailings of content of quality indexes of the iron related to magnetite from the places of processing, hrn.; T_{XB}, T_{XX} is paying for the use of earth under tailing dumps wastes of processing of breeds of opening and milltailings of content of quality indexes of the iron related to magnetite, that is taken on 1 τ , hrn.; z''_{TB}, z''_{TX} is a sum of

conditionally-permanent charges on the exception of breeds of opening and milltailings, thousand hrywnas.

Expression (26) after transformation and introduction of conditional denotations will write down as

$$\Pi_P = \mathcal{D}[\gamma_K A_1 - A_2 + \Pi p_X] - Q_B(A_3 - \Pi p_B) - B A_4 - 3''_K - T_{III}, \quad (27)$$

$$\text{Where} \quad A_1 = u_K - 3'_{TK}, \quad (28)$$

$$A_2 = 3'_{ДТР} + 3'_{\Pi P} + (1 - \gamma_K)(1 - u_X)m_X, \quad (29)$$

$$A_3 = 3'_B (1 - u_B)m_B, \quad (30)$$

$$A_4 = m_B + m_{ГО}, \quad (31)$$

Πp_B is an income from realization of opening breeds; take that to their 1 T, hrn.

$$\begin{aligned} \Pi p_B = & u_B [\gamma_{TB} (u_{TB} - 3'_{ТТБ}) - (3'_{\Pi\Pi B} - 3'_{TB}) - \\ & - 3'_{\Pi B} - (1 - \gamma_{TB})(3'_{TBX} - m_{XB})] - 3''_{BB}; \end{aligned} \quad (32)$$

Πp_X is an additional income from realization of milltailings of content of quality indexes of the iron, related to magnetite, take that to 1 T of iron-ore mass, hrn.

$$\begin{aligned} \Pi p_X = & u_X (1 - \gamma_K) [\gamma_{TX} (u_{TX} - 3'_{ТХХ}) - (3'_{ТХ\Pi} - 3'_{TX}) - \\ & - 3'_{\Pi X} - (1 - \gamma_{TX})(3'_{ТХХ} - m_X)] - 3''_{TX}; \end{aligned} \quad (33)$$

$3''_K$ is a sum of conditionally-permanent charges at the production of concentrate, hrn.;

$$3''_K = 3''_B + 3''_{ДТР} + 3''_{\Pi}. \quad (34)$$

Optimization of balance-industrial supplies of hard minerals mathematically and logically legitimate only at complete them contoured, at $B=of\ const$, and the known quantitative, quality and technological descriptions. Because of it will write down expression (27) as a function of B ; Π ; B . In that case, when lose balance-industrial supplies in the loosening state, the amount of breeds of

opening that must be extracted at working off balance-industrial supplies determines as

$$Q_B = BK_{BF} + \Pi - B. \quad (35)$$

If we lose balance-industrial supplies in an array

$$Q_B = BK_{BF}. \quad (36)$$

Putting expressions (23), (34), (35) in (27) and taking into account the different exit of concentrate in accordance with (25), will get two cases, when lose balance-industrial supplies:

a) in the iron-ore mass removed from an array

$$\begin{aligned} \Pi p_0 = & B[\gamma_B A_1 - A_2 - K_{BF}(A_3 - \Pi p_B)] - A_4 + \Pi p_X] - \\ & - \Pi[\gamma_{II} A_1 - A_2 + A_3 - \Pi p_B - \Pi p_X] + B[\gamma_B A_1 - A_2 + \\ & + A_3 - \Pi p_B + \Pi p_X] + 3_K'' - T_{III} \end{aligned} \quad (37)$$

б) in the array of hard minerals

$$\begin{aligned} \Pi p_M = & B[\gamma_B A_1 - A_2 - K_{BF}(A_3 - \Pi p_B)] - A_4 + \Pi p_X] - \\ & - \Pi[\gamma_{II} A_1 - A_2 + \Pi p_X] + B[\gamma_B A_1 - A_2 + \Pi p_X] - 3_K'' - T_{III}. \end{aligned} \quad (38)$$

In careers is both connected at working off contacts losses balance-industrial supplies and obstruction to content quality indexes minerals in a stream iron-ore mass and independent from each other (flying away of pieces of iron-ore mass and breeds, cleaning out of ballast). The last are caused by technological reasons and are the proportionally liquidated balance-industrial supplies. At working off contacts determine dependence between the careening bone of losses of balance-industrial supplies and obstructive content of quality indexes of minerals of breeds the chart of working off contacts. Charts of working off working over off contacts brought to the chart longitudinal with flat hay-crops after correlation amounts of balance-industrial supplies, lose that obstructive content of quality indexes of minerals of breeds.

Thus, amount of balance-industrial supplies lose that and obstructive content of quality indexes of minerals of breeds in a career determine as

$$\Pi = \Pi_{\Delta} + \Pi_T = V_{\Delta} \rho_{\Pi} k^2 + \pi_T B = B_{\Delta} k^2 + \pi_T B, \quad (39)$$

$$B = B_{\Delta} + B_T = V_{\Delta} \rho_B (1 - k)^2 + \beta_T B = B_{\Delta} (1 - k)^2 + \beta_T B, \quad (40)$$

where Π_{Δ} , B_{Δ} is an amount accordingly balance-industrial supplies, lose that breeds of obstruction of content of quality indexes of minerals in "triangles" at working off near contact zones, thousands of τ ; Π_T , B is an amount of balance-industrial supplies and breeds of obstruction of content of quality indexes of minerals, lose that at cleaning out of working grounds of ledges (through technological reasons), thousands of τ ; V_{Δ} is a volume of near-contact zone, thousand M^3 ; ρ_{Π} , ρ_B is a middle closeness accordingly balance-industrial supplies lose that content of quality indexes of minerals of breeds τ/M^3 .

Determination of maximum for expressions (37) and (38), will put for them (39) and (40) and, taking into account, that $B = \text{of const}$, will find derivatives for k at working off a contact longitudinal split with flat hay-crops, will equate the last with a zero and will untie the got expressions relatively k . Value of k at working off a contact longitudinal split with flat hay-crops, at that objective functions (37) and (38) have at most, determine from expressions

$$k_{1_0} = \frac{A_2 - A_3 + \Pi p_B - \Pi p_x - A_1 \gamma_B}{A_1 \left(\frac{\rho_{\Pi}}{\rho_B} \gamma_{\Pi} - \gamma_B \right) - (A_2 - \Pi p_x) \left(\frac{\rho_{\Pi}}{\rho_B} - 1 \right)}, \quad (41)$$

$$k_{1_M} = \frac{A_2 - \Pi p_x - A_1 \gamma_B}{A_1 \left(\frac{\rho_{\Pi}}{\rho_B} \gamma_{\Pi} - \gamma_B \right) - (A_2 - \Pi p_x) \left(\frac{\rho_{\Pi}}{\rho_B} - 1 \right)}. \quad (42)$$

Value of symbols in expressions (41) and (42) answers denotation in formulas (28)–(33), and γ_{Π} and γ_B determine from expression (25). After formulas (41) and (42) determine to the normative determination parameter for the case of working off the contacts of spilt and correlation of amount and volume of balance-industrial supplies, lose that and breeds of obstruction of content of quality indexes of minerals at working off contacts with different content of quality indexes of the iron related to magnetite, in balance-industrial supplies, lose that and obstruction content of quality indexes of

minerals breeds in the case when ore mining and processing a factory can do all obtained balance-industrial supplies that is got at correlation of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass, corresponding normative value parameter for case acquisition contact longitudinal splits.

Calculations of normative values of parameter for the case of working off contacts longitudinal splits from expressions (41) and (42) simplify in comparing to the method of variants and continuous perfection of methodology of setting of norms of balance-industrial supplies of hard minerals. Determine AV content of quality indexes of the iron related to magnetite, in balance-industrial supplies infinitely thin layer on the hay-crop of ledge at working off a contact longitudinal split with flat hay-crops determine

$$a_r = \frac{I_{\Pi} \text{drl } \rho_{\Pi} c_{\Pi} + I_B \text{drl } \rho_B b}{I_{\Pi} \text{drl } \rho_{\Pi} + I_B \text{drl } \rho_B}. \quad (43)$$

Determine I_{Π} and I_B through the height of ledge and parameter at working off a contact longitudinal split with flat hay-crops, will get

$$a_r = \frac{k \left(\frac{\rho_{\Pi}}{\rho_B} c_{\Pi} - b \right) + b}{k \left(\frac{\rho_{\Pi}}{\rho_B} - 1 \right) + 1}. \quad (44)$$

From expression (44) determine the value of k , at that balance-industrial supplies infinitely thin layer on the hay-crop of ledge have the known content of quality indexes of the iron related to magnetite

$$k = \frac{a_r - b}{\frac{\rho_{\Pi}}{\rho_B} (c_{\Pi} - a_r) + a_r - b}. \quad (45)$$

Determine a_r , at that an ore mining and processing combine will get an income as a result of working off balance-industrial supplies, content of quality indexes useful to the component "bracks" and will

designate a_{δ} , for that a_{δ} will equate expression (45) in turn with expressions (41) and (42). Untiing got to equality, will find:

$$a_{\delta_0} = \frac{A_1(b\gamma_{II} - c_{II}\gamma_B) + (A_2 - A_3 + \Pi p_B - \Pi p_X)(c_{II} - b)}{A_1(\gamma_{II} - \gamma_B)}; \quad (46)$$

b) for the case of losses of balance-industrial supplies in an array

$$a_{\delta_M} = \frac{A_1(b\gamma_{II} - c_{II}\gamma_B) + (A_2 - \Pi p_X)(c_{II} - b)}{A_1(\gamma_{II} - \gamma_B)}. \quad (47)$$

Putting in expressions (46) (47) exit of content of quality indexes of the iron, related to magnetite in a concentrate from balance-industrial supplies lose and obstruction content of quality indexes of minerals of breeds (25), in that a will accept according to such, that equals c_{II} and b , and executing transformation, will get the value of "brack" contents of quality indexes:

a) for the case of losses of balance-industrial supplies in loosening iron-ore mass

$$a_{\delta_0} = a_X + \frac{A_2 - A_3 + \Pi p_B - \Pi p_X}{A_1} (a_K - a_X)(1 + \pi_K); \quad (48)$$

б) for the case of losses of balance-industrial supplies in an array

$$a_{\delta_M} = a_X + \frac{A_2 - \Pi p_X}{A_1} (a_K - a_X)(1 + \pi_K). \quad (49)$$

Value of symbols in expressions (44)–(49) answers expressions (28)–(33). Official [1,3–5] and driven to technical literature [2,6,8,9] methodologies of setting of norms of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals do not take into account a loss, that inflicts to the national economy an enterprise from an exception from earth and use of breeds of opening

and milltailings of content of quality indexes of the iron related to magnetite, that it is taken into account by the worked out methodology. Without harm from the exception of earth from and use of breeds of opening and milltailings of content of quality indexes of the iron related to magnetite expressions for brack contents of quality indexes simplification and accept:

a) for the case of losses of balance-industrial supplies in loosening iron-ore mass

$$a_{\sigma_0} = a_X + \frac{3'_{ДТП} + 3'_{ПР} - 3'_B}{u_K - 3'_{ТК}} (a_K - a_X)(1 + \pi_K); \quad (50)$$

b) for the case of losses of balance-industrial supplies in an array

$$a_{\sigma_M} = a_X + \frac{3'_{ДТП} + 3'_{ПР}}{u_K - 3'_{ТК}} (a_K - a_X)(1 + \pi_K). \quad (51)$$

Analysis of formulas (46)–(51) certifies that content of "brack" of quality indexes useful to the component takes into account the economy of enterprise and technology of exception of content of quality indexes of the iron related to magnetite in the stream of iron-ore mass only. Normative value of parameter of $k=k_H$ determine from expression (44), that after the substitution of $a_r=a_{\sigma}$ will write down as

$$k_H = \frac{a_{\sigma} - b}{\frac{\rho_{II}}{\rho_B} (c_{II} - a_{\sigma}) + a_{\sigma} - b}. \quad (52)$$

Determine normative correlation of amount of losses of balance-industrial supplies and obstruction content of quality indexes of minerals of breeds after a substitution in expression (52) as

$$\omega_H = \frac{B}{II} = \frac{\rho_{II}}{\rho_B} \left(\frac{c_{II} - a_{\sigma}}{a_{\sigma} - b} \right)^2. \quad (53)$$

During work of ore-mining enterprises there are cases, when a production capacity of ore mining and processing factory is on the exception of content of quality indexes of the iron, related to magnetite in the stream of iron-ore mass less than, than production capacity of quarry is on the booty of balance-industrial supplies of hard minerals. Such case is considered in-process [2] and "limitation of the second kind" is adopted. We are work out methodology of setting of norms of balance-industrial supplies of hard minerals at "limitation of the second kind" in relation to terms, when the use of opening breed and milltailings of content of quality indexes of the iron related to magnetite.

For the receipt of decision determine influence of loss from the exception of earth from. At "limitation of the second kind" a permanent size is an amount of the obtained balance-industrial supplies of $\bar{M}=\bar{M}_\phi=const$, where \bar{M}_ϕ is a production capacity of ore mining and processing factory on the exception of content of quality indexes of the iron related to magnetite in the stream of iron-ore mass. An amount of balance-industrial supplies of hard minerals, pay off that, is a size changeable.

Thus, at "limitation of the second type" of task of setting of norms of balance-industrial supplies of hard minerals it is set forth so: to find such amount of balance-industrial supplies of hard minerals, pay off that and norms of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass, at that a combine will get a maximal income, and the amount of the shipped iron-ore mass equals the production capacity of ore mining and processing factory on the exception of content of quality indexes of the iron related to.

So as at "limitation of the second kind" balance-industrial supplies of hard minerals not certainly, then, decide a task the method of progressive approximations, or analytically, at that the amount of contacts is proportional to the amount of balance-industrial supplies of hard minerals, pay off that. Legitimacy of this assumption is confirmed by experience of quarries of Krivbass and setting of norms of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass: the relative (actual and normative) losses of balance-industrial supplies and

obstructions of content of quality indexes of minerals have insignificant vibrations in years.

Doing foregoing assumption, determine the optimal value of parameter for the case of working off contacts longitudinal splits $k=k_{II_0}$ for "limitation of the second kind" in the case when lose balance-industrial supplies in loosening iron-ore mass. For reduction of record in expression (37) will designate

$$A_B = \gamma_B A_1 - A_2 - K_{BI}(A_3 - \Pi p_B) - A_4 + \Pi p_X, \quad (54)$$

$$A_{II} = \gamma_{II} A_1 - A_2 + A_3 - \Pi p_B + \Pi p_X, \quad (55)$$

$$A_B = \gamma_B A_1 - A_2 + A_3 - \Pi p_B + \Pi p_X; \quad (56)$$

Then will write down as

$$\Pi p_0 = B A_B - \Pi A_{II} + B A_B - 3_K "T_{III}. \quad (57)$$

Putting in (57) expressions (39),(40) and

$$B = \Pi + \Pi - B, \quad (58)$$

Determine a value at most the got expression on condition of $B=of\ const.$ Value to the parameter for the case of working off contacts longitudinal splits $k=k_{II}$, at that this expression has at most, determine as

$$k_{II} = \frac{\sqrt{\frac{\rho_{II}(A_B - A_{II})}{\rho_B(A_B - A_B)} - \left[\frac{\rho_{II}(A_B - A_{II})}{\rho_B(A_B - A_B)} - 1 \right] \times \frac{\pi_m(A_B - A_{II}) - \beta_m(A_B - A_B)}{\beta_\Delta(A_B - A_B)} - 1}}{\frac{\rho_{II}(A_B - A_{II})}{\rho_B(A_B - A_B)} - 1} \quad (59)$$

Taking into account the exit of concentrate (25) from the stream of iron-ore mass of balance-industrial supplies of hard minerals, pay off that and lose and obstructive content of quality indexes of minerals of breeds will have

$$A_B - A_{II} = \frac{(c - c_{II})A_1 - [(A_3 - \Pi p_B)(K_{BF} + 1) + A_4](a_K - a_X)(1 + \pi_K)}{(a_K - a_X)(1 + \pi_K)}. \quad (60)$$

$$A_B - A_B = \frac{(c - b)A_1 - [(A_3 - \Pi p_B)(K_{BF} + 1) + A_4](a_K - a_X)(1 + \pi_K)}{(a_K - a_X)(1 + \pi_K)}. \quad (61)$$

If lose industrially-balance supplies in the bowels of the earth, in expression (59) will accept [got taking into account expression (38)]

$$A_B - A_{II} = \frac{(c - c_{II})A_1 - [K_{BF}(A_3 - \Pi p_B) + A_4](a_K - a_X)(1 + \pi_K)}{(a_K - a_X)(1 + \pi_K)}. \quad (62)$$

$$A_B - A_B = \frac{(c - b)A_1 - [K_{BF}(A_3 - \Pi p_B) + A_4](a_K - a_X)(1 + \pi_K)}{(a_K - a_X)(1 + \pi_K)}. \quad (63)$$

The amount of balance-industrial supplies pay off that determines from expression (58), after a substitution for him expressions (39), (40) $k=k_{II}$ and $\mathcal{A}=\mathcal{A}_\phi$.

$$B = \frac{\mathcal{A}_\phi}{1 - \pi_\Delta k_{II}^2 + \beta_\Delta (1 - k_{II})^2 - \pi_T + \beta_T}, \quad (64)$$

where \mathcal{A}_ϕ is a production capacity of ore mining and processing factory from the exception of content of quality indexes of the iron, related to magnetite in the stream of iron-ore mass, thousand t; β_Δ , π_Δ it is a relation of amount of balance-industrial supplies and breeds of obstruction of content of quality indexes of minerals in near contact zones to the balance-industrial supplies of hard minerals, pay off that, part of unit

$$\pi_\Delta = V_\Delta \rho_B / \tilde{B}, \quad (65)$$

$$\beta_\Delta = V_\Delta \rho_B / \tilde{B}, \quad (66)$$

\tilde{B} is an amount of balance-industrial supplies of hard minerals, pay off that, in the initial (before optimization) variant of working off, thousand t; V_Δ is a volume of near contact zones in the initial variant

of working off balance-industrial supplies, thousands of M^3 ; ρ_B, ρ_B is a middle closeness of balance-industrial supplies and breeds of obstruction of content of quality indexes of minerals, τ/M^3 ; π_T, β_T is relative losses of balance-industrial supplies and bringing in of unstandard supplies on technological reasons, parts of units.

Analyzing principles of setting of norms will notice that at "limitation of the first kind" ($B=const, D=var$) determine the losses of balance-industrial supplies and obstructions of content of quality indexes of minerals during work with that an enterprise gets a maximal income as a result of working off the balance-industrial supplies of hard minerals, and at "limitation of the second kind" ($D=const, B=var$) the requirement of receipt of maximal income a mining ore enterprise is kept, but on condition of exception of content of quality indexes of the iron related to magnetite in the stream of iron-ore. Calculations, optimal value of parameter for the case of working off contacts longitudinal splits of the second kind" always more than first. A sum of income at "limitation of the second kind" will be both less and more than at "limitation of the first kind", but income, that it is got on 1 τ of balance-industrial supplies pay off that, always less than, than at "limitation of the first kind".

Thus, at "limitation of the second kind" at most will get an income due to the increase of losses of balance-industrial supplies and reduction of obstruction to content of quality indexes of minerals, id est due to unusing of balance-industrial supplies. As researches of setting of norms of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals showed for the terms of iron-ore quarries of Krivbass in the case when sum of income at "limitation of the second kind" less than, than at "limitation of the first and at certain correlation of production capacities of quarry and ore mining and processing factory, there is possibility to get a greater income, than at "limitation of the second kind" and this is "limitation of the third kind" [2,6,10,12], what responds to the condition, that the amount of content of quality indexes of minerals in the stream of iron-ore mass that is got at working off the maximal amount of balance-industrial supplies of hard minerals with the certain size of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass equals the production capacity of ore mining and processing factory on the exception of content of quality indexes of the

iron related to. At "limitation of the third kind" the optimal losses of balance-industrial supplies and obstructions of content of quality indexes of minerals in the stream of iron-ore mass determine from the mining-and-geological and technological terms of booty of balance-industrial supplies of hard minerals and exception of content of quality indexes of the iron related to magnetite.

Optimal value of parameter for the case of working off contacts longitudinal splits $k=k_{III}$ at "limitation of the third kind" determine if in expression (23) will put (39), (40) and will untie relatively for the case of working off contacts longitudinal splits k

$$k_{III} = \frac{\sqrt{\frac{\rho_{II} - \left(\frac{\rho_{II} - 1}{\rho_B}\right) \frac{B(1 - \pi_T - \beta_T) - \Delta \Phi}{V_{\Delta} \rho_B} - 1}{\rho_B} - 1}}{\frac{\rho_{II} - 1}{\rho_B}} \quad (67)$$

The got results testify that setting of norms of balance-industrial supplies after the degree of preparedness to the booty and realization of norms in a production is an effective measure in area of guard of bowels of the earth, that will allow to decrease the losses of balance-industrial supplies and obstructions of content of quality indexes of minerals in iron-ore mass.

Implications

1. Optimization of balance-industrial supplies after the degree of preparedness to the booty together with the system of averaging out of content of quality indexes of minerals in the stream of iron-ore mass carry out separately for every category of balance-industrial supplies of hard minerals and for all categories simultaneously. The relative (actual and normative) losses of balance-industrial supplies and obstructions of content of quality indexes of minerals have insignificant vibrations in years.

2. Worked out methodology of setting of norms of balance-industrial supplies after the degree of preparedness to the booty, norms of the industrial supplies prepared to the booty on the careers of Krivbass determine taking into account the losses of balance-industrial supplies and obstruction of content of quality indexes of

minerals in the stream of iron-ore mass at a booty. Methodology of setting of norms of preparedness of the balance-industrial supplies prepared to the booty is modernized and adjusted to the use on the stage of planning. Indexes and coefficients characterize correlation of separate constituents of balance-industrial supplies, type of transporting of iron-ore mass and unevenness of extractive works.

3. The set dependences specify on the existent mining, technological, organizational and other terms of mining from ore bodies or areas of iron-ore deposits and show that passing of ore mining enterprises to work with normative preparedness of balance-industrial supplies to the booty will allow on iron-ore careers to bring down the obstruction of content of quality indexes of minerals in the stream of iron-ore mass on 1,5–2.%.

4. Correlation of amount and volume of balance-industrial supplies, lose that and breeds of obstruction of content of quality indexes of minerals at working off contacts with different content of quality indexes of the iron related to magnetite, in balance-industrial supplies, lose that and obstructive content of quality indexes of minerals breeds in the case when an ore mining and processing factory can do all obtained balance-industrial supplies that is got at correlation of losses of balance-industrial supplies and obstruction of content of quality indexes of minerals in the stream of iron-ore mass, corresponding to the normative value of parameter for the case of working off contacts longitudinal splits.

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INNOVATIVE TECHNOLOGY OF MANUFACTURING CHARGES FOR SPLITTING OF BLOCK STONE AT PLACES OF BLASTING WORKS

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Abstract

The question of decorative block stone mining is considered. Decorative stone is widely used in various branches of the national economy - architecture, construction, and artistic masonry. There are about 300 deposits of ornamental stone in Ukraine. The main condition for its extraction is the reflection with whole blocks for further processing.

It is substantiated that there are currently a number of explosive and mechanical methods based on the application of various types of equipment to reflect the block stone. In Ukraine, in order to reflect the block stone, mainly smoky gunpowder, detonating cord and imported charges are used. It is revealed that the smallest number and length of cracks in the granite is formed during the work on the extraction of block stone using the Finnish charges such as K-tubes.

The authors developed elongated explosive charges of K-tubes type of local production from low-energy explosives, that are safe application techniques and the environment and obtain characteristics such as volume of gases in the amount of 310 l/kg, caloricity 2165.8 kJ/kg and low detonation speed, that provide soft reflection of block stone. The charges are made directly at the site of blasting operations. The technology of making charges at places of realization of explosive works with the help of a manual screw mixer MUNSCH MAK-32 is worked out. In order to verify the quality of explosive mixture production in the field of blasting operations, the detonation speed by the Dotresh method was determined. The detonation speed of the domestic K-tubes is 1830 m/s, the sensitivity to impact on the device 1 is 36-41%, and the lower threshold of frictional sensitivity for shock-shifting on the K-44-3 device is 3650-3740 kg/cm².

The distribution of the combustible component along the height of the charge of the K-tubes was determined by infrared spectroscopy based on Fourier transform (ISFT). It is determined that the distribution of components is uniform, and the domestic method of making charges in the field of blasting exceeds the safety characteristics of the application of the Finnish analogue.

Key words: decorative stone, elongate charges, explosive mixture, technology, safety, infrared spectroscopy.

Introduction. In Ukraine, about 300 deposits of facing (decorative) stone have been explored, which are represented mainly by high-strength rocks: granites, labradorites, andesites, gabbros, and others. [1]. Among the well-known quarries are: Limited Liability Company Malinovsky granite quarry (Vinnytska region), Open Joint Stock Company Novopavlovsk granite quarry (Dnipropetrovsk region), Open Joint Stock Company Starokrymsky quarry (Donetska region), Limited Liability Company Emelyanivsky quarry (Zhytomyr region), Open Joint Stock Company Mukachevo quarry (Transcarpathian region), Private Joint-Stock Company Zaporozhne Generation Plant (Zaporozhye region), Ivano-Frankivsk Special Park (Ivano-Frankivsk region), Shamraev granite quarry (Kyiv region), Vlasovsky quarry (Kirovograd region), Open Joint Stock Company Uspensky quarry (Luhansk region) and others. At present, the productivity of each of the quarries is at least 1000 m³ per year; reserves of granite, for example, in the quarries of Zhytomyr region make an average of about 3.5 million m³ in an array.

Decorative stone is widely used in various branches of the national economy - architecture, construction, and artistic masonry. Unlike deposits of other minerals (iron ores, coal, building rocks), the methods of extraction and processing for facing stone deposits should provide:

- preservation of the stone's properties;
- the maximum possible output of the developed blocks of minerals;
- preservation of the solidity of the array of minerals;
- reduction of stone's expenses at its extraction;
- the maximum possible complex use of raw materials.

At present, there are a number of explosive and mechanical methods based on the application of various types of equipment to

reflect the block stone. Classification of the methods of separation of monoliths extracted from the array and their subsequent processing in the conditions of the mining enterprise is very diverse [2]. In this case, there are general classifications that indicate or streamline all existing methods and their combinations, or a particular method or type of natural stone. Classifications of general character use drilling, cutting, shredding and explosion as the main ways of the block stone breaking during its extraction. Along with this there are classifications that streamline known methods of action on an array of rocks, dividing them into instantaneous, dynamic or static [3].

Despite the large loss of valuable raw materials in mining, explosive technologies and stone reflection tools continue to be used and improved throughout the world because of their high productivity [4].

Along with this the dusty methods of mining monoliths find a mass application. In a large number of stone quarries throughout the world, wedge methods continue to be used alone or in combination with other known mechanical methods [5].

Currently, explosive, chemical and physico-chemical methods of block stone mining are mainly used.

At present, in Ukraine, to reflect the block stone, mainly smoky gunpowder, which is used in 11% of Ukrainian quarries, detonating cord and import charges are used.

Smoke gunpowder has a low detonation speed of about 500 m/s at a density of 1.0-1.5 g/cm³. It can be said that its explosive transformation takes place in deflagration mode [6]. The volume of gaseous products of the explosion of smoky powder is on average about 280 l/kg, caloric value 720 kcal/kg.

Smoke gunpowder has a number of positive properties:

- low rate of explosive combustion (300 - 600 m/s), which has a longer impact on the array than with any other explosives with higher detonation velocities;

- low peak pressure during explosive combustion, which minimizes the destruction of an array in the area of local explosion.

But smoky powder has a high sensitivity to mechanical influences, as well as high hygroscopicity. Although smoky gunpowder has a flash point of 290-310 °C (563.15 - 583.15 K), it easily explodes when a spark comes from a stone's stroke, a metal on

metal, which makes it charging in a hole very dangerous. High hygroscopicity leads in misty or rainy weather to refuse of its ignition, and instability of its properties during explosive combustion. Large linear mass of smoky gunpowder per running meter of a hole, high risk of exploitation, harmful working conditions and low production culture make it impossible to further use of smoky gunpowder.

Properties of block decorative stone and requirements to the methods of its extraction. Facing stone is a decorative facing material used for processing facades of buildings. It can be as a natural origin – a natural stone, and industrial production – an artificial stone.

An artificial decorative stone can imitate almost any natural material, while it is much easier, more convenient in the installation and is better adapted to any architectural and construction task. One type of artificial cladding stone is the so-called "white stone", that is made on the basis of concrete mixes with internal spatial reinforcement.

In Ukraine, about 300 deposits of facing (decorative) stone are known, which are represented mainly by high-strength rocks, such as granites, labradorites, andesites, gabbros and others [7]. Decorative stone is widely used in various branches of the national economy – architecture, construction, and artistic masonry. Facing stone is used both for the external processing of the facades and pedestals of buildings, and for the interior design of walls, columns, fireplaces and other interior details.

Unlike deposits of other minerals (iron ores, coal, building rocks), the methods of extraction and processing for facing stone deposits should provide [8]:

- preservation of the stone's properties;
- the maximum possible output of the developed blocks of minerals;
- preservation of the solidity of the array of minerals;
- reduction of stone's expenses at its extraction;
- the maximum possible complex use of raw materials.

Modern explosive charges of foreign and domestic production. Finnish scientists conducted the tests, during which they compared the cracks formed around the hole in the extraction of block stone

using K-tubes, black powder and some other explosive of Finnish manufacture, as well as detonation cord with a weight of explosive like 40 g/m. It was found that the smallest number and length of cracks in the granite is formed during the work on the extraction of a block stone using K-tubes (the destruction of the stone is caused by the explosion and the appearance of cracks only by 2 cm thick block, in contrast to the detonation cord with a weight of explosive like 40 g/m, with the use of which were found single cracks up to 65 cm). The presence of cracks was similarly observed when using other explosives during the extraction of a block stone, although of a shorter length than in the case of the detonation cord use with a weight of explosive like 40 g/m.

Abroad, in all modern stone quarries, a soft explosion technique using special plastic charges such as plastic charging tubes (K-charges) of the Finnish firm "Forsit" is used [7]. The explosive composition of the K-charges of the Finnish firm "Forsit" for the block stone reflection has a detonation speed of 1.8 - 2.1 km/s with a critical diameter of ~ 17 mm and a density of 0.95 to 1.05 g/cm³. In this case, the volume of gases released during the decomposition of explosive is up to 160 l/kg. The caloric content of this explosive composition is 358 kcal/kg.

Blast mixture of tubular K-charge contains nitroesters (a mixture of nitroglycerin and nitroglycol) and silica. Nitroesters cause headache when exposed to human skin or from the effects of air polluted by them. This requires the use of personal protective equipment (rubber gloves, respirators, and like that). In addition, the explosive mixture of K-charges has insufficient water resistance, which prevents their operation in flooded hollows.

The advantages of K-charges are [9]:

- simplicity of charging;
- absence of explosive mixture's placement, which increase the safety of work on a career;
- simplicity of regulation of the amount of explosive mixture in the hollow;
- the presence of an air layer between the charge and the walls of the hole, that in turn reduces both the pressure peak, which is created during detonation and the amount and size of the cracks in the separating block;

- the use of detonation cord makes it possible to initiate K-charges in the hole, practically simultaneously, thus creating load levels along the entire length of the hole;
- reduction of the emission of harmful substances into the atmosphere, as a result of a smaller amount of allocated gases (160 l/kg), even compared with the black powder (280 l/kg);
- reduction of explosive mixture's expenses in comparison with the smoke gunpowder.

The charges of the same type are developed in Sweden ("Gurit-A"). However, they have a somewhat high detonation speed (4.0 km/s), which prevents a soft section of the block from the array [7]. In Russia, in 1993, the Leningrad Mining Institute, in collaboration with Special Design and Technology Bureau "Technologist", created for block quarries the tubular charges of "Granilin", which are analogous to the abovementioned foreign plastic charges [10]. Low speed of detonation at high density, low sensitivity to mechanical influences, manufacturability when charging holes, safety during transportation and use ensure the prospect of charges "Granilin".

In Ukraine, the mining company "Lagran" offers the non-explosive technology for block stone extracting in quarries using the chemical pressure generator (CPG) "LITOCOL". CPG is intended for use in mining for the separation of rocks (granite, labradorite, gabbro, etc.) from an array, the separation of concrete blocks in construction and reconstruction [11].

The disadvantages of these gas generators are that they are not well-studied, namely the effect of the combustion rate of the gas generator on the quality of the breakdown of the block, factors affecting the combustion rate of the mixture (chemical additives, composition density, component shredding, dimensions of the cartridge case, preparation formulation). Also, the disadvantages include hygroscopicity, sensitivity to various actions, as well as the formation of toxic gases in the combustion of the main component of the gas-generating mixture, which complicates the process of manufacturing, transportation and proper application of these cartridges.

Based on the advantages and experiments conducted by foreign scientists, as well as the increased demand for block stone products, the development of the design of prolonged charges of domestic

production (such as K-tubes) is an urgent task. However, the transport of explosive charges from the manufacturer to the site of blasting requires the availability of a permit for transportation and special transport, which complicates the use of this technology.

Since there are no such charges of local production in Ukraine, the task is to create own safe and low-energy explosives for the use of charges of the type like K-tubes in the place of blasting operations and, accordingly, a qualitative, safe and economically competitive means of a block stone reflecting.

The purpose of the research: development of the technology of domestic extended-charge charges of the type like K-tubes of local production is an actual scientific and practical task.

Research methods: determination of the efficiency of the explosive mixture on the basis of potassium perchlorate, proving the homogeneity of the explosive mixture in the tube by the method of infrared spectroscopy.

Investigation of domestic elongated charges

In Ukraine, charges of the type like K-tubes of local production with a safe technique of application and ecology of low-energy explosives are developed. The charges are made directly at the site of blasting operations [12-14].

The appearance of developed elongated charges is presented in Fig. 1.

The method of explosive mixtures manufacturing from non-explosive materials for reflecting of a block decorative stone by mixing potassium perchlorate with catalysts of decomposition such as metal oxides, and diesel fuel (nitromethane), which is used as a fuel [15, 16]. Experimental samples of low-speed explosive mixtures were made for the purpose of working out technology and studying their explosive characteristics.

For research the following components were used:

- potassium perchlorate (potassium hlornokyslyy) TC 6-09-3801-76;
- manganese (IV) oxide GOST 4470-79;
- diesel fuel GOST 305-82.
- nitromethane TC 6-09-11-876-77.



Fig. 1. Domestic elongated charges of the type like K-tubes for splitting of a block stone

These mixtures should be filled into plastic tubes with an internal diameter of up to 27 mm, based on the critical diameter of the detonation of this composition, and accordingly, such charges may be placed in bore holes of 36-42 mm in diameter, which are most used in Ukraine when the block stone is reflected.

Physico-chemical and explosive characteristics of the developed explosive composition are summarized in Table 1.

The technological scheme of explosive mixtures manufacturing consisted of four main stages:

- preparation of components: drying, grinding, sifting;
- mixing of components;
- manufacture of polymeric membranes;
- manufacture of elongated tubular charges.

As a shell of the tube, polyethylene of high pressure PEHP 15803-020 was used.

Table 1

**Physico-chemical and calculated thermodynamic characteristics
of the composition on the basis of PP**

Characteristics	Value for mixtures	
	with diesel fuel	with nitromethane
Oxygen balance, %	+25.67	+37.7
Heat of explosion, kJ/kg	2165.8	858.1
The temperature of the explosion, K	2003.65	1523
Volume of gases, l/kg	334.4	374
Density of the composition, g/cm ³	0.99–1.02	1.05–1.10
Sensitivity to impact according to GOST 4545, frequency of explosions in the device 1, %	36	41
Lower threshold of friction sensitivity under impact shift according to GOST 3 50835-95 in K-44-3 device, kg/cm ²	3650	3740
Exhaustion, ml	85	95
TNT equivalent	0.51	0.55
Critical diameter of detonation, mm	5–19	10–12
Detonation speed, km/s	1.8–2.1	1.0–1.55

Non-explosive materials such as potassium perchlorate with decomposition catalysts and combustible components like diesel fuel (DF) or nitromethane (NM) are mixed at the plant and fitted into plastic tubes, the ends of which are hermetically sealed with stoppers. Such tubular charges with an explosive mixture are transported to the site of blasting operations. Transportation of explosive charges requires special transport, the coordination of the route with the traffic police and the protection of dangerous explosive cargo. Typically, charges are shipped in quantities that provide long-term performance of blasting operations. And since explosive charges are need to be kept, for this purpose it is necessary to have guarded warehouses.

As an alternative, the explosive composition, that consisted of non-explosive components (potassium perchlorate, manganese dioxide and diesel fuel or nitromethane) is proposed to be manufactured directly at the site of blasting, that is, at the quarry station [17]. Non-explosive materials only after mixing form a blend with explosive characteristics. This allows to exclude the transport of explosive mixtures from the manufacturer by special transport

means on the territory of the country, which is related to the safety of the population and reduce the number of protected storage areas.

The given task is solved by the fact that at the enterprise the plastic tubes are made, filled with early prepared potassium perchlorate with decomposition catalysts and seal hermetically with corks.

Production of explosive charges consists of four main stages:

- 1) preparation of components: drying, crushing, screening;
- 2) mixing of components in powder type;
- 3) manufacture of polymeric membranes;
- 4) manufacture and equipment of elongated tubular charges.

Stages 1-3 are carried out at the enterprise and only the 4th, which includes the dosage of diesel fuel (nitromethane) and the final equipment of explosive charges takes place directly on the ground of blasting operations.

Since the tubes are filled with non-explosive material, they are sent to a career by conventional road or rail transport, where they can be stored in ordinary material depots. Combustible components such as diesel fuel or nitromethane are purchased by companies at local plants or gas stations. Only after the filling of the combustible component into the tubes with early prepared non-explosive material (potassium perchlorate with metal oxide) the mixture becomes explosive.

Therefore, the manufacture of explosive charges is carried out only on the ground of blasting by filling the required amount of combustible component into plastic tubes, filled with a mixture of PP with a decomposition catalyst. For this purpose, the cork is opened and filled with the calculated amount of combustible component.

For the purpose of an experimental verification of the described method, polyethylene tubes with a diameter of 20 mm and a length of 400 mm were made. Also the polyethylene plugs and stoppers for sealing tubes were made. Potassium perchlorate was dried in a stove at a temperature of $85 \pm 5 \text{ }^\circ\text{C}$ ($358.5 \pm 5 \text{ K}$) to a moisture content of no more than 0.2%. Then PP was chopped in a ball mill and sifted through a sieve with a diameter of the holes of 400 microns. Manganese (IV) oxide was wiped through a sieve with a diameter of holes of 100 microns. The components of the explosive composition were mixed in a laboratory mixer with Z-shaped blades. First,

potassium perchlorate was charged, then a catalyst, which is presented by manganese oxide was added and stirred for 10 minutes. The prepared mixture was filled into the polyethylene tubes and closed with stoppers.

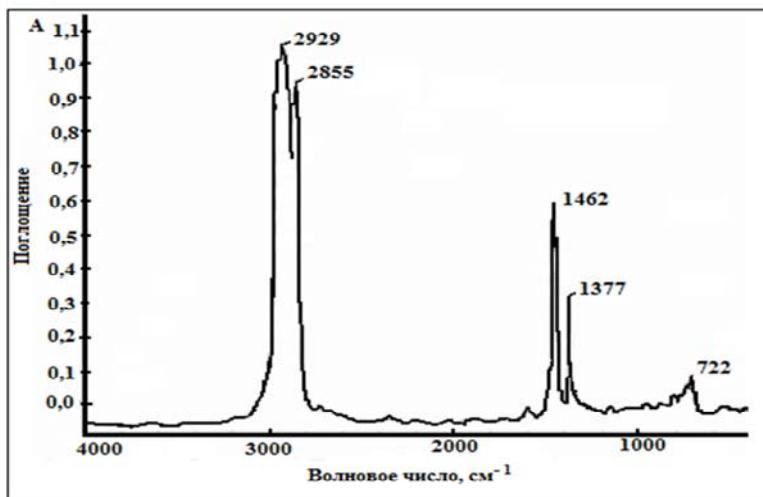
Prepared polyethylene tubes with potassium perchlorate and decomposition catalyst and separately diesel fuel in the canister were transported to the test site to obtain an explosive mixture and to determine the explosive characteristics. To mix the components, the polyethylene cylinder containing PP was taken, placed vertically and the carble was opened from the one side dispensing the estimated amount of diesel fuel. In one cylinder contains 120 g of PP with a catalyst. To obtain an explosive mixture of PP and diesel fuel in the ratio of 88:12 it necessary to add 19 ml of diesel fuel. Uniform distribution of diesel fuel in the mixture is observed up to a concentration of 14-15%, with an increase of concentration, there is a drain of fuel. It has been established that for full immersion of PP by diesel fuel in a tube of 20 mm in diameter and 400 mm in height, a holding time of 50 minutes is required. The method of manufacturing an explosive mixture in the field of blasting does not require the cost of additional expensive equipment: it is enough to have milled glasses or graduated cylinders.

In order to verify the quality of explosive mixture production at places of realization of explosive works, the detonation speed was determined by the Dotresh method. The initiation of an explosive charge was carried out with the help of one of modern means of initiation such as an electrodetonator ED-8. Three parallel tests were conducted. In all cases, the explosive composition detonated reliably, the detonation speed was 1830 m/s. This allowed us to conclude that the method of making charges at places of realization of explosive works is workable.

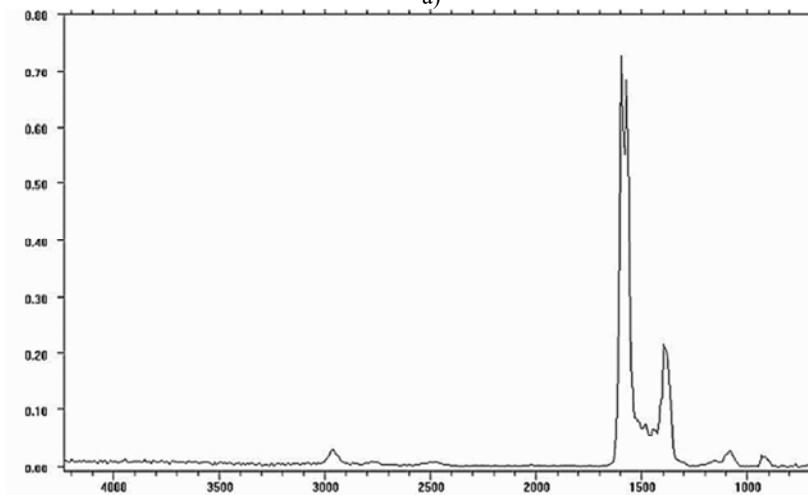
Also, to study the uniform distribution of the combustible component throughout the volume of elongated charge, a qualitative and quantitative analysis of components of the investigated industrial explosive substance was carried out using the method of infrared spectroscopy based on Fourier transformation (ISFT).

To do this, initially the infrared spectra of combustible components: diesel fuel and nitromethane were considered, which belong to the class of alkanes and nitro compounds, respectively.

Schematic images of the characteristic regions of the spectra of these compounds are presented in Fig. 2 [18, 19]. IR spectrum is the plot of absorption or transmission dependence (in %) from the frequency (wave number) ν (cm^{-1}).



a)



b)

Fig. 2. IR spectra of: a) diesel fuel; b) nitromethane

From Fig. 2a) it is seen that the IR spectra of alkanes are characterized by the appearance of absorption bands due to C-C and C-H bonds (CH_3 3000-2800 cm^{-1} , CH_2 1400-1300 cm^{-1} , C-H \sim 700 cm^{-1}).

From Fig. 2 b) it can be seen that compounds having a nitro group have two strong absorption bands in the range of 1650-1500 cm^{-1} and 1390-1250 cm^{-1} , as well as a band of variable intensity in the range of 920-830 cm^{-1} . It is known that nitromethane absorbs at 1580 and 1375 cm^{-1} [20].

To conduct the researches on the uniform distribution of the combustible component, tubes with potassium perchlorate were prepared according to the previously described technology, then the calculated amount of diesel fuel or nitromethane in ratios of 95:5 and 90:10, respectively, was added. After impregnation of the material, each three samples at different height of the tube (top, center, and bottom) were selected and a qualitative and quantitative research based on infrared spectroscopy based on Fourier transformation was conducted. The obtained spectrograms were identical in qualitative and quantitative composition.

The results of ISFT obtained with device Cary 630 FTIR, for a sample of PP without combustible components, are presented in the Fig. 3.

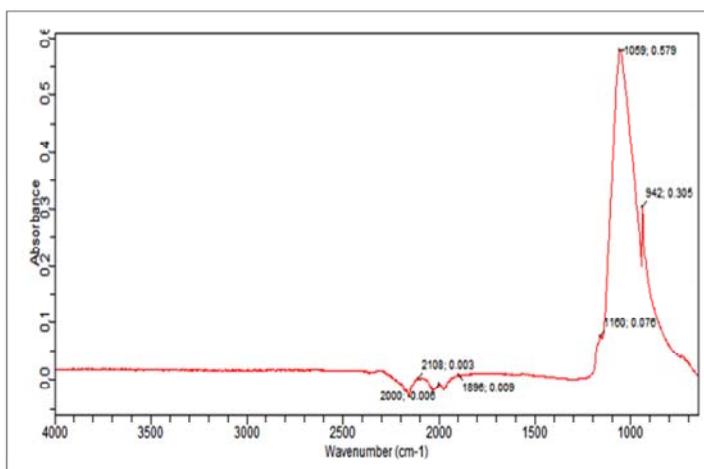


Fig. 3. Infrared spectrum of potassium perchlorate

From Fig. 3 it is seen that the PP has one strong broad absorption band in the range of $1160\text{-}1059\text{ cm}^{-1}$ and one variable band at 942 cm^{-1} , corresponding to the anion ClO_4^- [20]. The presence of weak bands in the range $2000\text{-}1896\text{ cm}^{-1}$ is due to the fact that the initial potassium perchlorate is a technical product, therefore, the presence of a small amount of impurities, such as chloride ions Cl^- and sulfite SO_4^- , can't be ruled out.

The results of ISFT, which are obtained for a sample of potassium perchlorate with diesel fuel as a combustible component in the ratio of 95:5 are presented in Fig. 4.

From Fig. 4 it is evident that in addition to the absorption bands inherent in potassium perchlorate, there are bands that are characteristic of diesel fuel. The sample has two absorption bands of medium intensity at $2986\text{-}2924\text{ cm}^{-1}$ and $2870\text{-}2845\text{ cm}^{-1}$, which correspond to the valency antisymmetric oscillation of the structural fragment CH_3 . The absorption bands of weak intensity in the range of $1480\text{-}1378\text{ cm}^{-1}$ correspond to deformation oscillations of the structural fragment CH_2 .

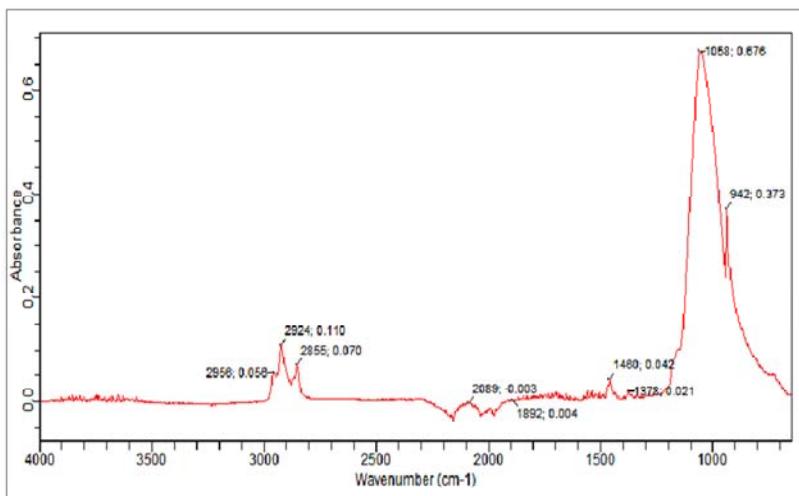


Fig. 4. Infrared spectrum of a mixture of potassium perchlorate with diesel fuel

In the course of the research, IR spectra of three samples of PP with diesel fuel were determined. The quantitative composition of the components was judged according to the area of the peaks. It is concluded that the samples are identical [21]. The error of measurement is not more than 2%.

At the next stage, the samples of potassium perchlorate with nitromethane as a combustible component were considered similarly to the previous researches. The results of infrared spectroscopy are presented in Fig. 5.

From Fig. 5 it can be seen that in addition to the absorption bands inherent in potassium perchlorate, there are bands that are characteristic of nitromethane. The investigated sample absorbs at 1580 and 1375 cm^{-1} , which corresponds to the full extent of nitromethane.

In the course of the study, infrared spectra of three samples of potassium perchlorate with nitromethane were determined. The quantitative composition of the components was judged according to the area of the peaks. It is concluded that the samples are identical [21]. The error of measurement is not more than 2%.

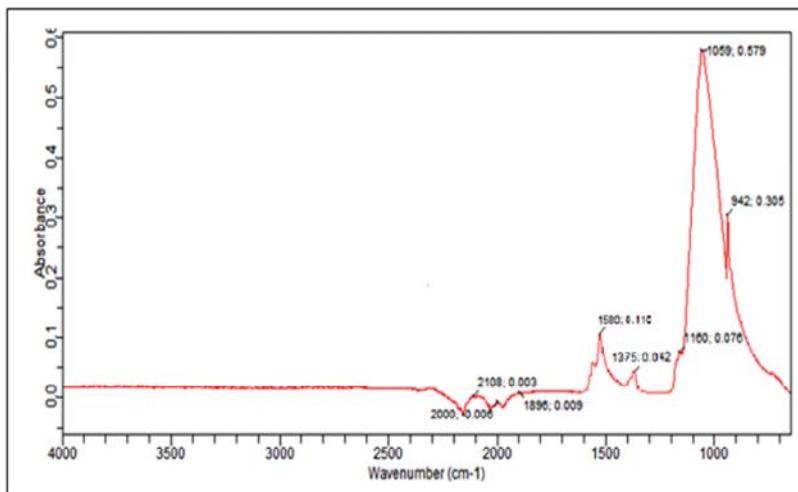


Fig. 5. IR spectrum of a mixture of potassium perchlorate with nitromethane

In addition, it is determined that the optimum content of combustible components: diesel fuel (DF) is 5-6 %, nitromethane – 10 %, while the physical stability of the composition is maintained without draining of the fuel down the charge. An increase of fuel content in the case of diesel fuel up to 10-12% somewhat reduces its physical stability and does not reduce the critical diameter of the detonation and increase the speed of detonation. The critical diameter of the detonation of the sample with diesel fuel containing of 10% remained at the sample level like as the sample with diesel fuel containing of 5% – 19-20 mm, the detonation speed at the same time remained at the point of 2.0 km/s.

A similar situation is observed with nitromethane also. Increasing of the combustible component amount does not improve the explosive characteristics, but it reduces the physical stability, which is inappropriate for the means of a block stone reflecting.

It was also found that for full immersion of potassium perchlorate with diesel fuel (nitromethane) in a tube of 20 mm in diameter and 400 mm in height it is necessary to hold up to 20-30 minutes. After this time, tubular charges of local production at the places of realization of explosive works are ready for use to reflect the block stone.

The manual screw mixer MUNSCH MAK-32 was used for dosage of components in the polyethylene shell (Fig. 6).



Fig. 6. The manual screw mixer MUNSCH MAK-32

The nozzle of this mixer rotates 360 ° along with the hot air nozzle built-in thermofen. The maximum productivity is 35 kg/h, which is sufficient for the preparation of charges at the places of realization of explosive works. The power of this device is: 220 V, 50 Hz. The length of the extruder is 570 mm. The installed capacity of the extruder is 3 300 Watts.

Consequently, the dosing of the components into the shells at the places of realization of explosive works using a manual screw mixer MUNSCH MAK-32 is not labor-intensive, due to the average performance of the used easy management equipment and does not require a lot of time as well as energy of employees on a career and special skills compared with more sophisticated equipment.

The main advantages of elongated tubular elastic charges are:

- easy charging;
- the possibility of spatial allocation of charges;
- the presence of air space between the charge and the walls of the hole reduces the peak pressure during detonation, which helps to reduce cracks in the separable block;
- the use of a detonation cord or an electrodetonator makes it possible to initiate tubular charges located in the hole, practically simultaneously, which provides practically equal loading of the pressure of the explosion along the entire length of the charge;
- absence of contact of the personnel with the explosive;
- reduction of the explosive cost compared with the bulk charge;
- reduction of the emission of harmful gases into the atmosphere due to the small volume;
- reliability of the charges operation from the staff of initiation;
- preparation of explosive charges of blasting on the basis of non-explosive components directly at the site of blasting operations.

Proceeding from the above advantages and experiments, as well as increased demand for products made from block stone, in Ukraine, the development of domestic construction of elongated charges of career production (such as K-tubes) is proposed.

Conclusions

1. The properties of block decorative stone and requirements to methods of its extraction, the performance of which will ensure an

increase in productivity while maintaining the quality of minerals, are considered

2. A domestic low-speed charge of the type like K-tube for reflection of a block stone has been developed. It fully meets the requirements for the means of block stone extraction in economical mode.

3. The technology of making charges at the places of realization of explosive works using a manual screw mixer MUNSCH MAK-32 has been exhausted. This eliminates the transport of explosives from the manufacturer by means of special transport on the territory of the country, reduces the danger to the population and the number of protected warehouses.

4. To verify the quality of explosive mixtures manufacturing at the places of realization of explosive works, the detonation speed by the Dotresh method has been determined. The initiation of an explosive charge was carried out with the help of one of the most used means of modern initiation such as the electrodetonator ED-8. In all cases, the explosive composition detonated reliably, the detonation speed was 1830 m/s.

5. Due to the method of infrared spectroscopy based on Fourier transformation, it is determined that the distribution of the combustible component (diesel fuel and nitromethane) along the height of the elongated charge is uniform. This indicates that the domestic method of making charges at the places of realization of explosive works is workable, and the very charge of the type like K-tubes is qualitative.

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ELECTRICALLY STIMULATED PHASE TRANSFORMATIONS IN BITUMINOUS COALS

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Abstract

A comparative analysis of physical and chemical changes in coal subjected to the impact of electric fields of weak density with coals after a sudden outburst, as well as those selected from the outburst-prone and non-hazardous zones of coal seams, has been performed. In the studies, physical methods were used, including X-ray analysis, electron paramagnetic resonance, thermogravimetric analysis, differential scanning calorimetry, laser diffraction analysis of particle sizes, IR spectrometry, nuclear magnetic resonance, Raman spectroscopy and cross-polarization. It is shown that the destruction of the coal organic mass can be caused not only by mechanical or thermal effects, but also by weak electric fields. The scientific novelty consists in the fact that for the first time the identity of the nature of the destruction of coal organic mass has been shown both subjected to the mechanochemical activation and to the impact of weak electric fields on the previously destabilized coal microstructure. The basis of the destruction mechanism in both cases is the regularity of the action of thermal fields. Practical significance is determined by the use or consideration of the obtained regularities in technologies of coal processing in other products. The results of the research can be useful in developing methods for suppressing or reducing in the coals the potential for outburst.

Introduction. Global fuel resources, particularly the discovered coal reserves, provide the ground for the optimistic prognosis for the

coming 59-100 years. With the view to at least partially replacing conventional coal burning, alternative technologies of generating energy are on the rise and will most likely be at the peak of their development during the next two decades. However, in chemical industry, coal still remains a most valuable (sometimes the sole) source of raw material for manufacture of various chemical products and fuels. It can be assumed that the trend of using coal in combustion technologies will stand in certain countries in the first half of the 21 century and chemical technologies of coal processing and new materials production will get a new momentum. Thus there is (and hopefully will be) practical need in scientific findings obtained as a result of research into phenomena, properties and regularities manifest in coals under the impact of external physical fields. In this context, nature gives us a valuable lesson to learn by comparing the obtained results with experimental research of coal samples under the impact of individual physical and mechanical fields and in conditions of their aggregated influence.

In nature, phase and structural transformations occur in the system of coal organic mass-gas under the simultaneous impact of several external factors, the principal factors being weak strength of electric and magnetic fields, temperature and conditions of mechanical stresses effect in coal seams. Under the impact of the complex of physical fields, the coal organic mass undergoes specific changes of molecular organization resulting in the increased surface of phase transformations, concentration of free radicals and other moving particles. Coals provide favourable conditions for the formation of free atomic carbon and crystallization of various carbon phases. The role of catalytic reactions and hard surfaces in carbon-forming processes increases. Most of the above phenomena and possible physical or chemical processes are studied as separate or occasional events, not related to the evolutionary character of carbon formation, irrespective of the kinetic factors influence [1]. There is no relation between the peculiarities of the mechanism and conditions of carbon formation with regularities of manifestations of certain coal properties, particularly of outburst hazard.

It is assumed that every above-mentioned factor played a leading role at a certain stage of carbon physical and chemical properties formation, i.e. we can apply the Le Châtelier-Braun principle to coal-

gas system under the condition that temperature, weak electric and magnetic fields, pressure and mechanical stresses are acting simultaneously. The conditions and factors specified above are instrumental in the system achievement of both thermodynamic and chemical steady equilibrium. The results of studies into certain physical properties of coals, reasons for emergence and preservation of their instable microstructure as an active state potentially capable of initiating and developing physical and chemical processes - cannot be interpreted unequivocally within the frames of thermal and dynamic ideas. On the whole, the drawbacks of certain research approaches can be related to the fact that the system comprising nano-sized components [2] was previously studied mostly on the basis of the methodology intended for the research into micro-sized objects [3,4]. Published works scarcely pay any attention to mechanisms of carbon nanophases origin. Hence, peculiarities of free atomic carbon physical properties, mechanism of hard nanoparticles formation from carbon gas and their properties are of scientific interest. Such nano-sized phases as carbon and hydrocarbon chains, graphene [5] etc, were not as a rule considered, whereas in the result of physical and chemical transformations, these phases acquire new properties [6] and can produce a significant effect on the coal physical properties formation [7].

Porosity (internal specific surface area) as one of the fundamental coal properties is studied by many scholars [8]. Internal specific surface area influences diffusion, adsorption, filtration and chemical properties of coals which determine their chemical activity, gas permeability, etc. In certain physical and chemical conditions (e.g. in the process of carbonization [1]), the surface as a kinetic parameter is the surface of potentially active phase transformations. Gas-liquid fluids filling coal pores are genetically related to static components of the solid surface. Under external physical impact, they become chemically active, which is one of the factors inherent to carbon formation process. Destructurization and consequent degradation of such system start, as a rule, from the breaking of individual chemical bonds in coal components. This process may create the conditions initiating transition into the state of unstable equilibrium possibly with the risk of gas-dynamic phenomena development. The resulting coal-gas system in accordance with its new thermodynamic state can

produce a significant effect on the character of mechanical stresses distribution in macroscopic space of the rock mass [9].

No matter the change of what physical parameters responsible for the system equilibrium breaks it, the chemical reactions in coals result in phase and structural transformations, i.e. destructurization. In some cases, the finished product of such impact is outburst-prone coal whose microstructure is characterized by a great amount of inner energy which is potentially able to initiate the transformation of coal organic mass into gas [9]. The assumed history of coal acquisition of new properties testifies about evolutionary (regular) character of this process. Gas-dynamic effects in mines are still one of most important scientific problems so far lacking physically sound solution [10].

A problem worth scientific investigation is the research into the influence of weak electric and magnetic fields on phase transitions in mechanically pre-activated coals. The interest in this problem is caused by the fact that in natural conditions, tectonic activation is manifested as complex mechanical effects accompanied by the increase in electric and magnetic fields density, local temperatures, i.e. physical factors able to start active chemical processes.

Fields of weak density were not used in experimental research as a rule, since their low energy impact was obvious. But the current assumptions about negligible role of electric fields in the development of physical and chemical processes have been contradicted in the research into the influence of electric [11] and magnetic [12] fields of weak density on coal transformations, initiation of phase transitions and structural transformations in minerals of carbonate group (siderite, calcite) [13], on fractional composition of particles of diamond synthesized in high-pressure chambers [14] etc.

The cutting-edge character of this research arises from the necessity to create a unified system of physical ideas about the nature of coal solid components transition into gas without high temperature activation of chemical reactions. Moreover, it is still vital to study the problems of various carbon phases formation from free atomic carbon [15]. Research related to obtaining new products and processes with customized physical, chemical and technical characteristics is also of great practical interest.

The purpose of the research. To conduct a comparative analysis

of physical and chemical changes in coals extracted from different parts of coal seams and the samples subjected to the impact of electric fields of weak density. To show that the destructive processes in coal organic mass can be triggered both by mechanical and electric effects.

The goal set was achieved by research in two stages: 1 – complex study of coals not subjected to additional treatment; 2 – study of coals crushed in the mortar and treated by the electric field of weak density. In the analysis, we considered the known works of T.M. Khrenkova et al. in mechanochemistry of coal, the findings of researchers from Northern Caucasian Scientific Centre of Higher Education (Russia) and other research institutions. Comparative analysis of physical and chemical coal characteristics during the first and the second stages of the research is aimed at contradiction or justification of the principal scientific statements of the hypothesis about the mechanism of microstructure and phase instability formation [8].

Materials and methods. In experiments, we used gas coal (I) (carbon content 86.6%, hydrogen content 5.7%, vitrinite reflectance $R_0 \geq 0.83\%$, $Y=14$ mm, $V^{\text{daf}}=36.2\%$) and fat coal – (II) (carbon content 88.2%, hydrogen content 3.9%, vitrinite reflectance $R_0 \geq 1.09\%$, $Y=17$ mm, $V^{\text{daf}}=31.3\%$). The coal samples were extracted from hazardous and non-hazardous seams in terms of coal and gas outbursts. The samples were prepared from coal ground to fractions of 200/100 μm (according to the data obtained by laser diffraction analysis, the sizes of the original coal particles were 214.5 – 111.7 μm). The average mass of each sample ~ 1.28 g. Before the experiment, the coal was dried up at 35 °C for 48 hours. Maximum temperature of heating during electrophysical treatment did not exceed 320 K. Experiments were conducted according to the procedure described in [11]. Difference of potentials (1 – 300 V) and the current value (0.3 A) were set by the power source. Each sample was treated for four hours. X-ray phase analysis was conducted by ДРОН-3 unit. Certain physical and chemical characteristics were studied with the help of the following techniques and equipment: Mettler Toledo thermogravimetric analysis (TGA) and differential scanning calorimetry (DSC), LEICA DM ILM optical microscope, Shimadzu SALD-301V laser diffraction analyzer of particle sizes and C-2000 IKA calorimeter. Infrared coal spectral data were collected by IR Fourier spectrometer ФСМ-1201 with the run in the

spectral range of 400...5,000 cm^{-1} . The nuclear magnetic resonance (NMR) was studied according to the procedure developed at the Institute for Physics of Mining Processes of the National Academy of Science of Ukraine (IPMP NASU) [16].

Results of experiments and their discussion. It is assumed that the coals treated by weak electric field differ from the original ones and can generally shape the character of subsequent physical and chemical processes. Electric fields of weak density and thermal activation within the temperature range 310 – 320 K were used in the research related to the development of destructive and gas generating processes in gas (I) and fat (II) coals.

Spectra of coal nuclear magnetic resonance (NMR-spectra) were registered with NMR radiospectrometer of broad lines designed at the IPMP NASU. The density of the constant magnet field was 4,600 Oe, field uniformity $2 \cdot 10^{-6}$ Oe/cm, resonance frequency 19.6 MHz. The NMR spectrum of gas-saturated coal (Fig. 1) comprises a narrow component – Lorentz line (5) and broad spectral component – Gaussian line (3), whose parameters contain information about the amount of hydrogen in coal structure. Since the spectra obtained by means of ^1H NMR method of broad lines are the sum of the derivatives from the absorption lines, superposition of the first derivatives from Lorentz and Gaussian lines is used for interpolation of ^1H NMR experimental spectra [16].

To study the parameters of methane desorption, the samples were dried up at 363 K and saturated with methane for 15 days. After that they were placed in NMR spectrometer where ^1H NMR spectra were registered in the process of methane desorption.

Moisture content in both original samples was higher than in those treated by electric field. This must be the reason why the areas of ^1H NMR spectrum narrow lines for these samples were also bigger. The corresponding areas of broad lines (from coal organics) practically do not differ in value within the limits of the experimental error. The main parameters of NMR signal: signal area S , proportional to the number of studied nuclei in the unit of matter volume; broad line ΔH – distance between the maximums (A/m). Table 1 lists experimental data whose nomenclature corresponds to ^1H NMR spectrum presented in Fig. 1.

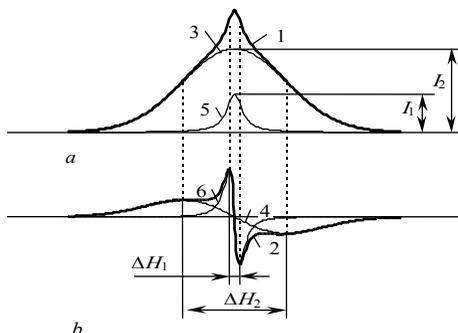


Fig. 1. NMR spectrum absorption line of coal containing fluid (a), and derivative from the absorption line (b): 1 – spectrum of absorption; 2 – first derivative from the spectrum of absorption; 3, 4 – wide component of the absorption spectrum and its first derivative respectively; 5, 6 – narrow component of the absorption spectrum and its first derivative respectively

Table 1

Characteristics of coals treated by electric field

Gas coal (I) treated at $T = 320$ K; moisture content before measurement 2.14%								
Spectrum №	I_1	ΔH_1	I_2	ΔH_2	S_n	S_b	S_n/S_b	I_2/I_1
1	1.53	6.71	0.734	0.403	0.905	12.830	0.070538	0.479739
2	1.48	6.52	0.717	0.357	0.786	12.068	0.065131	0.484459
3	1.58	6.62	0.723	0.349	0.775	13.076	0.059269	0.457595
mean	1.53	6.617	0.725	0.370	0.822	12.658	0.065	0.474
Gas coal (I) original; moisture content before measurement 2.27%								
Spectrum №	I_1	ΔH_1	I_2	ΔH_2	S_n	S_b	S_n/S_b	I_2/I_1
1	1.36	6.18	0.7422	0.563	1.265	10.521	0.120236	0.545588
2	1.63	6.21	0.704	0.401	0.864	12.6	0.068571	0.431902
3	1.77	6.65	0.748	0.452	1.032	14.713	0.070142	0.422599
mean	1.5877	6.347	0.731	0.472	1.054	12.635	0.083419	0.460618
Fat coal (II) treated at $T = 320$ K; moisture content before measurement 0.86%								
Spectrum №	I_1	ΔH_1	I_2	ΔH_2	S_n	S_b	S_n/S_b	I_2/I_1
1	1.61	6.2	0.459	0.363	0.511	12.495	0.040896	0.285093
2	1.68	6.23	0.437	0.384	0.514	13.1	0.039237	0.260119
3	1.5	6.66	0.38	0.426	0.495	12.487	0.039641	0.253333
mean	1.597	6.363	0.425	0.391	0.507	12.694	0.03994	0.266124
Fat coal (II) original; moisture content before measurement 1.09%								
Spectrum №	I_1	ΔH_1	I_2	ΔH_2	S_n	S_b	S_n/S_b	I_2/I_1
1	1.42	6.08	0.329	0.705	0.696	10.81	0.064385	0.23169
2	1.5	5.85	0.295	0.593	0.529	10.991	0.04813	0.196667
3	1.54	6.40	0.329	0.707	0.698	12.331	0.056605	0.213636
mean	1.532	6.11	0.318	0.668	0.641	11.377	0.056342	0.207572

I_1, I_2 – amplitudes of the broad and narrow spectrum lines; $\Delta H_1, \Delta H_2$ – their width; S_b, S_n – their areas.

The obtained spectra allowed to determine the change in ^1H NMR spectrum narrow lines amplitude (intensity), which are related to change in methane content in time, and to define the typical time of desorption T_{dec} (Fig. 2, Tab. 2). The spectra were registered in identical conditions (the study was conducted by Ye.V. Ulianova, Dr. Sc. (Tech.).

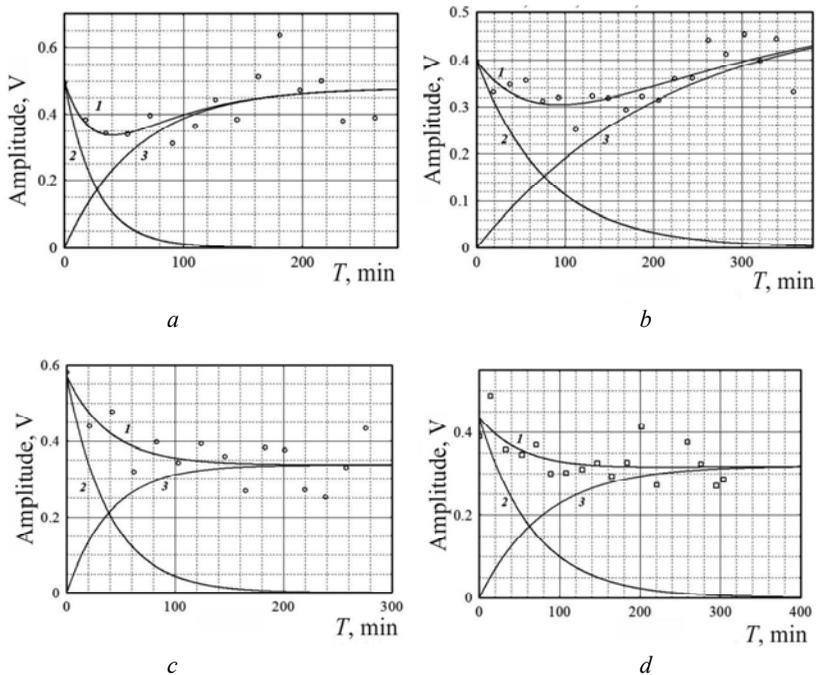


Fig. 2. Change in the amplitude of ^1H NMR spectrum narrow line of coal gas saturated sample during methane desorption (points on graphs) and decomposition of the obtained dependence (curves): a – gas coal (I) original, b – treated gas coal (I), c – fat coal (II) original, d – treated fat coal (II)

Table 2

Change in methane desorption time T_{des} from original coals and coals treated by the electric field

Coal	Gas coal (I)		Fat coal (II)	
	original	treated	original	treated
$T_{\text{des}}, \text{min}$	25.9	79.5	39.1	68.0

In this experiment, the dehydrated samples saturated with methane were studied in the spectrometer which was not isolated from the atmosphere air that is why the obtained experimental results (points in Fig. 2, a-d) reflected how coal samples absorbed the atmosphere moisture whose hydrogen contributed to the registered ^1H NMR spectra. Thus, experimental dependences, which reflect the change in the amplitude of gas-saturated coal ^1H NMR narrow line related to time during desorption, do not show its gradual drop to zero. Using the procedure described in [17], it is possible to plot a dependence reflecting methane desorption from the sample on the basis of the data obtained. In the data processing, we used the interpolatory dependence (curve 1 in graphs):

$$y = a \cdot \exp(-t/T_{\text{des}}) + c \cdot [1 - \exp(-t/T_{\text{sorp}})],$$

where a and c – amplitude indices reflecting the content of water and methane in a sample, T_{des} – typical time of methane desorption from a coal sample, T_{sorp} – typical time of atmospheric moisture desorption by a coal sample, t – time of the experiment.

The first term in the right part of this equation describes methane desorption (curve 2 in graphs), the second term - atmospheric moisture desorption (curve 3 in graphs).

The typical time of methane desorption increases significantly in the treated coals. Comparison of desorption from the original sample and the one treated by the electric field allows to conclude that gas saturation of the latter decreases only by a few percent, while methane emission rate increases significantly. Hence, for gas coal (I), methane emission rate is three times higher, for fat coal (II) – 1.7 times higher. Decrease in gas saturation of the treated coal samples is related to additional gas release by the coal organic mass which underwent the processes of destructurization under the influence of electric field.

Similar results of the research into coal samples taken from seams located in hazardous zones (coals with faulty microstructure) are discussed in [16]. This work also lists the results of methane desorption from coals extracted from safe zones and from the zone of abrupt coal and gas outbursts. Methane desorption rate is almost identical for all samples. The ejected coals are characterized by a

low intensity of gas emission. Small gas content of the thrown coals is probably related to the loss of reservoir properties. Perhaps, the reason for sharp attenuation of gas emission is deterioration of coal organic mass chemical activity caused by the decrease in excessive energy reserve in microstructure.

Mass loss of gas saturated samples due to methane desorption was studied at 363 K by the gravimetric method. The obtained dependences (Fig. 3) for fat coal (II) show that a sample of gas saturated original coal loses 0.063 g in 100 sec, and 0.088 g in 400 sec. Mass losses of coal treated in electric field are 0.065 g and 0.075 g in 100 sec and in 400 sec respectively. Thus, the original sample mass loss rate is higher than that of the treated one, the same being true for the gas coal (I) samples.

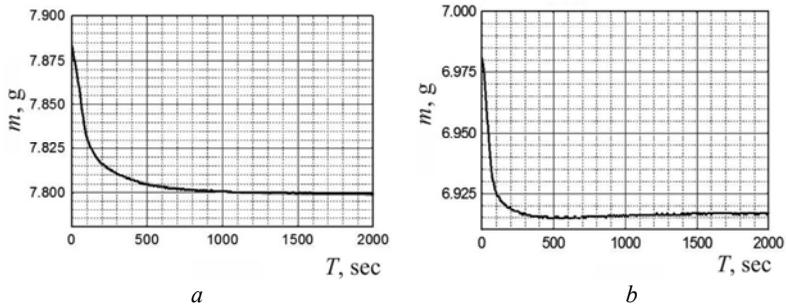


Fig. 3. Dependence of mass loss by fat coal (II) sample on time:
a – original, *b* – treated

The samples of coal used in the experiments after crushing were characterized by increased chemical activity relative to the initial ones. According to data of electronic paramagnetic resonance (EPR), the density of paramagnetic centres (PMC) increased from $N = 6 \cdot 10^{18}$ PMC/g to $1.4 \cdot 10^{19}$ PMC/g. Additional treatment with an electric field leads to an increase in the PMC concentration: ($N = 8.3 \cdot 10^{19}$ PMC/g). In the coal organic mass of outburst-prone seams and zones, according to the data [16, 18, 19] there are changes in molecular organization and structural chemical transformations reflecting the state of mechanical activation of the coal organic mass. From the analysis of the EPR lines of coal from outburst-prone and non-hazardous zones it follows that the broadening of the EPR line

(for example, up to 35 mT) for the outburst-prone coal is due to free radicals and is described by the Lorentz equation. The broadening of the EPR line occurs in the cases of mechanodestruction, thermal destruction, and coal electrodestruction, that is, due mainly to the thermal mechanism, the breaking of chemical bonds, can be due to the high content of impurities in the composition of the coals. This trend is also maintained in the case of coal treatment in weak magnetic fields. However, in the latter case, the broadening of the EPR signal can be due to spin exchange in the presence of oxygen and paramagnetic metal ions. Here the mechanism can be caused by the laws of the magnetic scenario (spin-selective) chemical reactions [20].

Unlike two-phase structures, coal as a multimer comprises associated macromolecules which include crystal-like and amorphized components. The change in their composition or concentration can lead to supramolecular restructuring [21]. Coal samples are known to acquire microscopic electric momentum in external electric field and become electrets. In practice, coal undergoes the processes of polarization in the following conditions: temperature range 293-473 K, external electric field density $10^3 \dots 10^5$ V/cm, the time of charging (with respect to method) is 5...180 minutes.

The time of coal electret charge relaxation is, as a rule, $10^4 \dots 10^6$ sec. Thermal electrets are characterized by the most stable electret state, with relaxation time $5 \cdot 10^6$ sec. According to [22], thermal electrets have three mechanisms of electret polarization. Relaxation of electret state is almost always 2...5 orders of magnitude slower than Maxwell relaxation – relaxation of free electric charge carriers.

In our experiments, preliminarily crushed coals were treated at the temperature 295...323 K, external electric field density 5...320 V/cm (weak fields), frequency 50 Hz, for 240 minutes. It was established that longer time of coal sample treatment does not result in significant increase in electret charge. Researches into coal ability to accumulate electric charge usually incorporate external fields of high density [22] that is why it is interesting to study the effect of weak electric fields, particularly the time of electret charge relaxation.

Increased chemical activity of the crushed coals treated by the weak electric field can be explained by catalytic reactions which become more intense due to the growing concentration of chemically active centres on coal inner surfaces. It is known [23] that uncompensated electric charges create a quasi-static electric field in the space around the electret. Numerical modelling allowed to define the impact of the point charge field on the degree of chemical bond stability (by the example of small molecules). Thus, when the bond approaches the charge at some critical distance, it breaks [24]. The model is based on quantum and mechanical correlations. We assumed the temperature of the chemical reaction occurrence to be 0 K, i.e. the field has enough energy to overcome energy barrier which is needed to break the bonds of molecules including CO and N₂. Treatment of gas (I) and fat (II) coals confirmed their electret nature. Experiments were conducted according to the procedure described in [11]. Potential difference was set by the power supply at 299 V, with electric field density 290±15 V/cm, time of each sample treatment was ~ 4 hours.

After mechanic and electric treatment, coal samples acquired electret potential difference (U_{EP}), which was determined by means of compensation technique [22]. Value of the induced electric and chemical activity A_i for ejected fat coal (II) (Zasyadko mine, Ukraine) was $4.3 \cdot 10^{-5}$, from the safe zone – $0.7 \cdot 10^{-2}$. In [25], we find the results which are similar in terms of value and trend of electret parameters.

Fig. 4 shows dependence of electret potential U_{EP} change on time, which is the proof of super slow relaxation of electret state induced by the weak field. Analysis of coals treated and not treated by electric field (safe, prone to outburst and ejected, Tab. 3) allowed to establish that the time of electret state relaxation increases with the growth of structural defects concentration (in crystallites and coal polymer matrix) and electric charges. Consequently, reserve of energy stored in defects increases in coal nanostructure. Similar effects take place in physics of metals [26].

Authors of [27] claim that in the transition zone between “quiet” coal and outburst-prone coal there is a correlation between defects concentration, their energy parameters and coal macroscopic properties (in this coal seam zone, the charge concentration rise is

caused by the growth of defects concentration, hence the increase in charge energy). It is seen in Figure 4 that U_{EP} was registered continuously during the experiment (the coal sample was in the field with 5 V/cm density).

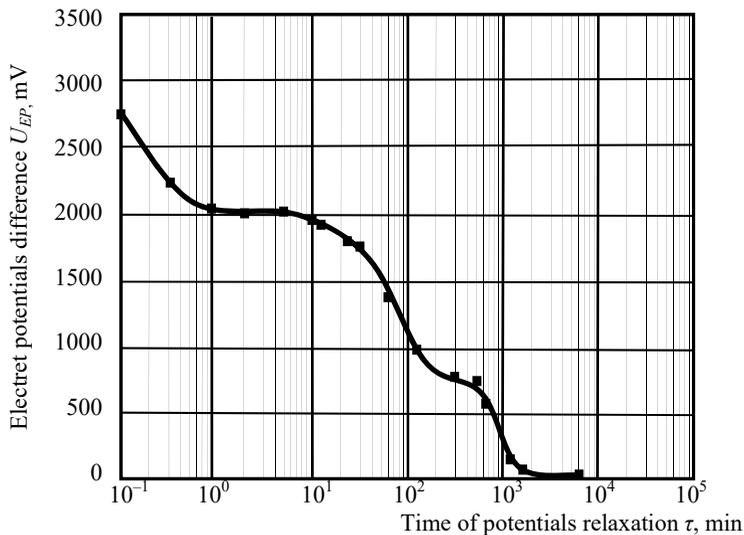


Fig. 4. Dependence of electret potential relaxation character on time in the sample of fat coal (II) from outburst-prone zone ($\Delta U_{IP} = 5\%$)

Table 3
Characteristics of electret state of bituminous coals treated by weak electric field

Fat coal (II)	Initial value of electret potential U_{EP} , mV	A_I	$N \cdot 10^{-18}$, PMC/g	Time of electret potential attenuation τ , h	Residue potential, $U_{EPres.}$, mV
Non-hazardous	2,100	$7 \cdot 10^{-3}$	60.0	117	0.88
Ourburst-prone	2,750	$9 \cdot 10^{-3}$	83.0	117	0.78
Ejected	13	$43 \cdot 10^{-6}$	6 → 14	117	0.63

We observed significantly lower values of electret potential difference in coals after the outburst than before it. In fat coals [27],

the maximum value of U_{EP} corresponds to outburst-prone states, and the minimum value – to the coals from the outburst zone whose potential abilities to induce and develop active chemical reactions are practically exhausted.

Table 3 lists the values of electret state stability caused by electrochemical activity (A_i) and slow relaxation of electret potential difference pre-treated by electric field. The parameters were measured at the ambient temperature, with the natural moisture content of the atmosphere, and the measurements were finished after several days of relaxation. Relaxation time of electret state for different coals was from $8 \cdot 10^4$ to $2 \cdot 10^6$ sec, which is orders of magnitude slower than Maxwell relaxation (~ 30 sec). The defects concentration rises as a result of coal mechanical crushing and additional treatment by electrical field. However, it should be noted that electric treatment of coal at ambient temperature does not practically lead to increased PMC concentration. With the growth of treatment temperature, PMC concentration in regard to original mechanically activated coal rises 2...5 times. Comparison of the data in [20, 26, 27] with our experimental data indicates that the values for the time of electret charge relaxation are similar. Besides, we can observe the like trends for the change in physical characteristics of coals treated in electric field as described in [27-29]. The obtained research results testify to the destructive character of the impact produced by even weak electric fields on coal. Analogous results may be expected for electric fields effect on coal seams. For the first time, the correlation between the ability of coal to accumulate electric charges and its proneness to abrupt outbursts of gas and coal was analyzed in [29].

X-ray structural analysis established that all diffractograms of the studied samples comprise two strongly diffused maximums related to angles of 2Θ : 24 and 43 degrees, Fig. 5 [11]. In the region of the first maximum are the lines with the following values (nm): 0.455; 0.424; 0.403 – weak line; 0.371 and 0.338. In the region of the second maximum on the diffractograms of all studied samples, we can fix the lines that can be related to crystalline phase with interplanar distance $d = 0.198...0.200$ nm, close in value to the second most intense line of graphite $d = 0.202$ nm.

Location of the first maximum insignificantly changes from

sample to sample, while the half-width remains almost the same. After electric treatment, the coal line intensity deteriorated. Increase in the half-width testifies to the decrease in particles dispersity and to the growth of the “amorphism” degree in general [30].

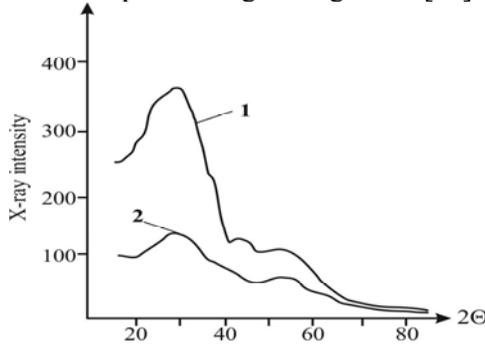


Fig. 5. X-ray diffractogram of gas coal:
1 – before treatment; 2 – treated by electric field

Destructive processes in coals are accompanied by the growth in smaller fractions share and, consequently, the increase in internal specific surface, the average particles’ size and their share decreasing by 3-8%, Fig. 6.

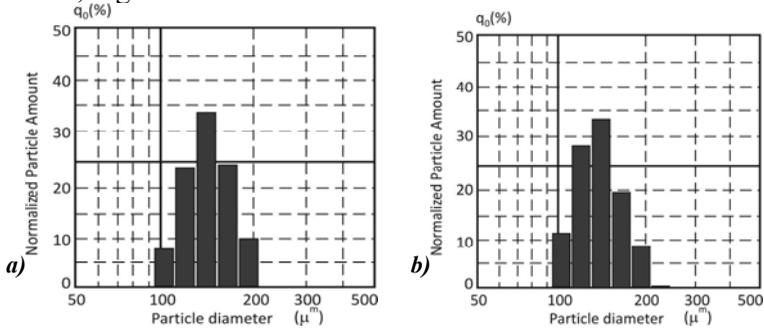


Fig. 6. Character of particles distribution by size:
1 – original sample; 2 – sample after treatment in electric field

Formation of mobile components (radical, gas) is related to the destructive processes in coal solid phase. Diffractograms prove that the structure “amorphism” degree increases, which is confirmed by

the data about the distribution of particles by size and the results of the research into electronic paramagnetic resonance, which is the evidence of growing concentration of paramagnetic centers. It is shown in [31] that ejected coal has a much bigger sorption surface than the unthrown coal from the same seam. The size of coal sorption surface depends on a number of factors. As is mentioned in [32], increase in coal sorption surface can be effected even by heating during samples preparation.

Detailed analysis of infrared spectroscopy of fat and gas coals presented in [11, 12] confirms the destructive character of processes in the studied coals stimulated by weak electric field treatment. The above data about the electronic paramagnetic resonance also provide the evidence of destructive processes development. Destructurization precedes transformation of a part of coal organic mass into gas phase. Thus, destruction of bridge aliphatic chains is proved by the decrease in optical density of bands 2920 and 2860 cm^{-1} , corresponding to valent and deformation fluctuations of C–H bonds. These bonds refer to the structures containing CH_2 and CH_3 -groups and can be determined by the bands decrease of 3000...3100 cm^{-1} in aromatic hydrocarbons. IR spectra of the coal treated in electric field are similar in character to the regularities established while studying coals taken from different spots of outburst-prone and safe coal seams [33].

To get additional information about changes in coal structure, we used spectroscopy of nuclear magnetic resonance on nuclei of carbon-13 (NMR ^{13}C) and spectroscopy of combination scattering (CS-spectroscopy) [2]. The coal was taken from outburst-prone zones and on the site after outbursts (coal from the seam h'_6 «Smolianinovskiy», Skochinskogo mine, Donetsk, Ukraine). The location was chosen because this bed yields the most outburst events (165) with average intensity of 289 tons.

Our research established correlation between coal outburst-proneness and total content of CH_2 - and CH_3 -groups, as well as the ratio of carbon atoms with different hybridization of valent electrons (sp^2/sp^3) [34]. Fig. 7 presents NMR CP/MAS ^{13}C spectra, taken by the technique of cross-polarization (CP) in order to single out hydrogen-containing fragments in coal matrix.

Thus, in coal samples from outburst-prone zones (spectra 1-2), we found excessive share of CH_3 -groups compared to their share in

coals taken from safe zones (spectra 4,5). Spectra of outburst-prone coal show the rise in carbon content in sp^2 -state, which is possible due to the growth of chain fragments containing CH_3 -groups.

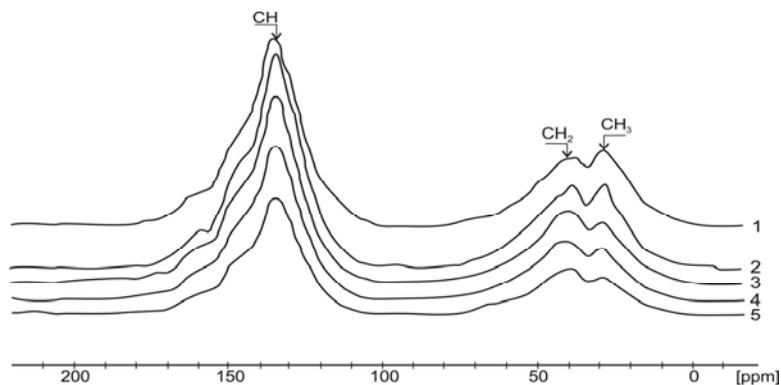


Fig. 7. NMR CP/MAS ^{13}C spectra of coal samples (seam h'6 at Skochinskogo mine): 1,2 – outburst-prone zone, 3 – after outburst, 4, 5 – safe zone [33]

NMR ^{13}C spectra of outburst-prone coals contain excessive share of CH_3 -groups (curve 1) compared to their share in coals taken from safe zones (curves 4,5). The share of CH_3 -groups in ejected coal (curve 3) dropped to the level corresponding to the safe zone (curves 4,5) and even lower. Fig. 7 clearly shows decline in intensity of peak from CH_3 -groups on the spectrum of ejected coal (curve 3). The values of integral CH_3 and CH_2 lines intensities of methyl and methylene groups calculated by Ye.V. Ulianova, testify to the decrease in their ratio (CH_3/CH_2) in ejected coal by 1.5 times.

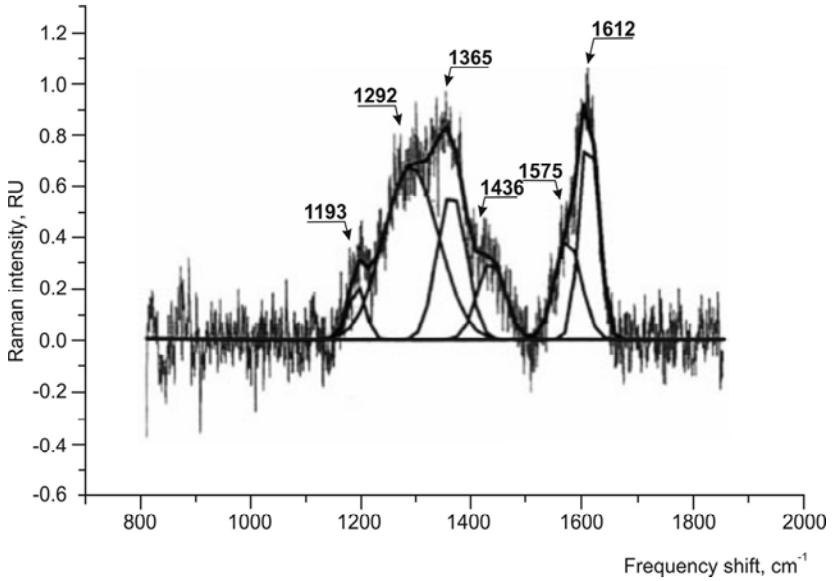
NMR of gas coal (I) from outburst-prone zone and outburst zone in seam h5 («Krasnolimanskaya» mine, Pokrovsk, Ukraine) also showed growing share of methyl fragments $-CH_3$ compared to their share in coal samples taken from the safe seam m_3 (Abakumova mine, Donetsk, Ukraine). Comparative analysis of sp^2 intensities; CH_2 , CH_3 characteristics of NMR ^{13}C spectra obtained with MAS and CP/MAS techniques allowed to identify the increase in $=CH$ -group content (carbon atoms in the structure of aromatic fragments and bonded chains in sp^2 -hybridization state) in the coal from outburst-prone zone compared to coal samples taken from the safe seam m_3 .

The technique of cross-polarization (CP/MAS) was used to enhance the signal from carbon atoms by attracting hydrogen atoms in the composition of fragments with $\equiv\text{C-H}$ -bonds [34]. Using state-of-the-art technology of NMR ^{13}C spectroscopy in the study of structural and functional transformations in multimers allows to register not only qualitative changes occurring in carbon-containing materials under the influence of different external factors, but also to make a quantitative assessment of carbon atoms distribution between functional groups.

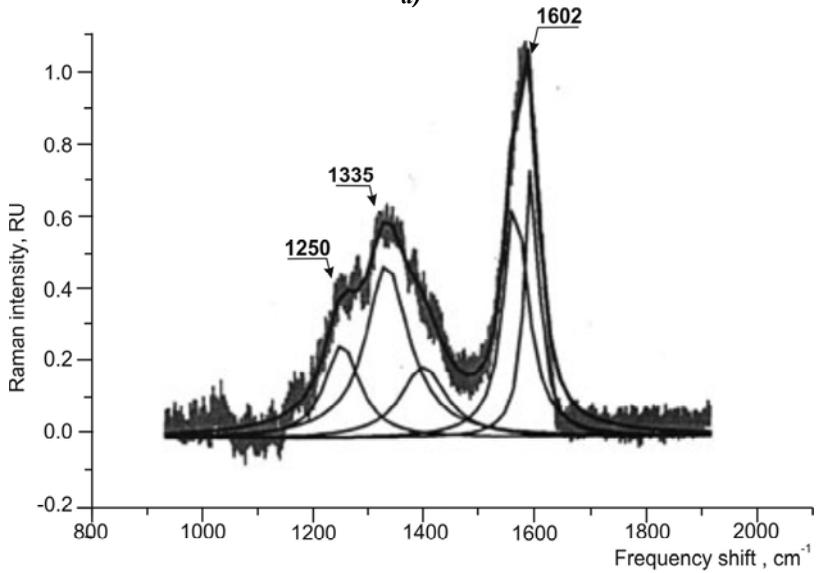
To confirm the data obtained with NMR spectroscopy, additional research with spectroscopy of combination scattering was conducted by the scientists of the Institute for Physics of Mining Processes of the National Academy of Science of Ukraine. Coal samples taken from seam h6 were studied with solid state laser Eurolase: wave length $0.473\ \mu\text{m}$, radiation output power 17 MW. Each spectrum was registered for 1 minute. Processing of the obtained data showed that in CS-spectra of coal samples from outburst-prone zone, Fig. 8a, (compared to samples from safe zones, Fig. 8b), in the region of D-band appeared additional bands with frequency shift values $\sim 1190\ \text{cm}^{-1}$ and $1430\ \text{cm}^{-1}$. CS-spectrum band of outburst-prone coal in the region of frequency shift $1160\text{-}1190\ \text{cm}^{-1}$ is related to fluctuations of C-C bonds in $=\text{C-C}=\text{C}$ groups. The band with the frequency shift value $1436\ \text{cm}^{-1}$ results from the appearance of $\equiv\text{C-C}\equiv$ bonds (polyene chains) [34, 35].

CS-spectra of outburst-prone coal comprise 6 components (Fig. 8a), whereas CS-spectra of coal from safe zones (Fig. 8b) not only seam h6, but also coals of carbonization ranks up to anthracite comprise maximum 5 lines. After outburst, CS-spectra contain 2 components (Fig. 8c). A similar pattern is typical for the study of coal treated by thermal field, which may be the result of one and the same mechanism of physical and chemical transformations and structural conversions in coals.

Maximum of G-band moves from frequency shift value $1612\ \text{cm}^{-1}$ to $1589\ \text{cm}^{-1}$, which is related to destruction of chain fragments with CH-groups. For CS-spectrum of coal from safe zone, frequency shift value of G-band is $1602\ \text{cm}^{-1}$.



a)



b)

Fig. 8. Combination scattering spectra of fat coal (II) from Skochinskogo mine: a– outburst-prone zone; b – safe zone; continued on next page

Continue of Fig. 8

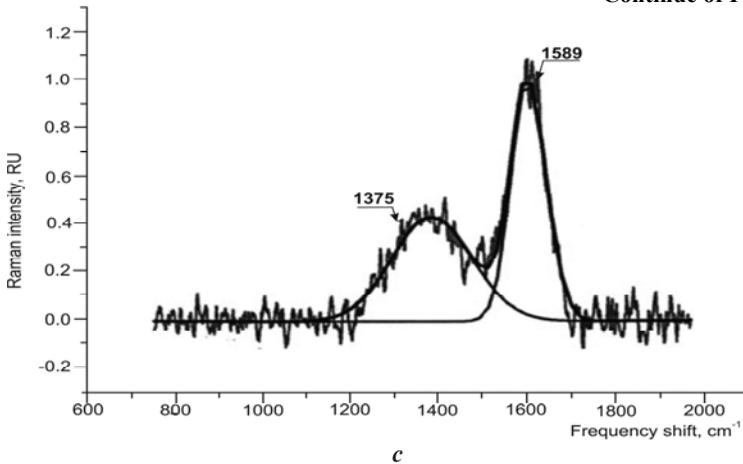


Fig. 8. Combination scattering spectra of fat coal (II) from Skochinskogo mine:
c – after outburst

The change in CS-spectra of outburst-prone and ejected coals confirms the conclusion inferred earlier – that $-\text{CH}_3$ groups increase in outburst-prone zones and that coal organic mass undergoes further destruction during outburst due to reduction of $=\text{CH}-\text{CH}=\text{}$ and $-\text{CH}_3$ groups.

Results of numerical modelling [24, 30] testify that carbon chains stability is determined by the action of excessive external electric charges and the number of carbon atoms in the chain.

Generalization of data allows to assume that during the outburst, methane is formed mainly at the expense of hydrogen in $=\text{CH}$ groups (sp^2 -hybridization) and $-\text{CH}_3$ groups (sp^3 -hybridization). The prerequisite for intense methane generation is achievement of the critical level of $-\text{CH}_3$ groups content and 4:1 ratio of hydrogen atoms in the composition of $=\text{CH}$ and $-\text{CH}_3$ groups. Ye.V. Ulianova believes this to ensure intense process development in the outburst zone.

Conclusions

The complex physical research of coals taken from different zones of outburst-prone coal seams and coals treated by weak

electric field resulted in the following conclusions.

1. The obtained nuclear magnetic resonance spectra (NMR ^1H) demonstrate a significant rise in the methane desorption time. Comparing desorption pattern of original coal samples and those treated by electric field, we can state that gas saturation of the latter decreases only by few percent, while the speed of methane generation rises considerably: three times – for gas coal (I), and 1.7 times for fat coal (II). Drop in gas saturation of treated coal samples is related to additional gas generation by coal organic mass which underwent electrically induced destructurization. The similar results were obtained from the study of coal samples taken from seams in hazardous zones, i.e. coals with destroyed microstructure. Rising coal temperature from 293 to 323 K while treating by electric field significantly stimulates the process of coal destructurization.

2. Mass loss of gas saturated samples due to methane desorption was studied at 363 K using gravimetric technique. It was established that mass loss rate is higher for original samples than for coals treated by electric field. The experiments' weak point was heating at one temperature value. Besides, the role of coal internal specific surface value was not defined.

3. Coal samples taken from the safe zone of outburst-prone seams demonstrated higher chemical activity after crushing than that of original coals. EPR allowed to establish that the density of paramagnetic centres increased from $N = (5-7) \cdot 10^{18}$ PMC/g to $(0.9-3) \cdot 10^{19}$ PMC/g. Additional treatment by electric field results in the growth in PMC concentration to $(3.4-5) \cdot 10^{19}$ PMC/g. It can be inferred from EPR lines analysis of coal from outburst-prone and safe zones that broadening of EPR line for outburst-prone coal is related to free radicals and is described by Lorentz equation. EPR line broadens in cases of mechanical, thermal and electrical destruction of coal, mainly because of thermal mechanism of chemical bonds breaking. Stability of bonds can be affected by high content of impurities in coals.

4. After mechanical and electrical treatment, the coal samples acquired electret potential difference (U_{EP}), which was studied according to compensation procedure. The value of induced electrochemical activity A_i for ejected fat coal (II) (Zasyadko mine, Ukraine) was $4.3 \cdot 10^{-5}$, from the safe zone – $0.7 \cdot 10^{-2}$.

The obtained dependences of electret potential U_{EP} change on time prove superslow relaxation of electret state induced by weak field. Analysis of coals treated and not treated by electric field (safe, prone to outburst and ejected) allowed to establish that the time of electret state relaxation increases with the growth of structural defects concentration (in crystallites and coal polymer matrix) and electric charges. Consequently, reserve of energy stored in defects increases in coal nanostructure.

Relaxation time of electret state for different coals was from $8 \cdot 10^4$ to $2 \cdot 10^6$ sec, which is orders of magnitude slower than Maxwell relaxation (~ 30 sec). The defects concentration rises as a result of coal mechanical crushing and additional treatment by electrical field. Comparative analysis of electret state for coals from different zones of inexplusive and outburst-prone seams and coals treated by electric field proves similar characteristics of coals especially from outburst-prone zones.

5. Destructive processes in coals are accompanied by the growth in smaller fractions share and, consequently, the increase in internal specific surface, the average particles' size and their share decreasing by 3-8%. The amount of the smallest fraction increases, and respectively decreases the amount of the biggest fraction.

6. Diffractograms prove that the structure "amorphism" degree increases, which is confirmed by the data about the distribution of particles by size and the results of the research into electronic paramagnetic resonance, which is the evidence of growing concentration of paramagnetic centres. It is known that ejected coal has a much bigger sorption surface than the unthrown coal from the same seam. The increase in coal internal specific surface may be the result of shear deformations, chemical reactions and the impact of electric fields of weak density.

7. Infrared spectra of coal treated in electric field are similar to regularities established in the study of coals taken from different parts of outburst-prone seams. Destruction of bridge aliphatic chains is related to the decrease in optical density of bands 2920 and 2860 cm^{-1} , corresponding to valent and deformation fluctuations of C-H bonds. These bonds refer to the structures containing CH_2 and CH_3 -groups and can be determined by the bands decrease $3000 \dots 3100 \text{ cm}^{-1}$ in aromatic hydrocarbons.

8. Using methods of nuclear magnetic resonance, cross-polarization, spectral combination scattering of outburst-prone and ejected coal, we established the effects of CH_3 groups growth in outburst-prone zones and continuing deconstructurization of coal organic mass after the outburst. This effect may be related to the lost stability of carbon and hydrocarbon chains caused by the rise of electret potential and its further relaxation. Numerical modelling proved that the action of external electric charges can lead to: first – bond breaking, and second - when the number of carbon atoms in the chain is reduced to the critical level, it results in arbitrary breaking of the chain. Thus, the obtained results indirectly confirm the hypothesis about mechanical and electrical stimulation of coal organic mass transition into gas.

The prerequisite for intense methane generation is achievement of the critical level of $-\text{CH}_3$ groups content and 4:1 ratio of hydrogen atoms in the composition of $=\text{CH}$ and $-\text{CH}_3$ groups, which is likely to ensure high intensity of phase transitions in coal organic mass.

9. It is demonstrated that destructive processes in coal organic mass may be initiated by mechanical and electric impact. Destabilization and destruction of chemical bonds under the electrical impact in the process of methane generation are most pronounced. Hence the electric field is such a factor that can participate in and produce a strong influence on the process of forming outburst-prone properties of coals.

Final considerations

To sum up, it can be assumed that natural generation and emission of gas in coals is mostly related to destruction of the coal solid phase. Among the reasons are the processes of complex deformation, electric field impact, or the influence of these two factors complemented by simultaneous thermal activation.

Gas generation in coals during electric field treatment is a low-temperature process. Similar results can be achieved by heating coal to the temperature 460 K and higher. The process of gas generation in electrically stimulated coals can develop for several reasons: breaking of the weakest chemical bonds in carbon and hydrocarbon chains; complete destruction of chains when the number of atoms in them is

lower than the critical level of the chain stability, formation of hydrocarbon molecules with unpaired electrons with subsequent formation of gas stable molecules; desorption of chemisorbed molecules as a result of inducing their bonds with inner surface by passing currents; graphene bonds breaking leads to the formation of an unstable chain which breaks down yielding atomic carbon with further bonding with oxygen and hydrogen. High rate of chemical reactions can be explained by the catalyzing impact of the surface-active centres.

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ENSURING SAFETY AND PROTECTION OF RESOURCE- SAVING TRIGENERATION OF COAL DEPOSITS GASES OF MINE AND METALLURGICAL PRODUCTION

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Abstract

The idea of the work is to apply the established explosion safety criteria, developed control algorithms, parameters, structural and functional schemes and software and technical management to fulfil the requirements of labor protection, explosion safety and reduce emissions of toxic and greenhouse gases in resource-saving steam turbine gas trigeneration of coal deposits, mines and metallurgical productions.

The object of the research is the processes of increasing safety, labor protection and efficiency of steam-turbine gas trieneration of coal deposits, mines and metallurgical industries.

The subject of research is the regularities of safe, accident-free and efficient operation of steam turbine gas trigeneration equipment of coal deposits, mines and metallurgical industries.

Methods of research and analysis of the results of scientific research on the explosiveness of methane-air mixtures and the reduction of emissions of toxic substances during their combustion, the method of signature mathematics for solving boundary value problems in information-control systems, the method of heuristic and constructive search for new solutions, the method of experimental research in industrial conditions.

The aim of the work is the development of scientific and technical fundamentals of labor protection, explosion safety, efficiency and reduction of environmental safety of a single information-dynamic resource-saving process of steam turbine gas trigeneration of coal deposits, as well as mines and metallurgical industries.

The basic results of the research are as follows:

- system-wide solutions, information and software and technical support for the management of energy-efficient gas triggering in metallurgical industries have been developed;
- a structural and technological scheme of a single information-driven resource-saving process for the extraction, transport and utilisation of gases from coal deposits and mines has been developed;
- a criterion has been established for efficient control of the fuel-air ratio in the controlled moderate undercooking regime, which ensures reduction of toxic gas emissions to the premises and to the area of operation of the trigeneration station and also to the emission of greenhouse gases of methane and carbon dioxide into the surrounding atmosphere.

Developed by SDU-P of the Safety Rules for the control of the station "Diya", boilers for the utilisation of gases from the coal mines of SE «Toretsugol».

1. INTRODUCTION

The rules for the design and operation of steam and hot water boilers set standard parameters that are mandatory for monitoring and ensuring their safe operation.

Inspection and analysis of the operation of boilers at many trigeneration stations showed that the control of the supply of gaseous fuels, total air and the amount of vacuum in the furnace is, as a rule, carried out manually. In this case, if the steam flow rate is changed, the optimum air / gas ratio ensuring complete combustion of the fuel is not maintained, and excessive heat with heated excess nitrogen would not be emitted into the atmosphere, and no toxic nitrogen oxides and carbon and prevent their entry into the premises of the CHP due to automatic maintenance in the furnace of the required range of pressure and overpressure.

Thus, ensuring the safety and protection of labor on the basis of informatization of a single resource-saving technology for the extraction, transportation and utilisation of hydrocarbon gases from coal deposits, mines and metallurgical industries requires in-depth development.

Achievement of the goals set in this work should be ensured by high-quality process management with the following: automatic adjustment of technological parameters; operative presentation of information to service personnel on the status of the technological

process and equipment; high accuracy of measurement of technological parameters; monitoring the actions of operators, managing the technological process, archiving the parameters of the technological process. Ensuring the safety of the technological process of gas trigeneration should be carried out by: developing information, mathematical and software management stations combining the parameters of gas supply and conversion, consumption of their energy in space and time into a single technological complex.

2. MAIN TEXT

2.1. Selection of a reliable and efficient type of boiler for a trigeneration station

In Ukraine, the most widely used are the trigeneration stations of TPP PVS in metallurgy. The blast gases produced in metallurgical production are poisonous and explosive, and they have large energy reserves that require their useful use, rather than flaring. For their utilisation in metallurgy the serial production, reliable in operation, multi-fuel boilers is mastered.

Analysis of the comparative characteristics of the recovered gaseous fuels of coal mines and metallurgical industries shows the following:

- they are all explosive and require special measures to ensure explosion safety during normal operation and especially in the mode of ignition and emergency stops;
- when they are burned, poisonous gases are formed, the most dangerous of them is nitrogen oxide, the reduction of its formation in the existing measures for labor protection and environmental safety in the area of operation of the boiler is usually not controlled and is not given due attention;
- according to the heat of combustion, there is an identity between coal gas and natural gas, between the mine methane of preliminary degassing and coke oven gas, and also between passing degassing methane and blast furnace gas, which indicates the possibility of using a reliable, decades-proven boiler construction in metallurgy for utilisation of coal mine gases .

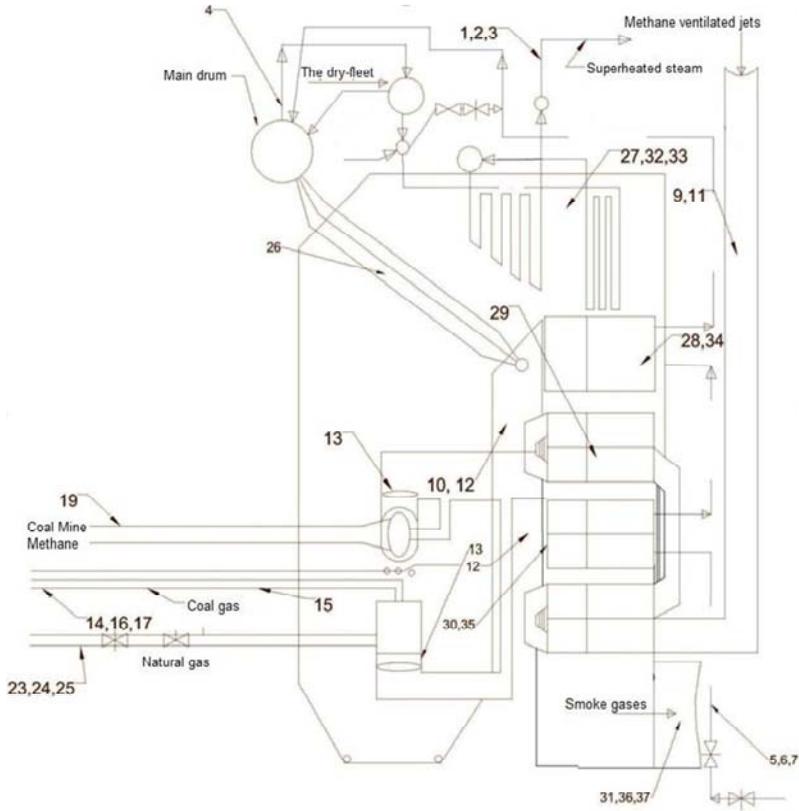


Fig. 1. Design of a multi-fuel boiler

The typical design of the boiler unit TP-150-1, which we have selected for trineration stations, with the indication of the parameter control points, is shown in Fig. 1. Its difference from a single-fuel boiler is that there is a lower stage of burners in the furnace for which natural or coal gas can be supplied with high heat of combustion and mine methane or coke oven gas with an average calorific value. And also there is an upper tier of burners for which blast furnace gas or mine methane is fed, with low heat of combustion.

Boiler TP-150-1 one-drum, vertical-water-tube with natural circulation. The boiler unit has a U-shaped configuration and is

equipped with a drum that is moved out of the heated zone. The combustion chamber of the boiler is designed for burning natural, coke, blast furnace, coal gas and mine methane, separately and in a mixture. The boiler is equipped with a coil-type superheater of mixed type with a corridor arrangement of pipes. To regulate the superheated steam temperature, the boiler is equipped with a surface coil reboiler. In the lower flue gas duct of the boiler there is a two-stage serpentine, installed in a section with an air heater, a water economizer of a boiling type. Air heater - two-stage, tubular, four-way. The total surface area of the heating surface is 9460 m².

The boiler capacity is 150 t / h, the working pressure is 34 kgf / cm², the superheated steam temperature is -420 ° C.

2.2. Information and technical support of safety, labor protection and efficiency of gas triggering in metallurgical industries

Since blast furnace and coke oven gases are explosive, they are poisonous by themselves and when burned in a furnace they form toxic oxides of nitrogen and carbon, we pay close attention to the issues of ensuring safety and labor protection during their disposal. For this, system-wide, software and hardware solutions and the experimental model of the safe control station "DIYA" was manufactured. Figure 2 shows the complex of technical facilities of the station.

The station is designed to control and control the boiler, namely:

- 1) regulation of vapour pressure in the steam receptacle by changing the supply of hydrocarbon gas;
- 2) regulation of the ratio of air to the left / gas with correction for O₂ concentration in flue gases;
- 3) regulation of the ratio of air to the right / gas with correction for concentration O₂ in flue gases;
- 4) regulation of rarefaction in the furnace on the left with correction according to the criteria of labor protection and safety;
- 5) regulation of rarefaction in the furnace on the right with correction according to the criteria of labor protection and safety;
- 6) regulation of the level in the boiler drum with supply of feed;

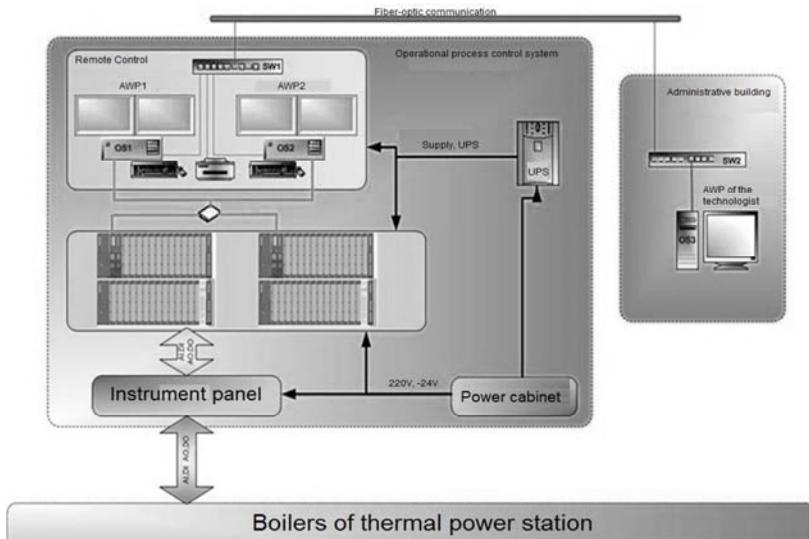


Fig. 2. Complex of technical facilities of the control station "DIYA"

- 7) water - regulation of the salt content of the boiler water;
- 8) monitoring the operation of the protection parameters;
- 9) ensuring the operation of the process equipment without the constant presence of operating personnel in the area where the equipment is located;
- 10) and on the technical and economic performance of the boiler, as well as its transfer to the information network.

Experimental studies in the industrial conditions of the developed mathematical, informational, software and structural-functional scheme were carried out at the existing boiler unit BKZ-220-100F # 2 of the PJSC "DniproAzot" TPP.

In accordance with the certificate and the protocol, the results of experimental studies in the industrial conditions of the control station and emergency protection of the boiler unit No. 2 BKZ-220-100F ("DIYA") are as follows. Within 72 hours of continuous, trouble-free operation of the control station of boiler unit No. 2 BKZ-220-100F performed all the functions specified in the Terms of Reference for the development of the automated process control system for the boiler unit BKZ-220-100F # 2 of CHPP from 11.10.13. It is

established, that emergency and normal boiler shutdown occurs when the set values of the settings are achieved with the correct display of mnemonic diagrams, graphs and in the event archive on the personnel. Based on the results of experimental studies in industrial conditions, automated process control system of boiler units of the thermal power station is allowed to industrial tests in the equipment of the PJSC "DniproAzot".

Analytical and experimental studies on the same boiler set the dependences of the air pressure drop on the air heaters and the oxygen concentration in the flue gases behind the superheater from the steam load, presented in Fig. 3

Based on the results of the studies presented in Fig. 3, the criterion of efficiency and ecological safety of controlling the air / fuel ratio according to the oxygen concentration in the flue gases measured at the superheater, calculated according to the expression (1):

$$K_{\text{э62}} = \frac{3,0 - (F_{\text{II}} - 100) \cdot 0,015 - K_{O_2}}{2,5 - (F_{\text{II}} - 100) \cdot 0,015} \cdot 20^{0,5[1 - \text{Sign}(2,5 - (F_{\text{II}} - 100) \cdot 0,015 - K_{O_2})]}, \quad (1)$$

where: $K_{\text{э62}}$ - the criterion of efficiency and ecological safety; F_{II} - steam flow, tons / h; K_{O_2} is the concentration of oxygen in the flue gas, measured at the superheater, %.

As a result of industrial tests with the help of the "DIA" station on boiler No. 2, regularities were determined on the basis of which a safety criterion was established to prevent emergency shutdown of the boiler due to the extinction of the torch and the labor protection criteria for the deposition of toxic gases into the CHP room from the discharge and overpressure in the boiler furnace. The values of the criteria should be calculated from the expressions (2) and (3) obtained by the methods of signature mathematics:

$$Kom = \frac{4 - Pm}{4} 20^{0,5[1 - \text{sign}(4 - Pm)]}, \quad (2)$$

$$K\sigma = \frac{4 - |Pm|}{4} 20^{0,5[1 - \text{sign}(4 - |Pm|)]}. \quad (3)$$

The regularities, on the basis of which the criteria of labor protection are established, are to prevent the formation of toxic gases

in the furnace and their entry into the premises of the TPP, as well as the safety criterion for the extinction of the torch in the furnace: to prevent an emergency stop boiler due to the extinction of the torch and the penetration of poisonous gases into the TPP plant, the discharge in the furnace of the boiler must be in the range according to the formulas of the labor protection criteria and security.

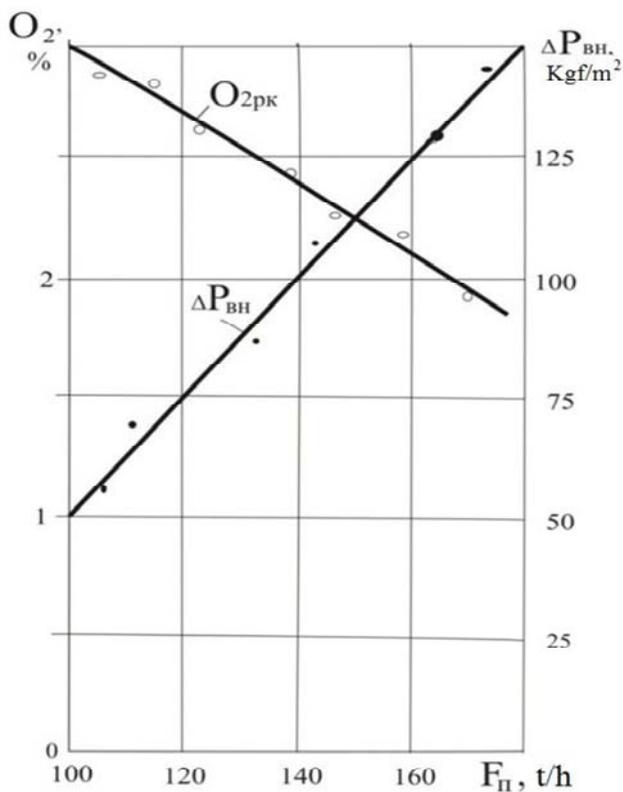


Fig. 3.-Dependences of oxygen concentration in flue gases and pressure drop on the air heater on steam consumption

Based on the research of the BKZ-220-100F boiler and multi-fuel boiler of TP-150-1 type, we patented the [pat] method of "Automatic control, monitoring, protection and signalisation of the boiler unit",

developed a structural and functional scheme for controlling a multi-fuel boiler in which in addition to pre-emptive actions on all control loops that are ensuring the stability of the boiler's dynamics, there is a self-adjustment unit for control actions in which, when the steam flow is changed, the task is to compensate for the change of energy due to a cheap blast furnace gas with a maximum economy of natural gas. The information management method is based on the task of improving the information control for monitoring, signalling and emergency protection of a multi-fuel boiler by measuring the boiler's output and generating control signals in the controller for experimentally measured energy-efficient dependences of air consumption, O₂ oxygen content and carbon monoxide CO in the smoke path on the boiler output maximum temperature in the furnace and the maximum ratio of productivity to the fuel consumption, that is leading to optimise the fuel combustion in boilers and furnaces and will provide fuel savings of electric power consumed by electric motors and blower fans exhausters, and reduced harmful emissions and greenhouse gases.

The task is solved by the fact that in the method of information control, monitoring, protection and alarming of the boiler unit by continuous measurement of fuel, air, carbon monoxide and oxygen in the smoke path, using sensors installed on the boiler, fuel pressure, air pressure, discharges in the smoke path, temperature inside the boiler, which are introduced into the controller, which generates signals to control units in the form of frequency converters for smooth control of the fan and a smoke exhaust fan, maintaining the specified content of carbon monoxide CO and oxygen O₂ in the flue gases and the maximum temperature in the furnace, additionally measure and enter into the controller the values of signals of the actual performance of the boiler, and with the help of the set points, the tasks of discharging in the furnace and the boiler output are entered into the controller, and the energy-efficient dependences of air consumption, oxygen content O₂ and carbon monoxide CO in the smoke path on the boiler output have also been experimentally measured. At the same time the controller generates signals to the actuator of the regulating organ for supplying fuel and to control units that support the air flow rates, oxygen content of O₂ and carbon monoxide CO in the smoke path, corresponding to the

experimentally removed energy-efficient dependencies of these parameters on the boiler output.

The controller can also monitor, signal and shut down the boiler when the emergency fuel pressure and air pressure.

Measurement of the performance and the formation of control signals in the controller based on the energy-efficient dependences of the air, the oxygen content and carbon monoxide content in the smoke path from the efficiency of the boiler unit is advantageously distinguishes the proposed method, since the boiler works with the maximum efficiency in the entire range of its efficiency.

In addition, the developed scheme manages smoke exhausters in accordance with the above stated criteria for labor protection and boiler safety when air pressures and discharges in the furnace are reached, which are regulated by these criteria.

To reduce the emission of toxic gases (NO and CO) into the sanitary zone of the station and into the premises of the TPP, the developed scheme provides for the management of the criterion of efficiency and environmental safety (expression 1), which maintains a controlled moderate non-combustion of fuel in the range of the excess air factor from 1.04 to 1.1.

On the basis of this patent and the results of the above industrial studies of the control station "DIYA", a working draft of the automated process control system of the boiler unit No. 9 of the TPP was developed and approved by the management of PJSC "Dneprovsky Metallurgical Plant".

A manual on safe management of a boiler for the utilisation of blast furnace and coke oven gases during start-up, normal operation, planned and emergency stops with the provision of operative information to the operator on the monitors of the operator station, an example of which is presented on the given start-up mimic diagram, is developed.

A manual on safe management of a boiler for the utilisation of blast furnace and coke oven gases during start-up, normal operation, planned and emergency stops is developed with the provision of operative information to the operator on the operator station monitors, an example of which is presented on the given start-up mimic diagram Fig.3. Calculation of the expected economical effect due to the primary combustion of blast furnace gas, while saving

natural gas showed that the payback period of the project implementation costs will not exceed 7 months.

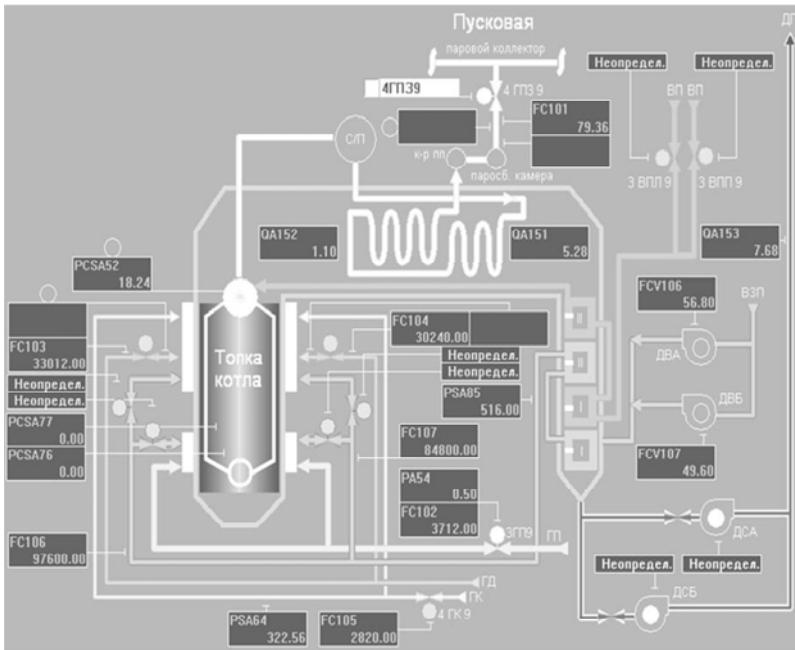


Fig. 3. Starting mnemonics of a multi-fuel boiler

2.3. Steam Turbine Trigenation of Gases of Coal Deposits and Mines

The structural and functional scheme of a tricogene generation station for the utilisation of coal mine methane is shown in Fig. 4. At the entrance of the multi-fuel unit, mine methane is produced, extracted by underground degassing wells with a low methane concentration of 10-40% and methane of high concentration, after a two-stage turbo-expander. For the utilisation of methane ventilation jets of the mine with a percentage content in the air up to 1% of CH₄, it must be heated by air heaters due to the heat of the exhaust flue gases and fed instead of air to a multi-fuel boiler.

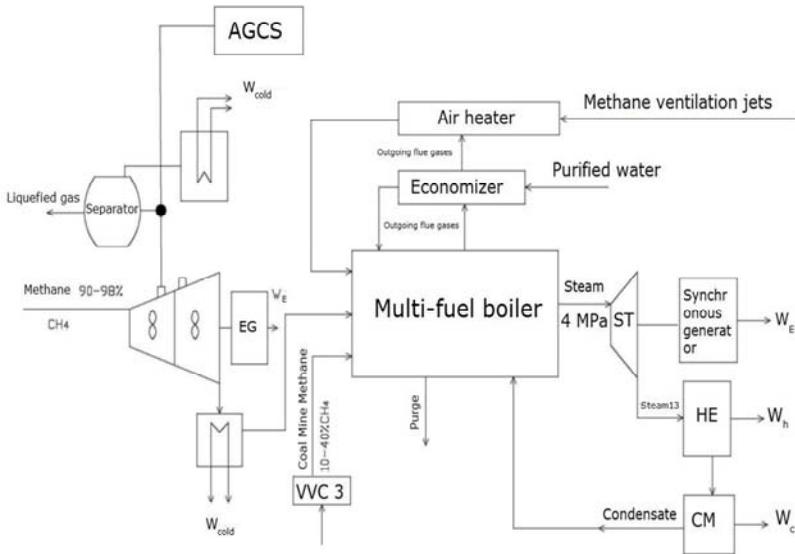


Fig. 4. Structural and functional scheme of a tricogeneration station for gases utilisation from coal deposits and mines

From the output of the boiler, high-pressure steam - 4 MPa is fed to the input of a steam turbine on the shaft of which a synchronous generator generating electricity is installed. Low-pressure steam with the output of the steam turbine is fed to the heat exchanger, where the heat energy W_T is generated for heating the supernumerary buildings. The heat leaving the heat exchanger is fed to the chiller (XM). The output is a cold W_X , sent to underground mining workings. The steam condensate is returned to the boiler after the heat exchanger and the refrigerating machine. The boiler input is also supplied with purified water from the chemical water treatment plant (HVO), heated in the economiser by the heat of the outgoing flue gases. Gas with a high concentration of methane after the first stage of the turbo-expander is fed to a separator that cleans it of liquefied propane-butane. After the separator, the gas is sent to the station for its preparation for transmission to the gas transportation system (GTS) of Ukraine. At the output of the training station, an automated system for the commercial recording of coal gas (ASKU UG) is

installed. The generated cold after the first and second stages of the turbo-expander is also sent to the mine to normalise the air temperature.

To develop a structural and functional scheme for the control system of a multi-fuel boiler of a tricogene generation station for utilisation. gases of coal mines on the basis of research results [mon] developed a structural scheme for the extraction and utilisation of hydrocarbon gas coal deposits and mines (Fig. 5). In addition to the existing normalised input parameters, additional input parameters and established safety and labor protection criteria, calculated on the basis of new parameters, that form control actions, they are presented in this scheme in addition to the existing normalised input parameters. providing non-accident and labor protection during operation of the boiler.

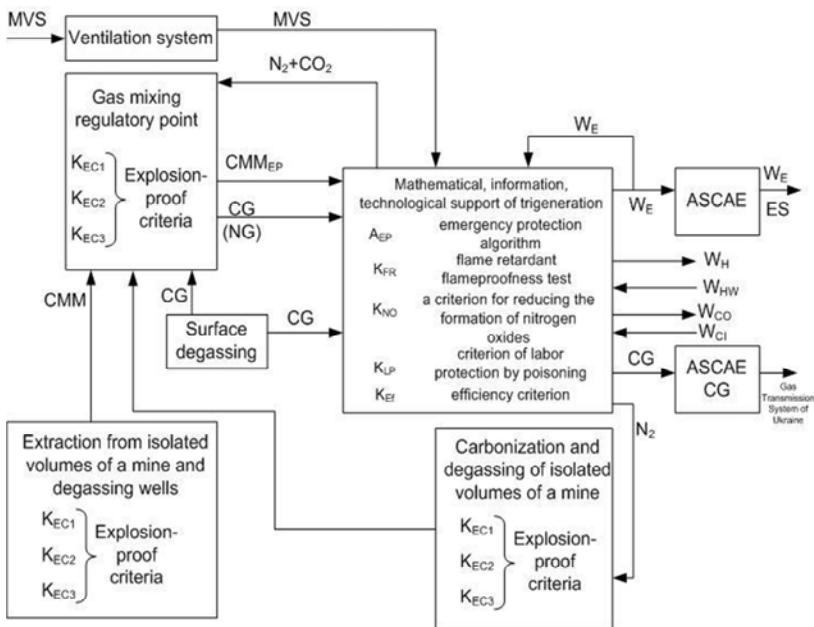


Fig. 5. A schematic diagram of the extraction and utilisation of hydrocarbon gases from coal deposits and mines

Distinctive advantages of this scheme are: the presence of the gas-mixing regulating point of the GSRP, in which, using the established explosion safety criterion, all mined methane is disposed of, including explosive concentrations; burning in the furnace part of methane ventilation jets; the provision of pre-emptive actions at the same time is ensured stable for the air flow regulators, the furnace discharge regulators, the gas flow rate and the water supply regulators; the existence of criteria for environmental safety and energy efficiency makes it possible to reduce the formation in the furnace of toxic gases (NO and CO) without reducing the efficiency of the boiler; the controller for analysis and self-adjustment of the boiler provides control functions that ensure trouble-free operation due to the flame extinguishing and labor protection requirements for gas contamination of the station premises with toxic gases; PC MTC with the help of M1 and M2 monitors provides the boiler operator with all operational information necessary to ensure safety start, in normal operation, with scheduled and emergency stops.

Based on the above research results it was developed «Safety rules of control station «DIYA» for boiler units of TPP at the disposal of «Toretskvugillya» gases.

Safety rules terms of control station DIYA of the TPP boiler unit of mine gases utilisation at "Toretskvugillya" set requirements for the design, complex software and hardware, manufacturing, installation, adjustment, maintenance and operation of the measurement station, control, signalling, information management and emergency protection of the TPP boiler units when utilising the gases from the mines of the "Toretskugol" (hereinafter referred to as Station DIYA).

The rules apply to DIYA stations, which are intended for controlling the boiler units of the TPP, which utilise the gases of the mines of SE "Toretskugol".

Construction of boilers installed in TPP plants for utilisation of coal mine gas should be typical, tested at least 5 years in the steel industry for the recycling of blast furnace and coke oven gas (hereinafter referred boilers).

Boilers must be manufactured and fit DNAOP 0.00-1.08-94 for regulation the structure and safe operation of steam and hot water boilers (Fig. 1), as well as stations equipped with 'DIYA'. It is

forbidden to operate TPP recycling gas from mines that is not equipped with DIYA stations.

Additional requirements for the development of a project for the safe and energy-efficient disposal of gas from the coal mines of the Toretskugol subsidiary.

Methane explosions that are dangerous for explosions include cogeneration stations (CS) and thermal power plants (TPPs) that is utilising coal gas, mine methane and methane ventilation jets.

Parameters of methods and means of information control, signalling, control and emergency protection shall be established in accordance with the lower explosion limit of methane equal to 3.5% vol. and the limiting oxygen concentration below 7% vol.

Projects for the construction and reconstruction of utilising the coal mines gases stations should include «Temporal management for the safe control of the station «Diya» by boiler units of TPP for the utilisation of gas from the mines of the «DP Torytskvugillya»» and must be coordinated with the Commission on degassing and combating gas-dynamic phenomena in coal mines of Ukraine.

In the projects of the station "DIA" should be provided sections: mathematical support, information support, system-wide solutions, software and hardware support and organisational support for putting the station into operation

Conclusions

1. The selection of a reliable and efficient multi-fuel boiler, most widely used at TPP PVS in metallurgy for a trigeneration station for the utilisation of gases from coal deposits and mines is justified.

2. The control station "Diya" was developed and tested in industrial operation at the cogeneration TPP of PJSC "DniproAzot" as part of the process control system by boiler units.

3. As a result of experimental studies in industrial conditions at №2 boiler BKZ-220-100F set criterion efficiency and environmental safety for controlling air-fuel ratio by the oxygen gas concentration in the flue gas as measured superheater.

4. A labor safety criterion has been established to prevent the gasification of the station premises with toxic gases (NO_x and CO) to control the excess pressure in the furnace.

5. A safety criterion is established for the emergency shutdown of the boiler unit for extinguishing the torch on the burners to control the discharge in the furnace.

6. A detailed design of an automated control process system for steam production by boiler unit No. 9 of TETS-PVS PJSC "Dneprovsky Metallurgical Plant" for the utilisation of blast furnace and coke oven gases was developed.

7. The structural-functional scheme of the trikgeneration station for utilisation gases of coal deposits and mines has been developed.

8. The structural scheme for the extraction and utilisation of hydrocarbon gases from coal deposits and mines is developed. The algorithm for the provision of explosion protection and anti-damage protection at all stages of a single information resource-running process from extraction to trigeneration is presented.

9. Safety rules for the control of the "DIYA" station at boiler units of the TPP at the disposal of the gases of the mines of the State Enterprise "Toretskvugillya" have been developed.

10. A manual on the safety of operation of automated boiler units for the utilisation of methane mixtures of a coal mine was developed.

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RESOURCE-SAVING METHODS OF POLYMINERAL ROCKS EXPLOSIVE DESTRUCTION

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Introduction. Polymineral rocks, which are mainly rocks of magmatic genesis (granites, syenites, diabases, gabbro-diabases and gabbros), are raw materials for crushed stone. Crushed stone – the main component for the production of non-metallic building materials (reinforced concrete products, ballast, etc.). Magmatic rocks because of their inherent high strength are developed in an explosive way, and the cost of funds for drilling and blasting is at least 25-30 % of the cost of crushed stone. The technology of the rock mass preparation for the production of rubble from polymineral rocks depends to a large extent on their physical and mechanical properties, structural defects (microjoints) and spatial orientation of rock-forming minerals (microstructures) and on the fracture of the massif (macrostructure) being fractured. The microstructure of the rock influences the character of explosive destruction only at the contact of the “explosive-rock”, in particular, the volume of the re-milled rock (substandard fractions) near the borehole charge, but its influence is also very significant at the stage of the rock mass mechanical crushing. As a result, it is the microstructure of the polymineral medium that influences the strength of concretes in which crushed stone made of igneous rocks is used as the filler.

The *macrostructure* of magmatic polymineral rock, i.e. the frequency of macrojoints location, their morphology and spatial position have a decisive influence on the character of the magmatic (intrusive) anisotropic massif destruction at the stage of the quasistatic action of the explosion. Typically, massifs of intrusive rocks are dissected by three macro-joints systems with different

morphological features. Two long vertical mutually perpendicular fracture systems, one of which is characterized by the opening between joints (gaping joints) of the order of 0.5-1.0 cm, and the other by tight fractures, form a so-called fracture-cracked grid on the massif surface. The main feature of intrusive bodies with a similar fracture tectonics is alternating with intervals of 40-50 m intensely fractured areas of the rock with virtually monolithic rocks. In addition, intrusive massifs on the upper horizons have a well-developed system of horizontal joints. As a result, a horizontal system and two mutually perpendicular systems with vertical joints that dissect the intrusive massif lead to the appearance of so-called parallelepipedal partings in it.

In order to achieve a good quality of explosive preparation of the rock mass in the design of the parameters for drilling and blasting operations (the specific flow of explosives, the grid geometry of borehole charges and the breaking direction), it is necessary to take into account the features of the *macrostructure* and the *microstructure* of such rocks. The macrostructure erodible explosion intrusion of massif can be accounted for by using fracturing plans usually forecast. With regard to accounting for microstructure, i.e. spatial orientation of rock-forming minerals, intergranular and intragranular microjoints, which, in fact, leads to anisotropy of the physical and mechanical properties of rocks and causes the nature of the destruction of the polymineral medium by the explosion at the contact “explosive-rock”, then this problem is solved by conducting special studies involving methods microstructural analysis. Microstructural analysis involves the selection of oriented samples of rock, the oriented ore production and polished sections, the establishment in them of rock-forming minerals and microjoints spatial orientation, and the drawing up of a qualified conclusion about the nature of the microstructure, usually reflected in microstructural (oriented) diagrams.

Exhaustive data on the nature of macro- and microstructures make it possible to develop optimal parameters for drilling-blasting operations for a particular massif of polymineral rocks (the size of the blast hole network and its location on the fractured rock block, the charge commutation circuit and their construction), which allow producing high-quality explosive preparation in the production of

mass explosions rock mass (the minimum yield of re-milled and oversize fractions, qualitative workup of bottom and the absence of exploration joints in the rear part of the massif).

Optimization of drilling and blasting operations, taking into account the macro-fracture of the massif, which is of a regional character for intrusive massifs, as a rule, is not difficult in the presence of predictive fracture plans. It should be noted that accounting microjoints forming minerals to improve explosive destruction is more difficult.

We have established that the volume of re-grinded fractions in the destruction of polymineral rocks by the explosion depends on two factors, namely, the explosive power and the intensity of the microjointing of the rock-forming minerals.

In this regard, a decrease in the volume of re-milled fractions during the destruction of such rocks by the explosion can be achieved by selecting explosives (mainly by their power) and using special designs of borehole charges. With the destruction of strong rocks now, mostly powerful emulsion explosives are used. To reduce the volume of re-milled fractions, these explosives are placed in polyethylene casings with a diameter smaller than the diameter of the borehole. The resulting gap between the charge and the rock reduces the dynamic impact of the explosion on the destroyed polymineral environment. As a result, the volume of finely dispersed particles of rock in contact with explosives decreases. However, due to the elasticity of the polyethylene sheath, it is practically impossible to achieve uniformity of the annular gap between the explosive and the surface of the hole, in addition, under the action of gravity, the charge acquires a sinusoidal shape, the length of the charge column increases and, as a result, the explosive mass increases by 30 % with the estimated mass.

In view of the foregoing, we have developed fundamentally new methods for destroying polymineral rocks with a complex structure, based on the use of combined charges of variable cross-section and their commutation schemes. The developed methods underwent an industrial check on the granite quarry “Sivach”, confined to the northwestern part of the Korsun-Novomirgorod granite pluton (Cherkassy region, Ukraine).

Below are the results of the research.

The purpose of the work is the presentation of new methods of explosive destruction of polymineral magmatic rocks of complex structure, based on the structure of the destroyed massif.

Methods of research. Structural analysis of rock massif and microstructural analysis of rocks destroyed by explosion, determination of rocks strength properties, laboratory and industrial experiments, analysis and generalization of experimental data used in the future to develop new resource-saving methods of polymineral rocks explosive destruction.

Structure of polymineral massifs of a complex structure (on the example of granite field “Sivach”).

The deposit of granites “Sivach” is located in the northwestern part of Korsun-Novomirgorodsky pluton (Fig. 1).

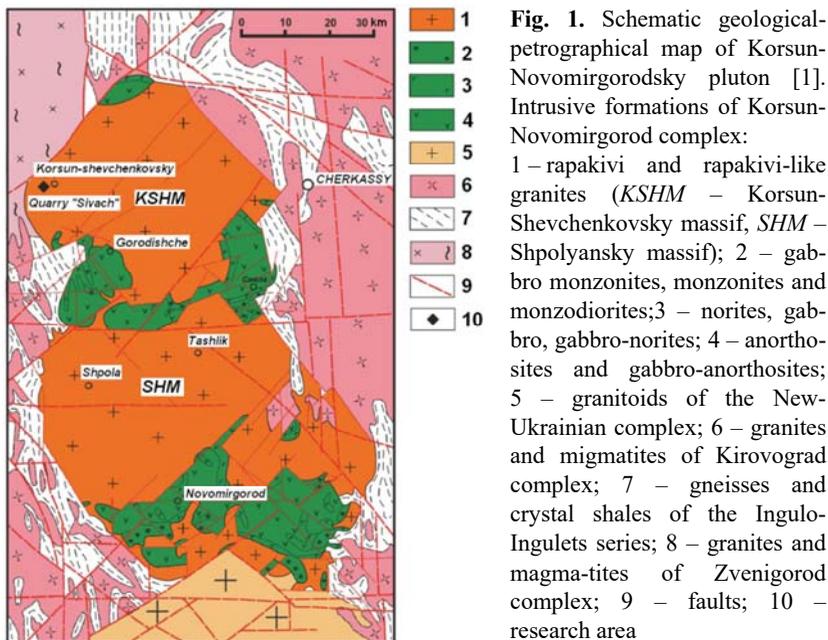


Fig. 1. Schematic geological-petrographical map of Korsun-Novomirgorodsky pluton [1]. Intrusive formations of Korsun-Novomirgorod complex: 1 – rapakivi and rapakivi-like granites (*KSHM* – Korsun-Shevchenkovsky massif, *SHM* – Shpolyansky massif); 2 – gabbro monzonites, monzonites and monzodiorites; 3 – norites, gabbro, gabbro-norites; 4 – anorthosites and gabbro-anorthosites; 5 – granitoids of the New-Ukrainian complex; 6 – granites and migmatites of Kirovograd complex; 7 – gneisses and crystal shales of the Ingulo-Ingulets series; 8 – granites and magma-tites of Zvenigorod complex; 9 – faults; 10 – research area

Structural of polymineral rocks features of magmatic genesis and their influence on the character of the explosion destruction. As already mentioned above, the granite deposit “Sivach” is located in the northwestern part of the Korsun-Novomirgorod pluton. A

characteristic feature of the intrusive formations of the Korsun-Novomirgorod complex is the significant development of hybrid rocks of gabbro-monzonite, monzonite and syenites. In the north, west and southeast of the complex, pluton breaks the plagiogniess of the Ingul-Inhuletic series by the Paleoproterozoic, metamorphosed and migmatized under amphibolite facies. To the east of the pluton widespread autochthonous granites and migmatites of the Kirovograd complex by the Paleoproterozoic. On a small plot along the northwest contact of pluton are developed amphibole gneisses and shale crystals of the Rosin-Tikich series Archean.

When studying the fracture-tectonic structure of the Sivach granite deposit, we established that the mountain massif is intensively dissected by three mutually perpendicular fracture systems (Fig. 2): tightly closed compression joints S , uncovered fracture joints Q and horizontal formation joints L (nomenclature G. Cloos [2]).



Fig. 2. Fragment of the working face of the granite quarry “Sivach”: the light arrows – compression joints S ; dark arrows – horizontal formation joints L

When fracturing massifs are destroyed by the explosion energy, orientation along the slope of the ledge with respect to the directions of the basic systems of vertical joints S and Q is of great importance. As noted above, the vertical joints that break the massif are in series, which leads along the line of slaughter in the quarry there are areas of highly jointed and practically monolithic rocks alternating between them. Such areas or zones of high fissure have a horizontal power of 40-60 m; the distance between the centers of the zones is from 50 to 100 m. In the case of the difference in the slaughter line with the extension of the system of joints S and Q after the explosion, significant exploration joints are formed in the rear of the rock

massif, which reduces the productivity of the operation of mining equipment. On sections of monolithic rocks, when the wrong direction is chosen for the explosive deflection in relation to the spatial position of linearly extended aggregates of mineral grains, it is observed not only an increase in the percentage of the yield of crushed fractions, a decrease in the strength of crushed stone, but also an overstress of the bottom. In addition, the intense macrojoints of mineral grains (especially quartz) with the use of highly explosives leads to the crushing of the rock mass at the contact “explosives-rock”, which is also characteristic for the zones of solidly, closed macrojoints development due to the features of the explosive impulse reflection from their surfaces. All this leads to irreparable losses of mineral raw materials.

Influence of polymineral rock properties on specific energy of destruction at explosive loading. To study the influence of physical and mechanical properties and structural features on the specific energy of the rocks destruction of a complicated structure under their dynamic load, we have developed a method for evaluating their changes in laboratory and field conditions (*Patent No. 95218* [3]).

The study of these changes was carried out on two series of rocks specimens, which were selected on a quarry “Syvach” in the form of a cube with an edge of 350 mm. Description of rock samples, are given in Table 1.

Table 1

Information on samples of rocks selected for the study of physical and mechanical properties and the specific energy of their destruction

Series number	Type of rock	Size samples, mm	Rock characteristics
1	Granite gray is large granular	350×350×350	Ovoid structure of the feldspar, with the intersperses of orthoclase light gray color with small pinkish-brown viscose grains,
2	Granite pink is coarse-grained		Oval structure, formed by large ovoid of feldspar (up to 5 cm in size), mainly orthoclase of pink color

For a detailed analysis of the physic mechanical characteristics of composite polymineral rocks, such as the strength of uniaxial

compression, density, velocity of longitudinal and transverse waves, Poisson's coefficient and Young's modulus, from the selected rock specimens (Table 1), models were developed for testing them according to current State Standards [4, 5]. For each series of tests on a stone carving machine using a diamond disk 250 mm in diameter, 10 models of cubic form with a rib size of 40 ± 2 mm were prepared.

The faces of the samples were treated with grinding powder, with their curvature not exceeding 0.05 mm. A rod screw for two mutually perpendicular faces carried out the control of the face surfaces of the specimens. The deviation of the faces from the parallelism did not exceed ± 0.1 mm. The prepared rock specimens were tested for strength under uniaxial compression using the hydraulic press PR-500 according to the appropriate method [6]. The general view of the module for loading rock samples and the typical diagram of "tension-strain" is shown in Fig. 3. Pressure control on the model was carried out by a pressure gauge, and the removal of transverse values deformation with strain gauges of displacement. The received information of test results in automatic mode was sent to the computer system for processing and construction the "tension-strain diagram".

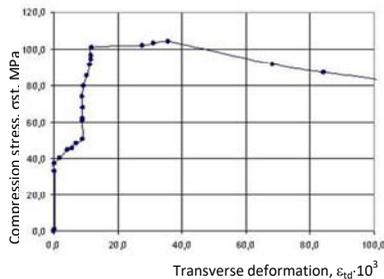


Fig. 3. General view of the PR-500 press unit for testing rock samples for uniaxial compression and a typical "tension-strain" diagram

The test results were entered into a computer database in the form of a table. The strength of the sample was calculated by the formula:

$$\sigma_{ct} = \frac{P_{max}}{F_0}, \text{ MPa}, \quad (1)$$

where $F_0 = a^2$ – the cube face area, cm^2 ; a is the edges size, and the area of cylindrical shape model face is – $F_0 = \pi d^2/4 \approx 0,785d^2$; d – sample diameter, cm .

The study of the rocks acoustic properties (propagation velocity of longitudinal and transverse waves) was carried out on a stand developed by the Institute of Geotechnical Mechanics N.S. Polyakov NAS of Ukraine, which is shown in Fig. 4 [7]. Measurements of the elastic wave's velocity propagation were carried out on three pairs of sample faces, and then their mean value was determined. The average calculated values of the elastic wave's velocities propagation with an accuracy of no more than 10 % and a reliability of 0.95 were achieved by testing at least 10 samples for each type of rock.

Young's modulus according to the measured values of the elastic waves velocities propagation in a rock sample of was determined by the formula:

$$E = \rho V_p^2, \quad (2)$$

where ρ – rock density;

V_p – longitudinal wave velocity.



Fig. 4. Stand for measuring the velocity of elongated and transverse waves propagation

To accelerate and simplify the process of determining the elastic parameters of rock samples, the V. Rantch and G. Cromfouls nomograms, which allow to operatively determine the Poisson's ratio in relation V_p/V_s , where V_s is the transverse wave velocity, were used [7]. The value of the acoustic, physical, and mechanical characteristics of the granite

deposit "Sivach" are given in Table. 2.

Table 2

Basic physical-mechanical characteristics of “Sivach” granites

Type of rock	Density, $\rho \cdot 10^{-3}$, kg/m ³	Compression strength, σ_{cs} , MPa	The extended wave velocity, V_p , m/s	Poisson coefficient, ν	Young's modulus E , MPa
Granite gray coarse	2,7	186,0	4450	0,24	18600
Pink granite coarse	2,8	180,0	4350	0,22	19200

From the prepared sections, transparent petrographical slabs were produced, in which structural features of the polymineral rock (granite) were investigated using a polarization microscope.

Below is a description of microstructures of the investigative rocks.

Granite gray coarse-grained (Fig. 5). *Microscopic description:* this section of gray granite clearly illustrates the secondary changes in plagioclase as a result of auto metasomatism, which is expressed in the replacement of the central particles of the minerals of the pelitomorphic mixture of carbonates, the sericite (finely scaly variety of muscovite), and the minerals of the zoizitum group – the product of the hydrothermal change of the plagioclase.

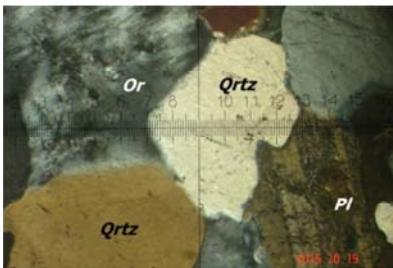


Fig. 5. Microphotography of a granite-gray coarse-grained structure (Sivach quarry, Korsun-Shevchenkovsky, Cherkassy region, transparent section, polarized light, an increase of 150×): *Or* – orthoclase, *Pl* – plagioclase, *Qtz* – quartz

Pink granite is large and medium grain (Fig. 6). The structure of the rock is avoidable, formed by large feldspar ovoids (up to 5 cm in size), mainly orthoclase of pink color, characteristic of rapakivi granites; under a microscope – a hypidiomorphic, granite.

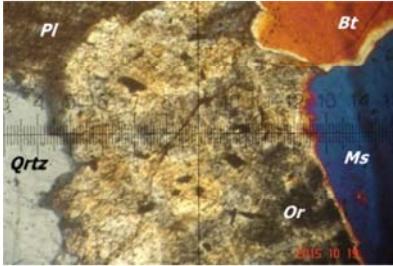


Fig. 6. Microphotography of a granite pink large and medium grain structure (“Sivach” quarry, Korsun-Shevchenkivskyi, Cherkassy region; transparent section, polarized light, increase 150×):
Or – orthoclase, *Pl* – plagioclase, *Qtz* – quartz, *Bt* – biotite, *Ms* – muscovite

Microscopic description. A distinctive feature of this grid is the presence of a large number of directed defects in the form of trace microjoints, which are observed in the field of view of the microscope in the form of strips of gas bubbles (planes of gas-liquid inclusions – the plane of GLI). The sizes of individual bubbles (inclusions) do not exceed 1-2 microns. Plane of GLI does not go beyond the grain of quartz. Plane genesis – jointing of quartz grains in the transition to low temperature modification with a decrease in the volume of high-temperature quartz by 0.86 % (the so-called $\alpha \leftrightarrow \beta$ transition).

Estimation of the specific energy of destruction of polymineral rocks for their dynamic load. The volume of fine particles (dust particles) formed on the contact of “explosive-rock” or “rock breaking tool-rock” under dynamic loading depends mainly on the density of defects in the structure of rock-forming minerals. The main sources of dust in such rocks are minerals, in which the density of structural defects is maximal. In quartz-containing polymineral rocks, such a mineral is quartz, the density of structural defects in which ranges from 2×10^5 to 5×10^3 in one cm^3 [8]. Specific energy of such polymineral rocks destruction for this reason essentially depends on the defectiveness of rock-forming minerals. Data on the specific energy of destruction of polymineral rocks are necessary for the development of new energy- and resource-saving methods of their destruction by means of high intensity dynamic loads (explosion, high-speed impact). Therefore, estimating the specific energy of rocks destruction is an extremely important element for the scientific substantiation of new structures of borehole charges destined for the destruction of hard rock’s with complex structure, as well as for the creation of new technologies for the processing of

mineral raw materials, using a dynamic load for further crushing of the exploded rock mass.

Existing methods for evaluating the structural and energy parameters of rocks under different types of loads include sampling in an massif, their testing in laboratory conditions for the determination of their physical and mechanical characteristics with different loading of rock specimens [4, 5]. Also known is a method for evaluating the structure, energy parameters of the destruction of rocks, which vary with different types of load on models, including drilling holes, core selection, tests to determine their physical and mechanical characteristics and energy indices when changing them under different load conditions, but at least three points per meter of the sample [6].

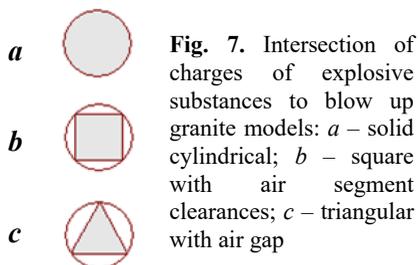
However, the method presented in this paper has disadvantages. In particular, in assessing the physical and mechanical properties, structural changes and energy indices of the rocks destruction selected for research, the spatial position of the collapsing massif and selected samples of rocks (cores, shrubs) and their structural features (spatial position of the linear-plane structure) are not taken into account. The linearity vectors of elongated mineral aggregates, etc., as well as the direction of joints development with different types of loading are not taken into account too. Therefore, in order to improve the method of estimating the specific energy of the destruction of anisotropic rocks under different types of load on models it is expedient to introduce new operations. There are: selection of oriented core from a geological prospecting hole or oriented ore lump from a massif (in a quarry, mine); preparation of several samples from selected cores (ore lump) for their destruction on a copra with the impact of a freely falling load as well as samples for the destruction by explosive; small fractions selection of the destroyed model and its estimation with the optical microscopy method of its destruction character by granulometric characteristics. The specific energy of destruction is then determined by the energy characteristics of crushing models made of polymineral rock. Such a sequence of operations can increase the accuracy, validity and reliability of the data obtained in the course of experimental research, and, consequently, reduces the cost of subversion and increases their efficiency in various methods of the destruction of anisotropic rocks.

Influence of structural features of granites on the specific energy of their destruction under dynamic load. Determination of the specific energy of the granites destruction by different types dynamic loads, that is, the freely falling cargo impact and the explosion of longitudinal charges with a different shape of the cross section was carried out according to our special experimental research methodology, which allowed in the laboratory and field conditions to evaluate the nature and specific energy of destruction of polymineral rock.

Granite blocks, taken outside the zone of influence of dynamic (explosive) loads, were sprayed with a diamond disk on a cubic shape (with a rib size of 40 ± 2 mm). In addition, cubic samples with a 5 cm rib size were obtained for the production of transparent petrographical sections to study the microstructural features of the granites studied. In addition, the core of exploration holes drilled in the field “Sivach” was used to select the required mass of explosive during experimental explosions in the landfill.

In order to determine the specific energy of destruction in the explosion in prepared models, both cubic and cylindrical, in the center of one of the faces, an explosive cavity with a diameter of 4-5 mm and a depth of $2/3$ of its height was drilled in which one of the faces was being drilled, in which forms of charge of different shapes were formed to assess the explosive usefulness.

In the cross-section, these charges represented a cylinder, an equilateral triangle and a square, which were inscribed in the circle, having inert gaps: air, water (Fig. 7).



Destruction of the model by explosive loads was carried out in landfills in a closed metal box, the inner surface of which is lined with rubber. The mass of charges for all models of the superior subterranean explosives substance was 150 mg. In total, 18 models were

blown up (three with square and triangular prisms – experimental and three with solid charges – controlling for each kind of granite). The purpose of the experiments was to verify the energy efficiency

of the design of borehole charges.

Granite models were also destroyed on a vertical crust freely falling from a height of 1 m with a weight of 21.4 kg. The energy of the incident cargo was 210 J. On a vertical crush by freely falling cargo, six models were destroyed. It was assumed that the energy purchased at the time of the impact completely transformed into the work of the destruction of a rock sample. In other words, the energy expended on the destruction of a rock sample, ultimately, is spent on the formation of new surfaces in the destroyed rock. Calculation of specific energy consumption per unit of newly formed surface based on the results of rock destruction in technological processes gave an opportunity to estimate the process of destruction in terms of its energy efficiency.

Investigation of the granulometric composition of the destroyed by impact and explosion of rock samples in order to determine the value of the newly formed surface was carried out by the method of sieve analysis [9]. Two sets of laboratory sieves with apertures of size 12.0 were used; 10.0; 7.0; 5.0; 3.0; 2.0; 1.0; 0.50 and 0.25 mm - for evaluation of the grading at the macro level and a set of sieves with apertures of 0.4; 0.315; 0.16; 0.10 and 0.05 mm. This allowed taking into account the role of inhomogeneities inherent in the rock at the micro level. When processing the granulometric composition of the destroyed model, the crushed material was analyzed on the following basic parameters: the intensity of the destruction of the model as a whole; definition of the total mass of the model; diameter of the middle piece; the area of the newly formed surface.

Once again, the surface of the destroyed samples, after separating them into fractions to determine the granulometric composition, was calculated according to the well-known formula [10]:

$$S_n = \frac{6}{\rho} \sum_{i=1}^n \frac{m_i}{d_i} - S_0, \quad (3)$$

where ρ – the density of the material of the model, kg/m^3 ; S_0 is the initial surface of the sample model, cm^2 ; m_i , d_i is the mass (g) and the diameter (cm) of the middle piece of the i -th fraction.

The diameter of the middle piece was determined by the formula

$$d_{mid} = \sum_{i=1}^i w_i d_i, \quad (4)$$

where $w_i = m_i/m$ – the content of the i -th fraction or i -th piece, the fraction of unit; m_i – mass of i -th fraction, g; m – total mass of all fractions, g; d_i is the average size of the i -th piece or i -th fraction, cm.

The energy intensity of the created unit of a new surface characterizes the value, which is called the specific surface energy (γ). When the rock samples are destroyed by the impact of freely falling cargo, the energy consumed by the unit of the newly created surface is calculated by the formula:

$$\gamma_u = \frac{mgh}{S_{ns}}, \quad (5)$$

where S_{ns} – the surface is formed during the impact; m – weight of cargo (21.4 kg); g – acceleration of free fall, 9.8 m/s; h – the height of the cargo drop (1.0 m).

When destruction of the same size rock samples by the explosion of explosive charges of mass M and the heat of explosion Q (kJ/kg), the energy used to create a unit of the newly formed surface – γ_u , was calculated as follows:

$$\gamma_u = \frac{MQ}{S_{ns}}, \quad (6)$$

where S_{nu} – newly formed surface during the explosion, cm²; M – mass of charge, kg; Q – heat of explosion, kJ/kg, and degree of crushing of granite samples according to the formula:

$$K_{d.cr} = D_m/d_{mid}, \quad (7)$$

where D_m is the average size of the sample rib, cm, d_{mid} , – the diameter of the middle piece.

According to the calculated values of the newly formed surface of destroyed samples of granites, the main energy indicators were determined: specific surface energy of destruction at freely falling load and during explosion; the diameter of the middle piece and the degree of crushing of rock samples. The results of experimental studies on the numerical values of the newly formed surfaces of gray and pink granites by explosion charges of various shapes and freely falling loads are given in Fig. 8.

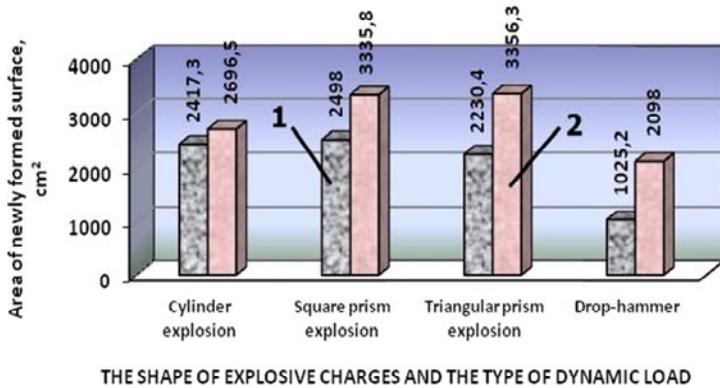


Fig. 8. The area of the newly formed surface during the dynamic destruction of specimens of granite deposit “Sivach”: 1– gray granites; 2 – pink granites

Analysis of the data shown in Fig. 8, shows that the energy of the destruction of granites is significantly influenced by the shape of charges of explosives, as well as the density of defects of the structure in the form of microjoints. It was established that the maximum energy of destruction is characterized by gray granites, destroyed by charges in the form of a triangular prism. The pink granites, destroyed by these same charges, possess the least energy of destruction. When the pink granites are destroyed by prolonged charges with a triangular cross-sectional shape, the degree of fragmentation using the energy of the explosion reaches its maximum value.

For extended charges with a square cross-sectional shape, the energy parameters of the destruction – the newly formed surface, energy, and the degree of crushing of the specimen differed little from those for cylindrical charges in the destruction of gray granites. Significant differences in the above characteristics are observed with the use of solid and cylindrical charges for the destruction of pink granites. In our opinion, this circumstance is due to differences in the peculiarities of the internal structure of gray and pink granites. The authors of the paper [11] found that for granites, the type of structural relationships between elements of its structure, which

include jointing and lamination, determines the nature of the destruction of an explosion. At the same time, because quartz in granites crystallizes the last of the granite melt, is a kind of “cement” that binds grain of other minerals, the nature of the destruction of granites in general depends on the strength of “quartz cement”. With the increase in the number of microjoints in quartz grains, the intensity of granite destruction with explosive loads increases.

To establish the degree of microjointing of quartz grains in gray and pink granites of the deposit “Sivach”, transparent petrographical slabs were made. In them, the intensity of microjointing in quartz grains was measured on microphotographs obtained with a polarizing microscope, equipped with a high resolution digital camera (Fig. 9, *a, b*).

Measurement of the intensity of jointing, carried out on microphotographs, allowed to establish the following. The intensity of microjointing K_j in quartz of gray granites is on average 60-75 microjoints / cm; in the quartz of pink granites the value K_j lie in the range of 95-120 microjoints/cm.

Thus, the intensity of microjointing in quartz grains of pink granites in 1.5-1.6 times exceeds the same indicator for gray granites.

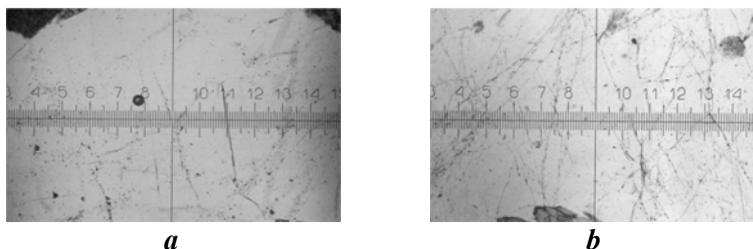


Fig. 9. Microphotographs of joints in grains of quartz gray (*a*) and pink (*b*) granites of the deposit “Sivach”, Korsun-Shevchenkivskiyi Cherkassy region, Ukraine (increase 240×, polarized light, polarizes are parallel)

The value of the coefficient of microjointing has been correlating with the values of the newly formed surface when the samples of granites “Sivach” are destroyed by explosive charges of various shapes. The newly formed surface of pink (more jointed) granites is formed on average 1.5 times more than the formed surface of gray (less jointed) granites.

It is important to note the fact that newly formed surfaces for gray and pink granites, which are destroyed by continuous cylindrical charges, which differ slightly from each other. The newly formed surface of pink granites with the use of continuous cylindrical charges exceeds the same indicator for gray granites by only 10 %, while the ratio of newly formed surfaces in the destruction of pink and gray samples of granite “Sivach” by charges with square and triangular cross-sectional shape is respectively 1.34 (square prism) and 1.50 (triangular prism).

Such a character of destruction, in our opinion, may be due to the predominance of stresses in the complex gradient field arising in a medium that is destroyed by the explosion of the charge of a triangular and square cross section, of the shear component of the stress field. The strength of the rock on the shift, as is known, is on average five times less than that of compression.

For the analysis of the granulometric composition of the destroyed explosion and the freely falling load of granite models, fractions of 0-100 microns were selected. These fractions are formed on the contact “explosive charge-rock”, therefore, the study of their granulometric characteristics gives maximum information on the nature and, respectively, the mechanism of charges action of cross-section various forms. For this purpose, an optical-optical method was used. In this interval with sufficient accuracy for this kind of research, it is possible to determine, by means of the light-optical method, the granulometric characteristics of the destruction products, such as average diameter, median and other quartile sizes (Q_{25} , Q_{75}), the uniformity coefficient of crushing and the asymmetry coefficient.

The method of studying the fractional composition of fine particles by the light-optical method involves the following operations: scattering of the destroyed rock on laboratory sieves, manufacture of the preparation for research under a microscope, mass measurement of particle sizes, processing of measurement results (construction of granulometric curves and their quantitative analysis). For the automatic construction of the cumulative curve according to the measurements used the standard program in the programming language BASIC. In the analysis of cumulative granulometric curves, several numerical parameters were used,

namely: \bar{d} – weighted average particle size, Md or Q_{50} – their median size, S_0 – sorting factor and S_k – asymmetry coefficient.

The results of performed experimental research on the efficiency of the use of various forms charges in the destruction cubic granite models are given in Table. 3.

Table 3

Characteristics of the granulometric composition of finely dispersed fractions of crushing granite models by charges of various forms (fraction 0-100 microns)

Charge form	Average particle diameter, μm	Quartiles, μm			Crushing uniformity coefficient, S_0	Asymmetry coefficient, S_k
		$Md ()$	Q_{75}	Q_{25}		
Cylindrical	19.44	6.41	17.14	2.23	2.77	0.93
Wedge	16.82	4.43	12.21	1.52	2.83	0.95
Square prism	28.82	14.85	34.43	5.58	2.31	0.86

Experimental studies on the estimation of the specific energy of polymineral rocks destruction by various cross-sectional shapes charges established the following.

1. The use of elongated cylindrical charges with a triangular and square cross-sectional shape allows to increase the efficiency of the use of explosive energy for the destruction of polymineral media of complex structure, in particular granites.

2. Microjointing of the mineral components that make up the rock, in particular, quartz, has a significant impact on the energy efficiency of charges of various shapes.

3. The most effective charges on the specific surface energy of the destruction of granites with intensively developed microjointing, are the extended charges of BP with a triangular cross-sectional shape.

4. Charges with a square cross-sectional shape and continuous cylindrical charges differ little in terms of the specific energy of destruction, but the charges in the form of an elongated square prism provide a smaller yield of crumbly fractions on the “explosive-rock” contact.

A new way of explosive destruction of polymineral rocks. The results of experimental studies on the destruction of polymineral

rocks by charges with different cross-sectional shapes were used as the basis for a new method of explosive destruction of strong anisotropic rocks of complex structure (*Patent No. 96511* [12]).

The method contains new technological operations, namely: placement of drilled holes close to the free surface of the ledge of combined charges of different shapes of the cross-section – triangular and square (Fig. 10).

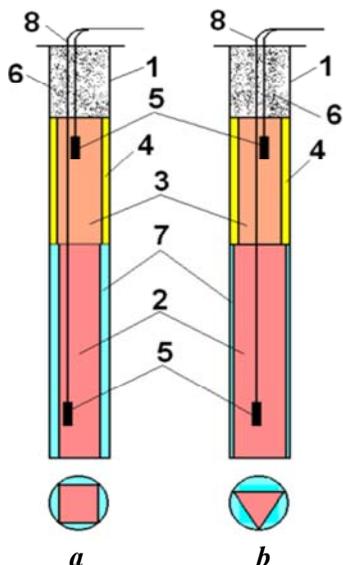


Fig. 10. Combined borehole charges:
a – charge in the intersection of a square prism; *b* – charge at the intersection in the form of a triangular prism;
 1 – hole; 2 – charge from the mixture TNT UH + AS; 3 – section of the conversion explosive DKRP-4; 4 – pyroxylylene powder; 5 – cartridge-guy; 6 – stemming; 7 – inert gap; 8 – non-electric initiation system “Impulse”

During the implementation of the proposed method of explosive destruction of strong anisotropic rocks of a complex structure in the conditions of the “Sivach” quarry, the experimental section of the formation of charges was carried out taking into account the high-resolution areas of increased jointing. In holes drilled in a massif with pronounced local jointing, combined charges of a variable cross section were formed (see Fig. 10).

In the experimental block in the series of holes located in a massif of cracked rocks, the lower part of the explosive charge was formed in the form of a triangular prism from a mixture of TNT + granular ammonium nitrate in proportion of 65/35 or an emulsion explosive of the type “Anemix”. In the upper part, there were sections of the conversion explosive – DKRP-4 or pyroxylylene

powder. The series of wells, located in the rear of the block in a more monolithic massif, were charged with combined charges, placed in the lower part of the charge column in the form of a square prism. The lower part was charged with industrial explosives with a mixture of TNT + granulated ammonium nitrate in the proportion of 65/35 or emulsion explosives of the type “Anemix”, and the upper – conversion explosive – the section of SCRP-4 or pyroxylin powder. The main and intermediate detonators, mounted from two T-400 busters, were installed; the holes were sealed with a gutter from the granite drainage.

According to the results of mass explosions at the experimental and control sites, an evaluation of the results of crushing the repelled rock mass was carried out. The quality of crushing was evaluated on the diameter of the middle piece with the measurement of the granulometric composition of the exploded rock mass using the skew-angle photoluminescence method.

Conclusion

The proposed method of breaking off strong polymineral rocks of a complex structure (locally jointing and jointing) can improve the quality of the exploded rock mass crushing and improve the technical and economic performance of mining enterprises. Moreover, the use of explosive charges in the form of a square and triangular prism can significantly reduce the volume of finely dispersed rock particles in mining quarries for the extraction of raw materials for building materials. The qualitative fragmentation of the exploded rock mass is achieved by adjusting the main parameters of the drilling operations – the geometry of the location and size of the holes, the use of new constructions of combined charges of the cross section various forms, and also the consideration of the direction and intensity of joint systems of different morphologies (gaping degree) in the destructive rock unit.

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IMPROVEMENT OF PERSONNEL PROTECTION EQUIPMENT IN BLASTING OPERATIONS IN UNDERGROUND MINE WORKINGS

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Abstract

The number of fatal accidents, serious injuries and long-term disability in the mining industry remains the highest in the industry. At the same time, the state of the level of labor protection, despite the constant improvement and implementation of all new measures and requirements for safety, remains unsatisfactory and does not comply with the accepted social standards.

In this regard, a comprehensive study of the environment in which work is conducted, from which the tasks of forecasting safety, analyzing the probability of occurrence of accidents, the development and timely implementation of measures to reduce the likelihood of emergence and development of emergency situations, are a promising direction in increasing the rates of injuries in Ukraine.

In dealing with the issue of safety in the manufacture of blasting operations, it can be established that the initial analysis in this case is subject to the following points. As you know, blasting works are a particularly dangerous technology. To avoid danger to people, they not only do not have to be in the explosion zone, but must also move to a safe distance from the blasting site. In work, the concept of creation of the explosive device of new generation is resulted and methods of its practical realization directed on increase of safety of explosive works for the enterprises of coal industry of Ukraine are shown, and also modern explosive devices of the raised safety are presented.

Introduction

The number of fatal accidents, serious injuries and long-term disability in the mining industry remains the highest in the industry. At the same time, the state of the level of labor protection, despite the constant improvement and implementation of all new measures and requirements for safety, remains unsatisfactory and does not comply with the accepted social standards.

The maximum profit from the sale of mineral resources (excluding oil and gas) in the extractive industries of the whole world is provided in the following sectors, shown as profits decrease: coal, copper, iron ore and gold.

At the same time, coal provides about 27% of the world's electric power production, and, for example, China's coal producer with a production of 3.68 billion tons, 70% of primary energy is extracted from coal [1, 2]. Demand for coal has not declined for a long time, despite the changing course of the energy policy towards green energy and renewable sources. According to the forecasts of the international energy agency, the demand for this type of fuel existing in the last decade will not be reduced within the next 20 years.

The technology of underground works at enterprises of different countries is fundamentally different in the level of mechanization, the ratio of energy used, the scale of production, and the state of labor protection. Innovative policy and security policy, regulatory and legal framework for investigating the causes of accidents and providing insurance payments also differ. Therefore, the statistical indicators of security of different countries are very different. The bulk of the world's mineral resources is extracted from high-performance enterprises in large companies and medium-sized companies. However, small mines, and mines with a large share of manual labor, employing about 15 million people only in Asian countries [3], also contribute a significant share in world production. At such enterprises, the level of labor protection remains consistently low for more than half a century, traumatism is generally not taken into account, and products are delivered to illegal markets[4].

Production activities at the coal enterprise in the "man-machine" system proceed in complex and unstable conditions. The situation in the domestic coal industry is complicated by the fact that Ukraine

has some of the most difficult conditions in the world for coal mining. These conditions are related to:

- a large (in terms of coal deposits) depth of mining (500-1300 m);
- layers of low power (0.6-2.2 m);
- high water cut (up to 30 m³ / hour);
- large gas release into the mine workings (absolute gas mobility of mines up to 200 m³ / min);
- the presence of gas dynamic phenomena - sudden emissions, mountain impacts, sudden gas evolution.

All this makes the natural factor prevailing in the coal mining process. It is necessary to take a correct account when planning almost all these processes or even only their components in the framework of the implementation of mining.

Therefore, coal mines of Ukraine should be considered from the position of "man-machine-environment", i.e., as ergatic systems of a high level.

A mathematical description of such a system is impossible on the basis of functional dependencies, since a large number of parameters of these dependencies require a powerful mathematical apparatus, but it is even more difficult to provide this mathematical apparatus with input parameters. Not only do these parameters have a variable stochastic character, but it is simply not possible to define such parameters, or even if they are their range of variation, within a mining enterprise. That is why the only correct and possible tool for analyzing the level of labor protection in the mining industry are statistical methods.

When analyzing the state of labor protection in the mining industry, a large number of factors must be taken into account, which depend on a number of indicators having a random character. They are determined by natural conditions, the design features of machines and mechanisms, the human factor, etc.

Thus, the conclusion is obvious about the prospective direction in increasing the rates of injuries in Ukraine:

- a comprehensive study of the environment in which work is conducted, from which the tasks of safety forecasting, analysis of the probability of occurrence of accidents, the development and timely

introduction of measures to reduce the likelihood of occurrence and development of emergency situations follow.

Small-sized technical means for blasting operations in the mining industry

In dealing with the issue of safety in the manufacture of blasting operations, it can be determined that the initial analysis in this case is subject to the following points. As you know, blasting works are a particularly dangerous technology. The explosion of industrial explosives is accompanied by an almost instantaneous release of huge energy, capable of destroying the mountain massif, various structures and objects in the explosion zone, and also releasing a large amount of gaseous toxic substances into the environment. In order to avoid danger to people, they not only should not be in the explosion zone, but must also move to a safe distance from the blasting site.

When handling explosives, special care must be taken since if this rule is not observed, accidental explosion and defeat of people caught by this explosion can occur. Therefore, special requirements for the storage, transportation of explosives, charging equipment, loading of blast holes and, most importantly, to the blasting industry itself should be observed.

In mines hazardous for gas or dust, blasting can be a source of ignition of an explosive methane-air or dust-air mixture, i.e. may cause explosion of the mixture with subsequent hazardous effects on people.

The danger of injury to people can arise when performing technological work related to drilling holes, cleaning rock mass and other works. Drilling of boreholes is done by manual or column electric drills or perforators with the use of electric or pneumatic energy, various equipment, cable or pneumatic networks, therefore it is necessary to apply general safety measures to exclude injuries to miners by machines, mechanisms, etc.

At present, the Ukrainian coal industry uses small-size explosive devices VP-50 and VP-75, which provide, respectively, the initiation of 50 and 75 series-connected electrical receivers of normal sensitivity to current, as well as their latest modification, the ZEUS

explosive device, capable of operating up to 150 standard electric detonators Construction in the conditions of coal mines, including hazardous by gas or dust.

The device developed by us is manufactured in accordance with GOST 12.2.020-76 and relates to group 1, the level of explosion protection of RV according to GOST 12.2.020-76, which is provided by:

- protection of the "intrinsically safe electrical circuit ИБ" type in accordance with GOST 22782.5-81;
- a special type of explosion protection - C in accordance with GOST 22782.3;

- special operating conditions - X according to GOST 12.2.020.

The explosive device "ZEUS" is a portable device designed to initiate electric detonators of normal sensitivity to the current during blasting operations in mines, including hazardous gases or dust.

The device "ZEUS" is used in the conduct of explosive work in coal mines, including hazardous gases or dust, by the method of electric detonation of electric detonators of normal sensitivity to current. The maximum number of electric detonators connected in series to the explosive circuit is 150 pcs.

The device "ZEUS" can also be used to conduct blasting operations at the open-cast mining facilities.

Operating conditions:

- ambient temperature - from 5 to 40C;
- atmospheric pressure from 660 to 900 mm. gt; p.
- relative humidity of the ambient air up to 100%. at an ambient temperature of 35 ° C.

The device (Fig. 5) consists of a plastic housing in which the energy and nutrition storage units are located. The upper part of the case is closed with a cover, on which are located terminals for connecting the explosive circuit. The display is carried out through the transparent windows in the lid. The case of the device is provided with metal braces for wearing on the waist belt. The instrument is controlled by means of a special key through a hole on the side surface.

The duration of the pulse applied to the explosive circuit is not more than - 4 ms. The electrical diagram of the "ZEUS" device is shown in Fig. 1.

Degree of protection from external influences - IP54 according to GOST 14254-80. Execution according to explosion protection according to GOST 12.2.020, GOST 22782.5-PB И6СХ.

The time for preparation of the device for issuing an explosive pulse at an ambient temperature of $20 \pm 50 \text{ }^{\circ}\text{C}$, no more than 25 sec. The device has a built-in control of the status of batteries with a light indication on the front panel. The battery is charged from the charger with a voltage of 220 V power supply.

The number of blasting cycles without recharging the batteries of the power supply for 30 days, at least - 100 cycles. The device provides signaling:

- on the inclusion (indicator "Charge");
- about the readiness of issuing a pulse into an explosive circuit ("Ready" indicator);
- about the discrepancy of the chain with the permissible parameters ("Chain" indicator).

Overall dimensions, no more than 110x150x50 mm. Weight of the device, no more than 0,8 kg.

Work with the device is carried out in strict accordance with the requirements of DNAPP 0.00-1.17-92 "Uniform safety rules for blasting operations", it is necessary to check the methane content in the place of shelter for the master blaster. If the methane content is $> 1\%$, the appliance is not allowed to operate.

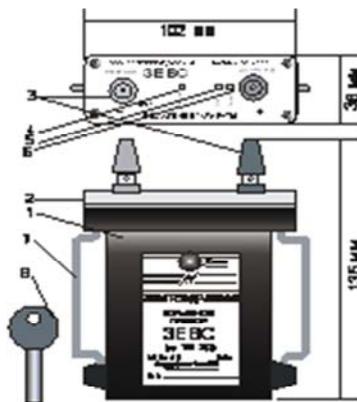


Fig. 1. The explosive device "ZEUS"

The maximum number of initiate series-connected electric detonators of normal sensitivity to current, no more than 150 pcs.

Resistance of the explosive circuit, not less than 5, but not more than -450 Ohm. The magnitude of the flammable impulse delivered to the explosive circuit is not less than $2 * ms$.

In accordance with §226 of DNAP 0.00-1.17-92 "Uniform safety rules for blasting operations," the minimum distance from the place of explosion to the place of shelter of the master blaster in coal mines should be not less than 50 meters.

Before operation, it is necessary to measure the resistance of the explosive circuit with a resistance meter "Sensor-1 A", or another similar device. The measured resistance should not exceed the permissible value - 450 Ohm. Technical specifications (TU U 31.2-30847608-004: 2010) for the explosive device "ZEUS" are given in the Appendix.

The device "ZEUS" was tested in the McNII certification center and further industrial tests in full in the order established by law. On the basis of the industrial tests carried out and the results adopted by the special interdepartmental commission, the bodies of the State Committee of Ukraine for Industrial Safety, Labor Protection and Mining Supervision issued a "Dozvil on the back of the operation No. 3292.10.30-29.52.1" dated 28.09.2010.

Operation with the device is carried out as follows:

- in the place of shelter, the master blaster connects the explosive circuit to the terminals on the body of the device;

- a special key is inserted into the hole on the side surface near which there is the inscription "Key". The power is turned on and the red "Charge" indicator lights up;

- after the green "Ready" light comes on, the device is ready to deliver an explosive pulse with normalized parameters;

- to produce an explosion, a special key is taken out of the device, and an electric impulse is applied to the explosive circuit;

- After the explosion, the explosive circuit is disconnected from the terminals of the device. The device is ready for a new cycle.

In case of failure of the explosion while charging the device or when the device is charged, do not remove the key from the device, wait 20 seconds. - the device itself will shut off and only then detach the explosive circuit and remove the key.

The device has an additional (service) alarm:

- blinking red indicator "Charge" after installing the key - indicates an unacceptable discharge of batteries (protection from deep discharge of batteries). Further work is impossible - the device is automatically switched off;

- blinking of the red indicator "Charge" after the production of an explosive pulse - indicates a critical discharge of batteries and the need to recharge them;

- flashing of the yellow "Chain" indicator after setting the key warns of a malfunction in the explosive circuit (protective function for safety purposes). Further work is not possible - the device will be automatically turned off. It is necessary to check the explosive circuit with a resistance meter and repair the malfunction.

The use of this device, together with its distinctive features, is already helping to improve the safety of blasting operations in coal mines of all categories in gas or dust and in quarries.

Explosive device of high danger

At present, a number of accidents are known in coal mines that are dangerous for gas or dust, connected with the violation of the rules for blasting operations and the use of explosive materials. In particular, these violations concern the finding of a person near the object of blasting at the time of the explosive process. These accidents lead to severe traumatization of personnel, up to the deadly (see, for example, the Dzerzhinsky mine, Dzerzhinskugol association, 2009 Severnaya mine, Dzerzhinskugol association, 2012, etc.).) Therefore, to prevent such accidents, a new generation of explosive devices is proposed, which prevents the supply of voltage to the explosive circuit when the person is positioned near the object of detonation (emergency situation).

The described design relates to the mining industry and is intended for the safe initiation of electric detonators connected to a sequential explosive circuit. The device can be used in coal and ore mines, which are dangerous for explosions of gas and coal dust, quarries, as well as other similar industrial facilities.

An explosive device is known, for example P-50, comprising a power supply unit, a removable switch, a voltage regulator, an alarm

unit, a burst duration limiter unit with terminals for connecting an explosive network, and a voltage converter comprising a storage capacitor, wherein the voltage converter output is connected to the voltage regulator input and the first input of the burst duration limiter, and the voltage regulator output is connected to the first inputs of the voltage converter and the signaling unit, a logic inverter and an adder, and the burst duration limiting unit is a unit for limiting the burst current, the power supply unit being connected via a removable key to the inputs of the drive for supplying the control circuit and the logic inverter, and also to other inputs of the voltage converter and of the signaling unit, the outputs of the logic inverter and the voltage regulator are connected respectively to the first and second inputs of the logical adder whose output is connected to the second input b loka limit the duration and current of the explosive pulse (see. Fig. 2 and Fig. 3).



Fig. 2. The explosive device "Zeus"



Fig. 3. The explosive device "Titan"

This explosive device is much more reliable in operation in comparison with the known ones, since the current and the pulse duration do not depend on the number of detonators. However, based on his shortcomings is the following. The resistance and inductance of the subversive circuit, that is, the wires connecting the explosive device with the electric detonators located in the hole, must correspond to the given values, since the length of this chain corresponds to the safe distance at which the explosive is to be

located during the blasting operations. However, as has been mentioned before, miners often violate the rules by winding wires into the bay or reducing the length of the subversive chain, which makes it possible to detonate the charge from a distance at which the miner can be traumatized and indeed injured by flying pieces of coal and rock or a concomitant explosion of methane-air mixture in place of ignition charge.

The basis for the creation of an explosive device of increased safety is the task of developing a device in which by monitoring the resistance and inductance of external connecting wires of an explosive circuit with an explosive device to withstand the minimum allowable distance between the explosive device and the object of the explosive action before the blast is applied, safety of blasting operations.

This object is achieved due to the fact that an explosive device containing a power source, a detachable key, a unit for limiting the duration of an explosive pulse with terminals for connecting an explosive circuit and a signaling unit, is provided with blocks for determining the resistance and inductance of the wires connecting the explosive circuit with the electric detonators for implementation actually their ignition, thus parameters of both connecting wires should not exceed the limits normalized.

In Fig. 4 is a block diagram of the proposed explosive device; in Fig. 5 - graphics. The device contains a power supply 1, a removable switch 2, a voltage regulator 3, a signaling unit 4, a block 5 for limiting the duration and current of the explosive pulse with terminals for connecting an explosive circuit, a voltage converter 6, which includes a storage capacitor, accumulator 7 for powering the control circuit, the logic inverter 8 and the adder 9, the resistance determination unit 10, and the unit 11 for determining the inductance of the wires connecting the explosive device and the load in the form of electric detonators, the AND gates 12 and 13, you lyuchateli 14 and 15, output terminals 16 and 17.

The output of the voltage converter 6 is connected to the input of the voltage regulator 3 and the first input of the unit 5 to limit the duration and current of the explosive pulse, and the output of the regulator 3 is connected to the first inputs of the converter 6 and the unit 4. The power supply 1 through the detachable key 2 is connected

to the inputs of the accumulator 7 and logic inverter 8, and also with other inputs of the converter 6 and the block 4. The outputs of the logic inverter 8 and the voltage regulator 3 are connected respectively to the first and second inputs of the logical adder 9 whose output is connected to the second in the course of block 5.

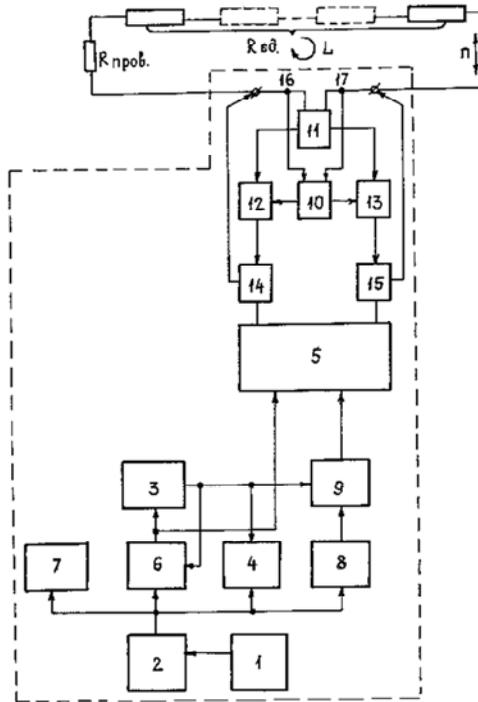


Fig. 4. Block diagram of an explosive device of increased safety

The storage device 7 for supplying the circuit is intended to power the 3 -14 units for the time of the explosive pulse and the discharge of the residual energy of the storage capacitor after the explosive pulse ceases. The outputs of the blocks 10 and 11 are connected to the inputs of the AND gates 12 and 13, whose outputs are connected to the input of the block 5 through the switches 14 and 15, respectively.

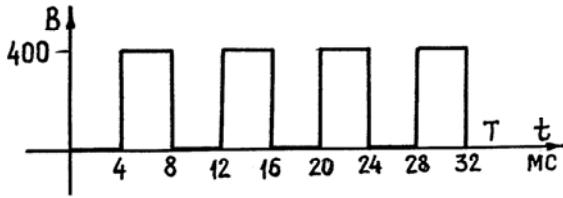


Fig. 5. Operating current graphs

The device works as follows. A removable key 2 connects the power supply 1 to the input of the accumulator 7 and the logic inverter 8, as well as to other inputs of the voltage converter 6 and the block 4 for signaling. Power is obtained by the control circuit of the device. In block 4, the red LED indicates power on. The signal to produce an explosive pulse at the second input of the unit 5 is absent, since there is no signal at the output of the regulator 3 about the charging state of the storage capacitor in the converter 6, and the input of the logic inverter 8 is the supply voltage.

The voltage converter 6 operates by charging a storage capacitor voltage controlled by a voltage regulator 3. When the voltage reaches a predetermined level, a signal is output at the output of the regulator 3, which is applied to the first input of the voltage converter 6 and stabilizes the voltage on the storage capacitor, block 4 and signals the readiness for the explosion, and then is fed to the second input of the adder 9.

Units 10 and 11 control respectively the resistance and inductance of the external wires and, when they correspond to the set values, a signal is applied to the AND elements 12 and 13. Block 4 signals the readiness of the explosion to fire the green LED, and then the signal is applied to the second input of the logical adder 9.

Units 10 and 11 control respectively the resistance and inductance of the external wires and, when they correspond to the set values, a signal is applied to the AND elements 12 and 13. Block 4 signals the readiness of the explosion to fire the green LED, and then the signal is applied to the second input of the logical adder 9.

The person operating the device receiving a signal about the readiness of the latter makes a decision to issue an explosive pulse

and turns off the power unit 1 with a removable switch. The power supply of the control circuit is then fed from the storage device 7. The voltage at the input of the logic inverter 8 is lost, and at its output there is a signal, which simultaneously with the signal from the output of the voltage regulator 3 generates a signal-resolution for the output of an explosive pulse at the output of the logical adder.

The unit 5 connects the explosive circuit to the storage capacitor, limiting the discharge current in the explosive circuit to a predetermined value, which guarantees reliable detonation of any number of electric detonators in the entire range of the device. If the measured resistance and inductance do not correspond to the specified levels, the signals from blocks 12 and 13 go to the inputs of the switches 14 and 15, which disconnect the unit 5. Using two 12 and 13 logic elements AND and two 14 and 15 switches makes it possible to exclude errors when leaving building one of the chains.

The safety characteristics of the proposed device are realized as follows. For blasting operations in Ukraine (and in all CIS countries) a single-core wire VP 2 x 0.7 is used, containing a wire of 0.7 mm in diameter in the conductor. The maximum outer diameter of this wire is 4.4 mm; the electrical resistance of a conductor with a length of 1 km is 50 ohms. Wire VP 2 x 7 is designed for laying temporary explosive lines and is designed for short-term operation at a voltage of 380 V and instantaneous - at a voltage of 660 V AC or 1500 V DC. It is allowed to operate wires with an instantaneous DC voltage of up to 3000 V.

Directly for the ignition of explosives using electric detonators of instantaneous action ED-8 with the following technical characteristics:

- electrical resistance:
for ED-8-E - 2,0 - 4,2 Ohm; ED-8-Zh-1,8 - 3,0 Ohm,
- length of lead wires:
For ED-8-E - 2000 - 4350 mm; for ED-8-Zh-2000 - 3250 mm,
- safe current (the upper limit of direct current, which, flowing through the electric detonators for 5 minutes, does not ignite them) - 0.20 A;
- a long flammable current (the lower limit of direct current, which flashes through the electric detonators for 1 min, ignites no more than 10% of electric detonators) - 0.22 A;

- safe ignition pulse - not less than $0,6 \text{ ms} \cdot A_2$;
- ignition pulse - no more than $2.0 \text{ msec } A_2$;
- response time - $2 - 6 \text{ ms}$.

The shape of the coil of the explosive device chain for various network distances is shown in Fig. 6.

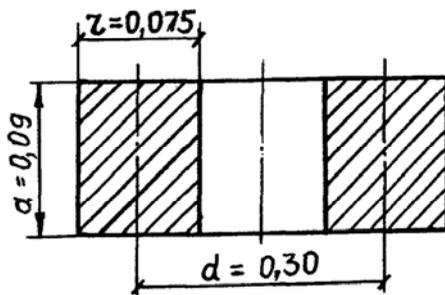


Fig. 6. Shape of the coil of the explosive device chain

At the same time:

- resistance of electric detonators:

$$R_{ed} = r_{ed} \times N \quad (3 \div 4,2) \quad (1 \div 1000) \quad (1)$$

$$R_{ed} = 4.2 \times 30 = 126 \Omega - \text{max};$$

$$R_{ed} = 3 \times 1 = 3 \Omega - \text{min};$$

- resistance of wires:

$$R_{prov} = r_{prov} \times N \quad (50 \Omega / \text{km} \times 0.1) \quad (2)$$

$$R_{prov} = 50 \times 0.1 \times 2 = 10 \Omega - \text{max}$$

- inductance of the wire loop:

$$L_{nem} = f(n, N) \neq \text{const.} \quad (3)$$

- the total resistance of the explosive chain is:

$$R_{\Sigma} = R_{эд} + R_{пров}. \quad (4)$$

- the minimum inductance of the wires of the explosive circuit is:

$$L_{\text{проб.мин}} = 100 \mu_0 \mu \ln \left(\frac{1,4 \times 10^{-3}}{0,35 \times 10^{-3}} \right) \frac{1}{\pi} \quad (5)$$

where μ_0 is the magnetic constant;

μ is the absolute permeability of matter

$$L_{\text{проб.мин}} = 0.1012 \times 10^{-3} \text{ HH};$$

When folding wires into a coil, we have:

$$L = \frac{\mu_0}{8\pi} \omega^2 d \psi, \quad (6)$$

where ω is the number of turns of wires of the explosive circuit connecting electric detonators to the explosive device;

d - diameter of the coil along the middle line;

ψ is a tabular value that depends on the shape of the coil.

Wherein:

$$N = 400 \text{ pcs};$$

$$r = 0.075 \text{ m};$$

$$s = \pi D^2 / 4 = 0.0177 \text{ m}^2;$$

$$l = 0.09 \text{ m};$$

$$d = 0.30;$$

$$\alpha = 0.3;$$

$$L_{\text{кот}} = 0.0436 \text{ HH};$$

The signal received in the circuit has the form.

$$X_L = \omega L = 2\pi 125 (L_{\text{проб.мин}} + L_{\text{кот}}) \quad (7)$$

$$X_L = 34,344 \text{ Ом}$$

$$L_{\Sigma} = L_{\text{проб.мин}} + L_{\text{кот}} = 43,75 \cdot 10^{-3} \text{ Гн.} \quad (8)$$

$$i = E/R - E/R e^{-tR/B} \quad (9)$$

where E is the amplitude value of the voltage applied to the explosive circuit; R is the total resistance of the explosive circuit.

The current of the explosive circuit is defined as:

$$i = 3,49 - 3,49 e^{-2955t} \quad (10)$$

$i_{\text{БИ}} = 0,22 \text{ A}$ - a long flammable current of detonators, triggering their 10%.

Short line: $R_{\text{эД}}' = 4,2 \text{ Ом}$; $R_{\text{проб.}}' = 0$; $L_{\text{проб.}}' = 0$; $L_{\text{кат}}' = 0$
 Long line: $R_{\text{эД}}'' = 126 \text{ Ом}$; $R_{\text{проб.}}'' = 10$; $L_{\text{проб.}}'' = 0,01012 \cdot 10^{-3} \text{ Гн}$;

$$L_{\text{кат}}'' = 0,04365 \text{ Gr}$$

The matrix of the obtained extreme values of the parameters of the explosive network is given below:

$$\begin{array}{l|l} R_{\Sigma} & \rangle 136 \text{ Ом} \\ X_{\Sigma} & \geq 35 \text{ Ом} \\ Z_{\Sigma} & \rangle 140 \text{ Ом} \end{array} \quad \left| \quad (11)\right.$$

Thus, monitoring of the resistance and inductance of the explosive circuit, realized by means of blocks 10 and 11, does not allow the inclusion of an explosive device on a short or collapsed blast line, thus ensuring safe production of blasting operations.

As follows from the analysis of the operation of mining equipment for various purposes, performed on the basis of many years of research, the operation of mining machines and mechanisms, in comparison with other, in particular, general industrial equipment, is characterized by a number of specific features that consistently violate the personnel's operating instructions in the maintenance mode. So, despite the prohibitions of the relevant paragraphs of the "Safety Rules ...", the relevant security interlocks are permanently disconnected or even withdrawn altogether, without any evaluation of the negative consequences of such a design change (see, for example, the accident at the Sukhodolskaya-Vostochnaya mine in 1992 and a number of others).

Therefore, in essence, the work of the system of locks from the incorrect actions of the maintenance personnel, which is given in this section, it is aimed solely at preventing unauthorized disconnection of such security systems or at least in a timely manner inform supervisors about such phenomena. At the same time, it seems that regardless of the type and functional reliability of the main protection, in such auxiliary security and indication devices there will be a need for many years, until the further development of industry, namely the change in the mentality of persons entering the

mining industry the use of such devices is economically and socially inappropriate.

Improvement of methods for ensuring safety during blasting operations

Currently in Ukraine, an explosive device of the type VP-75 [5] is already widely used, which allows simultaneous connection of up to 75 pieces of ED-8 electric detonators connected by an explosive wire VP2 x 0.7 containing one copper wire with a diameter of 0,7 mm. The absence of explosive internal and external electrical circuits in such a device makes it possible in principle to use it effectively in all categories of hazardous gas or dust. However, when analyzing the operating conditions of such an explosive device, there were cases when the personnel carrying out blasting operations performed them with a violation consisting in the fact that immediately at the time of the explosion the personnel was near the explosion site and could be injured by an explosive wave or a secondary explosion on elements environment.

Since purely organizational measures to prevent accidents in blasting operations are not always effective, a previously proposed explosive device of a new design that does not allow the use of explosives when people are in a hazardous zone was proposed [6].

The developed device also refers to products that convert energy stored in capacitors to a safe ignition pulse, used to ignite detonators used for explosive work in explosive corrosive environments, for example, coal mines, all categories in gas or dust, quarries and the like.

This deficiency, the new generation explosive device [6], which does not allow shortening or winding up of blasting wires (lines), while remaining directly near the object of the explosive effect, is deprived of the definitions of the degree [6]. As it was previously established, this effect in practice arises in practice by unauthorized cleaning during the operation of blasting lines, with the formation of an appropriate inductance, the value of which is further taken into account by the device circuit.

However, the statistics of injuries to miners carrying out blasting operations show that with arbitrary folding of blasting lines, the

diameter of the folding can be determined to a large extent arbitrarily, so that certain inaccuracies can be introduced into the created inductance of the device (see the patent of Ukraine [6]). To exclude this phenomenon in Fig. 7, 8 shows the construction of an explosive device, which is devoid of the indicated drawback, for which a standardized coil 2 is reinforced on the body of the explosive device 1 for reeling in the case of an explosive line with rings of the specified diameter.

To exclude the unauthorized removal of the standardized coil during the use of the explosive device, it is strengthened on the device with the help of a special lock 3 (Fig. 3) with its equipment on the surface of the person responsible for the safety of the production of the entire blasting complex. In this case, simultaneously with the reinforcement of the coil inside the housing, through the holes 4 of the coil (Figure 8), the ends of the wires of the blasting lines (not shown) are fixed in the housing 1 (Figure 7), all of which are performed operative special clamps 5 (Figure 7). These operations can also be performed only on the surface, that is, in a non-explosive environment with the help of a special lock 3, located in the protective ring 6 (Figure 9), followed by connection to the lock 7 (Figure 10).

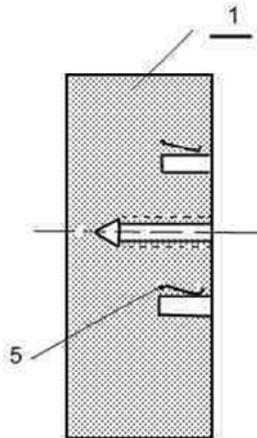


Fig. 7. Shell of an explosive device

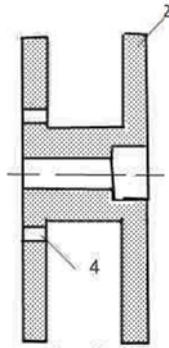


Fig. 8. Coil for winding up the blast line

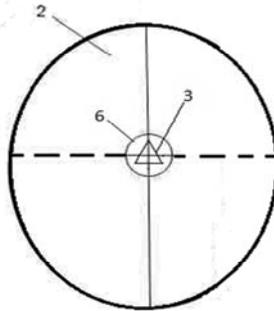


Fig. 9. Coil with lock and protective ring

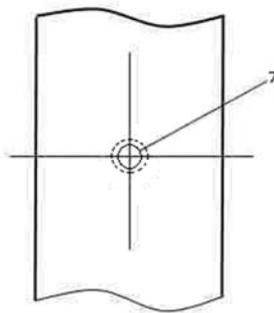


Fig. 10. Fixer of special lock

Conclusions

Thus, the proposed design of a new generation explosive device and explosive technology implemented with it are a sufficiently serious basis for a significant reduction in injuries at coal and gas industry enterprises of all categories in gas or dust.

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