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The contributors present papers analyzing the current state of ore mining and processing and suggest solutions of mineral mining problems through applying resource saving technologies
The book is intended for a broad mining audience of scholars, practitioners, postgraduates and students.

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PREFACE

 

We are glad to present the multi-authored monograph "Traditions and innovations of resource-saving technologies in mineral mining and processing".

To ensure further development of our civilization, the 21st century engineering becomes more sustainable in all spheres of human activity. This strategy becomes prevailing in mining as well. Solution of problems of determining efficient mineral mining systems, improving quality of mined raw materials by applying new advanced technologies of waste disposal and recycling, enhancing efficiency of mining through resource saving will allow increasing significance of environmental and engineering factors in mineral mining and processing.

The monograph presents articles covering the whole range of mining operations from mineral deposit development to end product manufacturing on the basis of efficient use of industrial waste materials.

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THE RESOURSE-SAVING TECHNOLOGY OF MINING COMPLEX STRUCTURED IRON IRE DEPOSITS

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In order to keep their positions in the world markets, mining enterprises of the Kryvyi Rih iron ore field using the deep-mine method need to develop a resource-saving technology for the development of the fields represented by complex-structure ore deposits. Development of the resource-saving technology must be carried out at the initial stage which is directly related to ore extraction and affects content of iron in the extracted ore mass. Growth of iron content in the extracted ore mass can be achieved through the use of selective development of the extraction blocks by means of the chamber development systems.

The existing procedure of determining structural components of the chamber system of development applied at the Kryvbas mines does not take into account thickness of the overlying strata on the side of the hanging wall of the cleaning chamber when calculating the exposure strike. Therefore, it is necessary to improve the procedure for determining the structural components of the chamber system of development when working out complex ore fields, in order to obtain high extraction rates.

For the development of the extraction block, it was suggested to carry out the cleaning works sequentially from the hanging to the lying wall of the complex-structure ore field with the use of the chamber system of development with leaving the non-ore or ore-containing inclusion in the pillar. This sequence of cleaning will reduce concentration of tensile and compressive stresses in the middle part of the non-ore or ore-containing inclusion which will contribute to a 1.5–2.0-time increase in its stability.

It has been established that stability of the cleaning chamber, in addition to its dimensions and physico-mechanical properties of the ore, is influenced by horizontal thickness of the inclusion, safety factor, its life span and the sequence of cleaning in the extraction block. Thus, at the safety factor of rocks of the non-ore inclusion less than 10-12, it is expedient to use the sublevel-chamber version of the development system, otherwise, the horizonal-chamber version.
Keywords: deep mining, iron ore, stress, stability, chamber system of development

1. Introduction

Today, iron ores are extracted by strip and deep mining. Poor ores (magnetite quartzites) are extracted by conventional strip mining and rich ores (ferruginous quartzite) by deep mining. As an exception, magnetite quartzite is extracted recently by deep mining at Ordzhonikidze mine. Gigant-Glyboka and Pershotravneva mines extracted this ore right up to 1997.

In the geological and mining context, the Kryvyi Rih iron ore field is a complex-structure field composed of single, parallel-approaching deposits and separated pockets with useful component content in the massif within 10-37 % to 58-67 % [1, 2]. In some regions of the ore deposits, there are non-ore or ore-containing inclusions (BOI) with the useful component content much smaller than the cut-off grade relative to the ore massif under development. The volume of reserves of non-ore or ore-containing inclusions with a content of useful component less than the cut-off grade makes 5–12 % for rich ores and 10-15 % for poor ores of the total field volume.

Development of the deposits represented by complex-structure ore deposits (CSOD) by deep mining with the use of conventional development systems results in a 3–6 % reduction of iron content in the extracted ore relative to the basic content of the useful component in the ore massif. With an increase in iron content in the extracted ore mass, the loss of ore is increased by 1.5-2.0 times, which leads to a lower mining efficiency, and as a consequence, to the loss of positions in the world markets. Thus, development of the resource-saving technologies that will enable efficient development of complex-structure ore deposits of the Kryvyi Rih field is of very high importance. It should be noted that modernization of the technological processes must begin from the first stage of production (massif destruction, extraction and delivery of ore) which will significantly improve technical and economic indicators of mining and processing.

2. Literature review and problem statement

The issues of working out the technology, criteria, and methods for controlling the process of raw dressing taking into account the indicators of energy efficiency, environmental and economic components were considered in [3,4]. To solve the problem of resource conservation, it is necessary to use an integrated approach to the concept of hierarchical management of ecological and economic systems taking into account features of the functioning elements and using the theory of organizational and technical management.

A series of studies aimed at establishing dependences of the extraction indicators on the action of rock pressure and the extraction sequence and determining the rational values of parameters of the main structural members of the mining systems were dedicated to the development of the complex-structure deposits [5, 6].

It was proved that mining, geological, and technical conditions influence efficiency of extraction of the field reserves. The main factors of successful development of the CSOD include the sequence of cleaning, rock pressure, intensity of works, number and stability of pillars, floor height, relative location of chambers and pillars in deposits of the main strike.

Experience of the mines of the Kryvyi Rih field has proved that efficiency of the development of the complex-structure deposits is influenced by the sequence of cleaning excavation, thickness and strength of the intermediate stratum (non-ore inclusion) and the mining system [7]. When developing the CSOD by the chamber method with leaving pillars, the number of the latter should be minimal, as they are stress initiators and complicate conditions for further development of deposits. When determining the zones of displacement and the zones of relief in mining the parallel bodies, it was proved that the rock pressure in the ore-containing rocks of the hanging wall is much lower than in the underlying rocks of the underwall [8-10].

It was established in [11-13] that the advanced development of the hanging wall layers reduces the rock pressure in the layers of the main strike. Such controversial conclusions on the sequence of cleaning have arisen because of the fact that these studies were conducted under different conditions and at different depths. The authors of study [14] have identified various zones of rock pressure variation determined by the advanced mining of one of the layers as well as a temporary lag of works and their spatial and mutual arrangement. The bearing pressure in rocks is distributed unevenly along the strike but focuses on the flanks of the excavated space. As a result, the zones of stress relief and concentration appear in the rocks between the deposits [15, 16]. Stress concentration can be reduced by means of bulk extraction while controlling the ore quality.

The results of the study on optimization of ore extraction and processing set forth in [17] have led to a conclusion that the indicators of efficiency of managing the processes of ore dressing significantly depend on accuracy of current information on parameters of the technological processes. In most cases, electromagnetic, ultrasonic, and radiometric methods are used in development of non-destructive ore testing methods.

Based on the critical analysis of the studies devoted to the issues of mining and processing of mineral resources, the following conclusions can be drawn:

1. Iron content in the extracted ore can be raised at the first stage by the use of resource-saving selective mining, without application of the dressing process. In this case, development of deposits with horizontal thickness of non-ore or ore-containing inclusions less than 12 m is offered to be carried out by the conventional deep-mine method with involvement of dressing works.

2. The negative effects of weakening of bearing capacity of the intermediate stratum which adversely affect its stability during formation of the next chamber are not taken into account in the advance extraction of deposits with non-ore or ore-containing inclusions.

3. There are no substantiated scientific and practical recommendations concerning development of the complex-structure deposits by the chamber method which enables not only growth of iron content in the extracted ore but also a differential approach to the issue of raising the chamber stability.

Thus, it is necessary to improve the resource-saving technology when developing the CSOD. This will ensure not only higher iron content in the extracted ore mass but also increased chamber stability. Therefore, it is necessary to determine how dimensions of the non-ore or ore-containing inclusion affect structural components of the chamber method.

3. The aim and objectives of the study

The study objective was to substantiate stable parameters of structural elements of the chamber system in the development of complex-structure ore deposits which will improve indicators of ore mass extraction owing to selective extraction. In order to achieve this objective, it was necessary to determine the maximum permissible steady width of the chamber roof exposure depending on the structural elements of the chamber system of development and thickness of non-ore or ore-containing inclusions in selective development of complex-structure ore deposits.

4. Materials and methods used in studying the stability of non-ore or ore-containing inclusions in the application of chamber development systems

Solution of many issues related to development of mineral resources and study of geological and tectonic development of the earth's crust are based on the results of experimental studies of the stressed state of the rock massif. These studies are determined by the massif breakage in the course of deep-mining operations resulting in technogenic disasters of geomechanical nature which have both positive and negative effect.

An elementary cube taken from a stressed body has, in a general case, nine stress vectors in its faces: σ_x, σ_y, σ_z (normal) and τ_{xy}, τ_{xz}, τ_{yx}, τ_{yz}, τ_{zx}, τ_{zy} (tangential) that form the so-called stress tensor characterizing the stressed state in a given point *O* of the solid body and having the form

$$S_{ij} = \begin{vmatrix} \sigma_x & \tau_{xy} & \tau_{xz} \\ \tau_{yx} & \sigma_y & \tau_{yz} \\ \tau_{zx} & \tau_{zy} & \sigma_z \end{vmatrix} = p_{ik} \times n_i, \qquad (1)$$

where σ is the internal force stress occurring in the massif, N/m^2, (t/m^2); τ are tangential stresses occurring in the massif, N/m^2, (t/m^2); p_{ik} is the cumulative stress, three mutually perpendicular areas around one point; n_i is the normal unit vector to the corresponding study area; *i, k* are indices of the coordinate axes *x, y, z*.

The indices indicate the strain direction and relative shifts which characterize the change of the parallelepiped shape and in which the coordinate plane an angle straining takes place that causes destruction of a non-ore or ore-containing inclusion [18].

In many cases of determining pillar stability, it is considered in mining as a fixed beam and in order to ensure its stability, the maximum stresses must meet the condition

$$\left.\begin{aligned}\sigma_{\max} &\le [\sigma],\\ \tau_{\max} &\le [\tau],\end{aligned}\right\} \quad \sigma \square \tau, \tag{2}$$

where $[\sigma]$ is the material strength limit, N/m^2, (t/m^2); $[\tau]$ are permissible tangential stresses, N/m^2, (t/m^2).

The authors of studies [8–10] argue that in calculating the pillar stability, the main criterion is strike but a zone of cracks is formed in rocks under the influence of pressure. Therefore, when determining maximum permissible stresses bringing about reduction in strength limit of the pillar consisting of rocks, it is necessary to take into account the massif structure and the time of its existence.

In most cases of deep mining, pillars have a rectangular shape, so the middle part of the exposure strike is the most dangerous and the maximum stresses are determined from formula

$$\sigma_{\max} = \frac{M_x}{W_x} \le [\sigma], \tag{3}$$

where M_x is the value of the maximum bending moment in the part z of the BOI exposure strike along the x axis, N/m, (t/m); W is the moment of resistance of the pillar.

It should be noted that deflection is the main component of the vector of shift of points in the rock massif, so the value of deflection is small compared with the pillar thickness, i.e. $w \ll h$.

The maximum stresses occurring in the pillar represented as a fixed beam are determined from expression

$$\sigma_{\max} = \frac{6 \times M_x}{l \times h^2}, \tag{4}$$

where l is the exposure strike (the pillar length), m; h is the pillar thickness (normal thickness of the BOI), m.

Studies [5, 11, 15] have proved that not all rectangular bodies can be regarded as a fixed beam in calculations of maximum stresses. In the case when thickness of the pillar is considerably less than its length, the pillar should be regarded as a thin rigid plate and not as a fixed beam.

In accordance with the Kirchhoff's first and second assumptions and Cauchy's formulas, we have obtained expressions for determining components of the tensor of stresses σ_x, σ_y, τ_{xy} in the plate through the function of deflection w in its middle plane

$$\sigma_x = -\frac{E \times z}{1-\mu^2} \times \left(\frac{\partial^2 w}{\partial x^2} + \mu \frac{\partial^2 w}{\partial y^2} \right);$$

$$\sigma_y = -\frac{E \times z}{1-\mu^2} \times \left(\frac{\partial^2 w}{\partial y^2} + \mu \frac{\partial^2 w}{\partial x^2} \right);$$

$$\tau_{xy} = -\frac{E \times z}{1+\mu^2} \times \frac{\partial^2 w}{\partial x \times \partial y}, \qquad (5)$$

where E is the Young's modulus; μ is the Poisson coefficient.

After corresponding transformations of expressions (5), conditions of the BOI stability at maximum stresses in its middle part are obtained:

$$\begin{cases} \sigma_x = \dfrac{6 \times M_{x\max}}{m_{BOI}^2} \le [\sigma], \\ \sigma_y = \dfrac{6 \times M_{y\max}}{m_{BOI}^2} \le [\sigma], \\ \tau_{xy} = \dfrac{6 \times H}{m_{BOI}^2} \le [\tau]. \end{cases} \qquad (6)$$

Let us consider the technological processes taking place in the development of ore fields by deep mining. Deposits of the Kryvyi Rih iron ore field are conventionally mined from the lying to the hanging wall. According to the performed analysis, it was found that it is expedient to mine from the hanging to the lying wall when developing the complex-structure ore fields by deep mining [5, 6]. However, mining operations must be carried out from the hanging to the lying wall when developing the CSOD. Let us consider how the extraction technology changes in the selective development of the CSOD with extraction from the hanging to the lying wall and the use of the chamber development method.

The proposed technology foresees a certain procedure for conducting mining operations depending on the mining and geological conditions of the CSOD while the development of the cleaning block is carried out in two stages: *Stage I*. Ore is extracted initially in the hanging wall leaving the non-ore inclusion in the cleaning block as a pillar; *Stage II*. The remaining ore in the lying wall is removed from the block depending on the sequence and priority of the mining operations.

In order to obtain high rates of extraction of the ore mass using the chamber development system, it is necessary to ensure stability of pillars, exposures and the BOI for the entire time of development in the cleaning blocks. Consequently, depending on the stage and sequence of mining operations in the extraction block, different loads will be applied to the BOI. Depending on loading of the BOI, a field of tensile or compression forces is formed in the massif [2, 6].

It is known from the theory of material resistance that if a specimem is evenly loaded over time, normal stresses grow in it to the ultimate compression strength of the material. As soon as the compressive stresses become larger than the compressive strength of the BOI or linear strains appear, the interchamber pillar will be destroyed. Thus, in order to maintain integrity of the BOI which is an interchamber pillar, it is necessary to fulfill the following condition during the cleaning works in the block

$$\begin{cases} \sigma \le \sigma_k \cong [\sigma_{st}], \\ \varepsilon = 0, \end{cases} \tag{7}$$

where σ are normal stresses, MPa; σ_k are critical stresses, MPa; $[\sigma_{st}]$ is the rock compression strength, MPa; ε are the linear strains.

In the event that the compression and tensile stresses are acting in time, normal stresses in the BOI initially grow and then fall. When loading is repeated, linear strains appear in the pillar significantly reducing compression strength of the rock. When the load increases in time, normal stresses grow in the pillar according to expression (7) and when the load falls, normal stresses do not reach the limit strength of the rocks which leads to the pillar destruction under the following boundary conditions

$$\begin{cases} \sigma \square\ \sigma_{\mathrm{k}} \cong \sigma_{\mathrm{v}} \square\ [\sigma_{\mathrm{st}}], \\ \varepsilon \ne 0. \end{cases} \tag{8}$$

In view of the above, it is necessary to determine parameters of the structural elements of the chamber system of development when working in the cleaning block from the hanging to the lying wall with a provision of stability of the non-ore or ore-containing inclusion.

5. The results obtained in the study of stable parameters of the cleaning chamber in stable ores

To obtain high extraction rates in the case when the cleaning block is represented by a complex-structure field, it is expedient to apply selective development of ore reserves [3]. The selective development of the cleaning block differs from the conventional mining in that the ore reserves are extracted in two stages.

The first stage involves ore extraction from the hanging wall of the deposit with dimensions of the structural elements determined by the procedure [19] with formation of a cleaning chamber. It should be noted that according to the method [19], the width (thickness) of the non-ore or ore-containing inclusion does not affect parameters of the cleaning chamber of the first stage.

After removal of the caved-in ore from the first cleaning chamber, the inter-chamber ore pillars and the ceiling are not caved-in at this stage. Therefore, when determining the time of existence of exposure and pillars, it is necessary to take into account the total time to be spent for extraction of the cleaning block (including the second stage). Thus, it is necessary to make changes in the existing procedure of determining time of existence of pillars and exposures for the cleaning chamber of the first stage. The time of existence of the exposure (t_o) and the pillars (t_c) for the cleaning chamber of the first stage when developing the block represented by the complex-structure field is determined by the formula

$$t_o(t_c) = t_v + t_p + t_z + t_{r1}, \tag{9}$$

where t_v is the time for the development of the caved-in rock mass from the cleaning wall of the chamber, month; t_p the time for preparatory and cutting work in the wall of the second stage of development (according to the practice, it takes 3–7 months), month; t_z is the time for drilling and blasting (caving-in) of a rock mass in the wall of the second stage of development (according to the practice, it takes 2–6 months), month; t_{r1} is the time for preparation and mass caving-in of pillars and ceiling around the cleaning chamber of the first and second stages of development (according to the practice, it takes 1–3 months), month.

In determining the parameters of cleaning chamber of the second stage, it is necessary to take into account the previous calculation values of the first cleaning chamber which include the chamber width along the seam strike and the width of the inter-chamber pillars with the following boundary conditions

$$\begin{cases} a_{II} = a_I; \\ c_{II} = c_I; \\ b_{II} \le b_I, \end{cases} \qquad (10)$$

where a_I, a_{II} is the width of the first and second cleaning chambers along the seam strike, m; c_I, c_{II} is the width of the inter-chamber pillar relative to the first and second cleaning chambers, m; b_I, b_{II} is the sloped strike of the exposure relative to the first and second cleaning chambers, m.

According to paragraph 4, it was established that when the normal thickness of the non-ore or ore-containing inclusion is 5 times less than the exposure strike, then in accordance with the theory of material resistance, the pillar should be considered as a fixed beam and as a thin rigid plate in other cases. Maximum stresses arise in the middle part of the exposure strike when the BOI is represented as a beam and determined by formula (3). Substituting the input values in formula (3) and performing corresponding transformations, we obtain an expression for definition of the maximum permissible stable strike of the BOI exposure

$$l_{\text{BOI}} = \frac{4 \times [\sigma] \times h_{\text{th}}^2}{q} = \frac{4 \times [\sigma] \times m_{\text{BOI}}}{a_{II} \times \gamma_{\text{BOI}}} = \frac{4 \times K_f \times f \times \text{K}_{\text{str.o}} \times m_{\text{BOI}}}{a_{II} \times \gamma_{\text{BOI}} \times \text{K}_z}, \qquad (11)$$

where $[\sigma]$ is the limit compression strength of the BOI rocks, t/m^2; h_{th} is the pillar thickness, m; m_{BOI} is the normal thickness of the non-ore or ore-containing inclusion, m; γ_{BOI} is the volume weight of the BOI rocks solids, t/m^3; K_f is the factor of conversion of rocks durability in stress; f is the coefficient of durability of rocks of the non-ore or ore-containing inclusion by the scale of Prof. Protodiakonov; $K_{\text{str.o}}$ is the coefficient of structural weakening of rocks by cracks (taken from 0.65 to 0.95); K_z is the factor of safety of rocks (taken 1.5-2.0).

The criterium of stability of exposures and pillars is satisfaction of inequality (12) in which values of the actual equivalent strike of exposure (11) are compared with geometric dimensions of the inclined exposure in the cleaning chamber of the second stage of development [19], m

$$l_{\text{th}} = \frac{a_{II} \times m_{\text{BOI}}}{\sqrt{a_{II}^2 + m_{\text{BOI}}^2}} \le l_{\text{BOI}}, . \tag{12}$$

In the case when the normal thickness of BOI is 5 times less than the strike of the exposure or the chamber width along the seam strike, stability of BOI is calculated as for a plate. The stresses occurring in the plate are determined by formula (6).

In accordance with the conditions of static equivalence, the internal moments occurring in the plate and expressed in terms of the strike of exposure of the plate are determined by the following differential equations

$$M_x = -D \times \left(\frac{\partial^2 \omega}{\partial x^2} + \mu \times \frac{\partial^2 \omega}{\partial y^2} \right),$$
$$M_y = -D \times \left(\frac{\partial^2 \omega}{\partial y^2} + \mu \times \frac{\partial^2 \omega}{\partial x^2} \right), \quad (13$$
$$M_{xy} = -D \times (1 - \mu) \times \frac{\partial^2 \omega}{\partial x \partial y},)$$

where M_x,M_y are the bending moments along the x,y axes, respectively; μ is the Poisson coefficient; D is the bending stiffness of the plate and ϵ is the physico-geometric characteristic of the plate in bending determined by

$$D = \frac{E \times m_{\text{BOI}}^3}{12 \times (1 - \mu^2)}, \tag{14}$$

where E is the Young modulus.

The moment of bending of the plate by the transverse forces is described by the differential equation

$$\frac{\partial^4 \omega}{\partial x^4} + 2 \frac{\partial^4 \omega}{\partial x^2 \partial y^2} + \frac{\partial^4 \omega}{\partial y^4} = \frac{q}{D}. \tag{15}$$

The differential equation (15) is solved by numerical methods with taking into account boundary conditions (16) while it should be borne in mind that BOI represents a fixed plate

$$\begin{cases} \omega\big|_{x=0}^{x=a} = 0, \\ \dfrac{\partial \omega}{\partial x}\bigg|_{x=0}^{x=a} = 0, \end{cases} \text{and} \begin{cases} \omega\big|_{y=0}^{y=a} = 0, \\ \dfrac{\partial \omega}{\partial y}\bigg|_{y=0}^{y=a} = 0. \end{cases} \tag{16}$$

In the development of the fields with the use of systems with an open cleaning space, there are three possible options for formation of cleaning chambers which differ in the ratio of the chamber width to the exposure strike. For engineering calculations, we offer an equation for determining maximum bending moments for different ratios of the cleaning chamber length to the exposure strike

$$\begin{cases} M_{\text{x max}} = C_1 \times l_{\text{BOI}} \times m_{\text{BOI}} \times \gamma_{\text{BOI}} \times a_{II}^2 \Big|_{l_{\text{BOI}}^2 \geq a_{II}}, \\ M_{\text{y max}} = C_2 \times l_{\text{BOI}} \times m_{\text{BOI}} \times \gamma_{\text{BOI}} \times a_{II}^2 \Big|_{l_{\text{BOI}}^2 \geq a_{II}}, \\ M_{\text{x max}} = C_3 \times a_{II} \times m_{\text{BOI}} \times \gamma_{\text{BOI}} \times l_{\text{BOI}}^2 \Big|_{l_{\text{BOI}}^2 < a_{II}}, \\ M_{\text{y max}} = C_4 \times a_{II} \times m_{\text{BOI}} \times \gamma_{\text{BOI}} \times l_{\text{BOI}}^2 \Big|_{l_{\text{BOI}}^2 < a_{II}}, \end{cases} \tag{17}$$

where C_1, C_2, C_3, C_4 are correction factors of bending moments. They are taken accordingly.

Substituting the input values (17) in expression (6) and performing corresponding transformations, we obtain the formula for determining the maximum permissible BOI exposure strike depending on the width of the cleaning chamber of the second stage and the exposure strike

$$l_{\text{BOI}} = \frac{[\sigma] \times m_{\text{BOI}}^2}{3 \times C_1 \times a_{II}^2 \times \gamma_{\text{BOI}}} = \frac{К_f \times f \times К_{\text{str.o}} \times m_{\text{BOI}}^2}{3 \times C_1 \times a_{II}^2 \times \gamma_{\text{BOI}} \times К_{\text{z}}}. \tag{18}$$

In the case when the chamber length along the strike is greater than the inclined exposure strike determined for the camera of the first stage (Fig. 5*c*), the stable strike of the exposure is determined from the expression

$$l_{\text{БРВ}} = \sqrt{\frac{[\sigma] \times m_{\text{БРВ}}^2}{3 \times C_1 \times a_{II} \times \gamma_{\text{БРВ}}}} = \sqrt{\frac{К_f \times f \times К_{\text{стр.о}} \times m_{\text{БРВ}}^2}{3 \times C_1 \times a_{II} \times \gamma_{\text{БРВ}} \times К_{\text{з}}}}. \tag{19}$$

Thus, according to the results of theoretical studies, parameters of the structural elements of the chamber system of development for mining complex-structure ore fields in various mining and geological conditions are determined.

Reliability of the results obtained in theoretical studies can be proved with the help of laboratory or mathematical modeling. When creating an object in a laboratory environment, it is necessary to observe a correct reproduction of the rock massif by producing equivalent materials [20]. However, the disadvantage of this modeling method consists in large tensions and long time required for both model creation and development. Mathematical finite element modeling is the most effective method. This method allows one not only to create a model of a corresponding size but also change its characteristics in a short time.

Thus, simulation of the change in the stress field in a rock massif around the cleaning chambers at various stages of development was conducted with the help of the ANSIS software system.

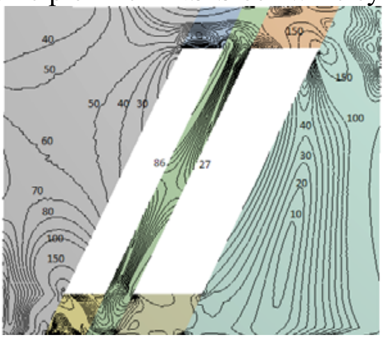

Fig. 1. Results of simulation of the CSOD development from the hanging to the lying wall at compression strength of the non-ore inclusion equal to 160 MPa; the simulation stages: initial, intermediate, and final, respectively

A total of 9 series of studies were conducted. They differed in physical and mechanical properties of the ore massif and the BOI. All other indicators (development depth, horizon level, thickness) remained unchanged.

When conducting studies on the model, the field of equivalent stresses in the massif around the cleaning chambers and in the middle part of the non-ore inclusion was recorded at different development stages. The safety factor of the non-ore or ore-containing inclusion is determined by the expression

$$K_{st} = \frac{K_{str.o}}{K_z}, \tag{19}$$

where $K_{str.o}$=0,65-0,95 is the factor of structural weakening of rocks by cracks (taken 0.85); K_z=1,5-2,0 is the safety factor of the rocks (taken 1,5).

At a horizon level of 90 m, the inclined strike of the exposure is 104 m, and, taking into account the expression (19), it is 58.9 m. According to the simulation results, it was found that the exposure maintains its stability at the level of 90 m (the inclined exposure strike of 58,9 m), the cleaning chamber width of 25 m at the BOI strength greater than 120 MPa. In the case when strength of the rocks is less than 120 MPa, the exposure strike will be unstable, the BOI and the cleaning chambers will be destroyed.

6. Discussion of the results obtained in the study of stable parameters of the cleaning chamber

According to the study results, the NDGRI procedure for determining structural components of the chamber method used in the development of complex-structure ore fields was improved. It will provide stability to the cleaning chambers for the entire period of the development of ore reserves and enable extraction of clean ore from the cleaning chambers. In order to maintain positions in the world market, the mining enterprises practicing deep-mine extraction must introduce resource-saving technologies at the first stage of extraction. This will raise iron content in the extracted ore mass by 2-4 % without additional capital and operating expenditures.

The sequence of cleaning works and the inclined exposure strike for determination of the structural members of the chamber system of development during the CSOD development were substantiated. Cleaning works in the extraction block should be carried out sequentially from the hanging to the lying wall of the complex-structure ore deposit by means of the chamber system of development with leaving of a non-ore or ore-containing inclusion in the pillar. This will reduce concentration of tensile and compressive stresses in the middle of the non-ore or ore-containing inclusion. This will make it possible to improve stability of the cleaning chambers at the contact with BOI by 1.5–2.0 times. It should be noted that the inclined exposure strike has a significant influence on stability of the cleaning chamber and depends on physical and mechanical properties and the horizontal thickness of ore and the BOI, the time of its existence, the sequence of cleaning works in the extracted pillar and the depth of the extraction development.

Thus, development of the extraction blocks of the presented CSOD by the chamber development systems will significantly improve ore mass extraction rates, and in some cases, exclude dressing of the ore mass from the production complex at the final stage. This will contribute to reduction of ore extraction costs and expansion of the world market.

It should be noted that there is a 2-time increase in the life span of the cleaning chambers and pillars in the development of the CSOD by selective method. In order to ensure stability of the cleaning chambers with taking into account the safety factor, a 1,5-time increase in dimensions of the inter-chamber pillars and the ceiling is necessary when the horizonal-chamber system of development is used.

Development of the extraction pillar by sublevel/ chamber development systems will make it possible to reduce pillar dimensions to increase the volume of cleaning chambers, thus expenses for preparatory and cutting works will increase.

The results of the performed studies can be used in the development of the fields of naturally rich iron ores containing non-ore inclusion. It has been established from analysis of the mining and geological characteristics of the Kryvyi Rih field that application of the selective development method with chamber development systems will reduce from 20 % to 10 % volumes of non-ore rocks dumped on the earth's surface. This will contribute to an increased output of marketable products with iron content in the ore mass of 63–65 % by 0.5 million tons at annual productivity of the enterprise 6.0 million tons.

These studies are innovative for conditions of the Kryvyi Rih iron ore field. For example, the issue of using a combination of development systems within an extraction block, e.g. the chamber system of development and the system with massive ore and rock caving and their sequence in extraction of cleaning panels was not yet solved.

7. Conclusions

Stability of the cleaning chamber exposure strike depends not only on the width and life span of the cleaning chamber of the second development stage but also on physical and mechanical properties of the non-ore or ore-containing inclusion. For example, at a horizon level of 75–90 m, stability of the cleaning chamber is ensured when its width along the strike does not exceed 15 m. In cases when the sublevel height is 25–30 m, stability of the cleaning chamber is affected by thickness and strength of the non-ore or ore-containing inclusion. For example, when the BOI durability is more than 12 and its horizontal thickness is more than 10 m, it is expedient to apply the horizon development version of the chamber system in stable ores. In other cases, in order to ensure high extraction rates, it is advisable to apply the sublevel-chamber versions.

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IMPROVING NORMATIVE LEGAL SUPPORT FOR LABOR PROTECTION TO REALIZE NOVATIONS IN OIL AND GAS EXTRACTION INDUSTRY

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Summary

The paper considers necessity of improving the technical support of the labor protection in the oil and gas extraction industry of Ukraine in connection with the latest technology introduction. The research provides general description of snubbing equipment and the efficiency of it applications in comparison with traditional technologies, that needs the productive layer blockage. The paper presents application results of snubbing equipment in the pilot project framework in wells of 2 500 m to 3 700 m depths and the hydrostatic pressure at these depths. The article analyzes the world-wide impact of the snubbing equipment installation that confirms the best conditions for the mineral resource protection and ecological safety and indisputable and significant economic effect.

The research gives recommendations and substantiates necessity of amendments to the current normative-legal act on labor safety "Safety rules in the oil and gas industry of Ukraine". The article presents main comparative tables of proposed changes to the normative legal act, taking into account the need for normative regulation of the application of the latest technologies and the implementation of the provisions of European legislation to the national.

Key words: snubbing equipment, well, oil and gas extraction industry, labor protection, implementation.

Introduction

Research subject is normative legal support of labor protection in the oil and gas extraction industry of Ukraine.

Novelty and topicality. New production technologies introduction, new equipment and machinery development is the integral part of state's investment and innovation strategy [1]. The oil and gas complex is one of the leading industries in Ukraine [2], therefore introduction of the best world practices in oil and gas industry has a positive effect on the fuel and energy complex dynamic development and country economy at all. Innovative organizational and technological measures in production require mandatory monitoring in order to determine the safety level for workers, as well as all persons who may be affected by such innovations.

Methodology for achieving the goal bases on system analysis method of normative maintenance for labor protection in the oil and gas extraction industry and advanced world experience generalization in the normative regulation for application of the latest technology and provision of the EU normative acts implementation to national legislation.

Modern research analysis. Ukrainian scientists pay very little attention to problems of snubbing system implementation at present. In this direction there are scientific developments of V. Vitryk, Ye. Kryzhanivskyy, I. Kostryba. Snubbing units are safer for use, provide better accident prevention at wells, as well as service personnel protection of the work performing high danger work contrary to traditional well repairing techniques by silencing [3].

Goal is to substantiate the necessity to amend the current normative legal act on labor protection "Safety rules in the oil and gas industry of Ukraine" taking into account the necessity of normative regulation application latest technologies and implementation provisions of EU legislation to Ukrainian legislation.

Brief basis of snubbing technology

We have a steady tendency to reduce the operational capacity of oil and gas wells along with the lack of the opening of new deposits with significant reserves in Ukraine today [5]. That in turn significantly affects the increase complexity, duration and work costs to maintain them in working condition, the amount of technical equipment necessary for this, as well as costs for acquiring and retaining fixed assets. Such work execution by traditional technologies that predict the productive formation silencing is a long-term and has disadvantage of high probability of productive formation pore space colimitation [6]. The efficiency of repaired well by productive formation silencing is sometimes worse than before repair in practice. Except this, a long time is required to return the well to a stable operating mode. Such technology of carrying out work on capital and current repairs oil and gas wells leads to deterioration of reservoir properties of layers and, consequently, to decrease their efficiency during operation. That is why oil and gas industry have urgent need of development and improvement of methods, technologies and technical means for well repairing, developing and intensification as well as search for new technical solutions at this time.

The US drillers invented snubbing technology for wells under pressures with sealed wellhead recess in the 1920s [7] and developed its technical equipment, that eliminates the disadvantages of classical method of productive layer silencing. Then in 1960 the snubbing units were substantially improved by the introduction of a volume hydraulic drive, and in 1980 they were manufactured on a self-propelled chassis of the automobile type, which greatly improved their safety, mobility and efficiency. The first snubbing units in Ukraine were introduced in the 1970s.

Snubbing systems can be used to provide a wide range of services today. In essence, a snubbing unit is a hydraulic drilling rig that can do everything that a conventional drilling rig should do, and it can operate under pressure in an unbalanced operating well. Snubbing unit with hydraulic rotary drilling machine can be used for grinding, drilling, side drilling or tasks related to the removal of stubs, cement or deepening of wells [8].

The oil and gas industry have become more responsible for the damage caused by liquids to wells silencing over the years. This helped popularize snubbing systems as a tool for drilling and preparation of wells for extraction, as opposed to their main role as a means of wells service and control. With drilling technology advances on the non-traditional shale market, the advantages of snubbing systems became apparent, since in these types of wells preparation, there is often a need to create lateral branches that extend over hundreds of meters [8].

At present in Ukraine, the use of snubbing facilities for development, capital and current repairs of gas condensate wells is carried out in deposits located in Kharkiv region [9]. Earlier such works were carried out in temporarily occupied territories of Luhansk and Donetsk regions.

The use of snubbing equipment was carried out on wells with depths from 2 500 m to 3 700 m and the pressure corresponding to the hydrostatic at these depths in 2012, as part of a pilot project. All works by the plan were carried out under the direct supervision of emergency rescue detachment representatives of the special purpose of the Ministry of Emergencies in Poltava region, which positively assessed gained experience.

According to the snubbing technology the wells were pre-assembled with insulating plugs, which were installed in the pump-compressor pipes with the help of geophysical equipment [8]. There were compared functional capabilities of traditional units for the repair, testing, development and maintenance of wells and snubbing facilities that actually confirms the fact that there were no operations performed by the first group of technical equipment that cannot be performed by the snubbing units. Snubbing units have a smaller mass and dimensions, characterized by better installation capacity and transportability, with lower operating costs in comparison with the traditional one for equal parameters. Due to the best technical equipment by means of automation, control and management of technological processes, snubbing systems create safer operating conditions for operating personnel, higher production culture and better accident prevention at wells.

If the equipment is properly selected and qualified personnel are employed, then snubbing technologies do not create a greater fountain hazard than traditional methods with productive layer silencing [3]. On the contrary, they create better conditions for the mineral resource protection and environmental safety and indisputable significant economic effect, that is evidenced by global experience in their implementation. For thirty years of its existence, in modern constructive design and standard complete set of snubbing installations became the prevailing technical tool, even in those oil and gas producing regions of the USA and Canada, where before that were powerful fleets of traditional repair units [7].

However, despite the high efficiency of such facilities and technology, their wide application in Ukraine is hampered by a number of items related to regulatory issues.

The main regulatory and legal act on labor protection in this area is the "Safety Rules in the Oil and Gas Industry of Ukraine" [4], approved in 2008. Safety Rules aim to create safe working conditions at wells with fountain or oil and gas manifestation conditions during ongoing and major repairs, and require wells silencing by drilling fluids. Thus, according to Section 12,4 of Section V «Drilling of oil and gas wells», all wells must be silenced if reservoir pressure exceed hydrostatic, those (according to the performed calculations) have the conditions of oil and gas manifestation at reservoir pressures below the hydrostatic. In accordance with the requirements of clause 12.10 of Section V of the Safety Rules, wells equip by slaughter shut-off valves, where work plans not foreseen preliminary silencing, it is necessary to stop the well, decrease pressure to atmospheric and withstand at least three hours.

Isolating plugs used in snubbing technologies perform function similar to a shut-off valve, and therefore, after installing and conducting preventive works aimed to check the absence of excess pressure at the wellhead, works are being carried out on installation of the snubbing unit. Dismantling of the fountain armature is carried out after the visual installation of the discontinuation of gas from the well and verification of the constancy of the level of liquid in it, which meets the requirements of the paragraph of the second clause 2,10 of the current Safety Rules.

Improving normative legal support for oil and gas extraction industry

Observing the regulatory requirements established by Safety Rules (subject to the use of traditional methods), the work performer, before the start of repairs and/or development of wells, shall do:

- delivers at the well the original components and equipment for preparation of silencing liquid or finished liquid in sufficiently volumes and has tanks for its storage;

- orders the contractor or to delivers to own wells mobile pumping units, compressor stations with necessary parameters and in sufficient quantity;

- equips the applied technological complex by necessary equipment for measure and control the processes parameters.

Upon completion the silencing complex works, performer shall do:

- removes the liquid from the well, collects, removes and disinfects it;
- causes influx of formation fluid into the well;
- brings it to a stable yield mode;
- eliminates all environmental damages caused by repair work;
- returns all applicable technical means to the place of permanent base.

Thus, a number of measures carry out in addition to the truly necessary technological operations, for the sole purpose the fountain safety providing.

Safety Rules do not contain a categorical ban on carrying out works on capital and current repairs of oil and gas wells without silencing at wellhead excessive pressure, in general. They only restrict such repairs using coiled tubing installations that significantly inferior to the snubbing, are much more expensive and do not guarantee fountain safety by themselves.

In 2013, there was initiated joint work of scientists, entrepreneurs and civil servants to introduce changes to current Safety Rules in the oil and gas industry of Ukraine. As a result, the elaboration of this legal act was envisaged in the regulatory activity plan for the preparation of draft regulatory acts for 2014. Moreover, a new version of Safety Rules draft was developed. The Draft was brought in line with the requirements of the current legislation of Ukraine and contained issues of snubbing facilities usage. At the same time, Draft containes the provisions of Council Directive 92/91/EEC of 03.11.1992 on minimum requirements for improving the safety and health of workers at the mining enterprises , where the raw material is extracted through wells (eleventh individual Directive within the meaning of Article 16 (1) of Directive 89/391 / EEC) [10] according the plan for implementation of EU acts in the national legislation. The comparative legal analysis of Draft and Council Directive 92/91/EEC is given in Table 1.

Table 1

Compare Draft and Directive 92/91/EEC provisions

Draft provisions	**Directive provisions**
Section IV 2.47. Communication, warning, alarm and control of air pollution	Article 6 The employer shall take the requisite measures to provide the necessary warn-

Draft provisions	Directive provisions
must be working and always ready for action, their performance should be checked at least once a month.	ing and other communication systems to enable assistance, escape and rescue operations to be launched immediately if the need arises.
Section IV 2.36. Workplaces should be kept clean, hazardous substances and waste should be removed.	ANNEX, Part A 2.1.1. Workplaces must be so organized as to provide adequate protection against hazards. They must be kept clean, with any hazardous substances or deposits removed or controlled in order not to endanger the health and safety of workers.
Section IV 1.19. Persons who have special education, have studied and passed OSH in accordance with the requirements of the current legislation may manage drilling, wells development and repair, wells geophysics, and also extraction and oil and gas preparation, facility construction for oil, gas and gas condensate deposit arrangement.	ANNEX, Part A 2.2. A responsible person who has the skills and competence required for this duty, in accordance with the national laws and/or practices, and who has been appointed by the employer, must at all times be in charge of every workplace when workers are present. The employer may personally assume responsibility for the workplace as referred to in the first paragraph, if he has the skills and competence required for the purpose, in accordance with national laws and/or practices.
Section IV 1.18. The employer is obliged to ensure the workers` training and their knowledge of issues labor protection in accordance with the requirements of the Model Regulations on the procedure for conducting training and knowledge verification on occupational safety issues...	ANNEX, Part A 2.5. Workers must be given the necessary information, instructions, training and retraining to ensure their health and safety. The employer must ensure that workers receive comprehensible instructions so as not do endanger their safety and health or those of other workers.
Section IV 1.6. High danger works are performed on the order of admission, its form is given in Annex 1. The list of such works, the procedure for the execution of the admission, and also lists of positions of persons who have the right to manage these works, shall be approved by the enterprise head.	ANNEX, Part A 2.8. There required by the safety and health document, a system of work permits must be introduced for carrying out both hazardous activities and usually straightforward activities which may interact with other activities to cause serious hazards. Work permits must be issued by a person in charge prior to the commencement of work and must specify the conditions to be fulfilled and the pre-

Draft provisions	Directive provisions
	cautions to be taken before, during and after the work.
Section V 3.1.2. (Paragraph 5) In the course of drilling, it is necessary to apply appropriate control devices to protect against sudden emissions.	ANNEX, Part A 5. Suitable well control equipment must be provided for use during borehole operations to protect against blowouts. Deployment of such equipment must take into account the prevailing well and operational conditions.
Section IV 2.54. In gas and dangerous places, a control and measuring equipment for automatic and continuous measurement of the concentration of gases in specified locations, an automatic alarm system, as well as devices for the automatic switching off electrical appliances and internal combustion engines shall be installed. The results of the measurement of the concentration of gases should be automatically recorded and stored.	ANNEX, Part A 6.1. Measures must be taken for assessing the presence of harmful and/or potentially explosive substances in the atmosphere and for measuring the concentration of such substances. Where required by the safety and health document, monitoring devices measuring gas concentrations at specified places automatically and continuously, automatic alarms and devices to cut off power automatically from electrical installations and internal combustion engines must be provided. Where automatic measurements are provided for, the values measured must be recorded and kept as stipulated in the safety and health document.
Section IV 2.55 Harmful substances that may accumulate in air from work areas should be sucked off at the point of their leakage and disposed of outside work areas, or dilute to a safe concentration.	ANNEX, Part A 6.2.1. Where harmful substances accumulate or may accumulate in the atmosphere, appropriate measures must be taken to ensure their collection at source and removal. The system must be capable of dispersing such harmful atmosphere in such a way that workers are not at risk.
Section IV 2.56. In work areas where workers can be exposed to harmful and/or hazardous substances, the necessary amount of ready-to-use personal respiratory protective equipment should be provided. The PPE choice is carried out in compliance with the Rules of choice and application of means of individ-	ANNEX, Part A 6.2.2. Without prejudice to Directive 89/656/EEC, appropriate and sufficient breathing and resuscitation equipment must be available in areas where workers must be exposed to atmospheres which are harmful to health. In such cases, a sufficient number of workers trained to use such equipment must be present at the workplace.

Draft provisions	Directive provisions
ual protection of respiratory organs…	The equipment must be suitably stored and maintained.
Section VIII 1.2. Production facilities for the exploration and construction of oil, gas and gas condensate deposits containing hydrogen sulfide and other harmful substances should be identified according to the classes of hazard of possible emissions and leakage of vapors and gases into the atmosphere in accordance with the requirements of the Procedure for the identification and accounting of high-risk objects. 1.3. Oil and gas company should develop a plan of measures for population and environment protection within the sanitary protection zone, as well as within the contour of the deposit.	ANNEX, Part A 6.2.3. Where hydrogen sulphide or other toxic gases are or may be present in the atmosphere, a protection plan detailing the protective equipment available and the preventive measures taken must be held at the disposal of the competent authorities.
Section IV 2.6. Emergency routes, emergency exits and approaches to them should be free of any items, provide the shortest route to a safe area or a safe evacuation point.	ANNEX, Part A 7.1. Emergency routes and exits must remain clear and lead by the most direct means to the open air or to a safe area, a safe assembly point or a safe evacuation point. 7.2. In the event of danger, it must be possible for workers to evacuate all workstations quickly and as safely as possible.
Section IV 2.7. The number, location and size of the evacuation routes and emergency exits are determined in accordance with the use, arrangement and size of the working areas, as well as the maximum number of persons that can be there according to building regulations. 2.8. The doors of the emergency exits must be opened outside and locked so that, if necessary, any person can easily and quickly open them without the use of additional	ANNEX, Part A 7 7.3. The number, distribution and dimensions of the emergency routes and exits depend on the use, equipment and dimensions of the workplaces and the maximum number of persons that may be present. Accommodation and rest rooms must have at least two separate escape routes situated as far apart as possible and leading to a safe area, a safe assembly point or a safe evacuation point. 7.4. Emergency doors must open outwards or, if this is impossible, be sliding doors.

Draft provisions	Directive provisions
means. 2.9. Ways of evacuation, emergency exits and approaches to them should be marked with signal colors, signs, inscriptions, signs of safety in accordance with the Technical Regulation of the signs of safety and health protection of workers... 2.10. Ways of evacuation, emergency exits must have sufficient emergency lighting.	Emergency doors should not be so locked or fastened that they cannot be easily and immediately opened by any person who may require to use them in an emergency. 7.5. Specific emergency routes and exits must be indicated by signs in accordance with the national regulations transposing Directive 92/58/EEC into law. 7.7. Emergency routes and exits requiring illumination must be provided with emergency lighting of adequate intensity in case the lighting fails.
Section IV 2.32. In closed workrooms a ventilation system that provides air exchange of standard multiplicity should be arranged, taking into account the nature of the production process and the physical load on the workers. Any disturbance of the ventilation system must be immediately displayed by the signaling devices. Ventilation devices must be installed in such a way as to exclude drafts. Deposits or contaminants that could pose a health hazard to workers due to air pollution should be removed immediately.	ANNEX, Part A 8.1. Steps shall be taken to ensure that there is sufficient fresh air in enclosed workplaces, having regard to the working methods used and the physical demands placed on the workers. If a forced ventilation system is used, it must be maintained in working order. Any breakdown must be indicated by a control system where this is necessary for workers' health. 8.2. If air-conditioning or mechanical ventilation installations are used, they must operate in such a way that workers are not exposed to draughts which cause discomfort. Any deposit or dirt likely to create an immediate danger to the health of workers by polluting the atmosphere must be removed without delay.
Section IV 2.33. The microclimate of industrial premises during working hours must meet the requirements of the DSN 3.3.6.042-99 "Sanitary norms of the microclimate of industrial premises".	ANNEX, Part A 9.1. During working hours, the temperature in rooms containing workplaces must be adequate for human beings, having regard to the working methods being used and the physical demands placed on the workers. 9.2. The temperature in rest areas, rooms for duty staff, sanitary facilities, canteens and first aid rooms must be appropriate to the particular purpose of such areas. 9.3. Windows, skylights and glass partitions should allow excessive effects of sunlight in workplaces to be avoided, having regard to the nature of the work

Draft provisions	Directive provisions
	and of the workplace.
Section V. 5.3. (Paragraph 4) When drilling for gas and pressure oil-saturated horizons it is necessary to provide remote control of devices for blocking wells and pipelines and reducing pressure in them in the event of an accident.	ANNEX, Part B 2. Where required by the safety and health document, certain equipment must be capable of remote control at suitable locations in the event of an emergency. Such equipment must include systems for the isolation and blowdown of wells, plant and pipelines.
Section IV 2.46. Communication, warning, alarm and control of air pollution must be working and always ready for action, their performance should be checked at least once a month. 2.47. The alarm signal supplied by the alarm system must be well audible in all places where workers may be.	ANNEX, Part B 3.2. Facilities for raising the alarm must be provided at suitable locations. 3.3. When workers are present at workplaces which are not normally manned, appropriate communication systems must be placed at their disposal.
Section IV 2.48. The location of the workforce in the event of an alarm and the routes to these places should be determined.	ANNEX, Part B 4. Where required by the safety and health document, safe assembly points should be specified, muster lists should be maintained and the necessary action should be taken.
Section IV 2.50 If the movement along the life-saving paths is complicated and there is possibility of creating an atmosphere that is not suitable for breathing, it is necessary to provide for self-restraint for use directly at the workplace.	ANNEX, Part B 5.3. Where escape routes are difficult and where irrespirable atmospheres are or may be present, self-contained escape apparatus must be provided for immediate use at the workstation.

Separate profile companies Naftogaz of Ukraine have considered the Draft and provided feedbacks. However, the procedure for approving this draft legal act did not take place due to the reorganization of executive bodies, including State Labor Service of Ukraine.

The State Labor Service of Ukraine has returned to the issue of revising and elaborating a draft of the new edition of the Rules. To date, the draft Rules and its accompanying documents have been made public on the official site of the State Labor Service for the purpose of receiving suggestions and comments.

It is worthwhile to note that in the part of application of snubbing systems, a new edition of the Rules is recommended to be supplemented with provisions concerning:

- the appointment of snubbing facilities for work on the development, capital and current repair of oil and gas wells without silencing at excess pressure on the wellhead;
- obligatory compliance with the technical specifications and operating instructions of the manufacturer in the site preparation, installation and operation of the snubbing equipment, the mandatory availability of the necessary documentation and personnel who have been trained in the prescribed manner; as well as directly requirements to the snubbing unit.

The main changes to Safety Rules [4] regarding the use of snubbing systems are given in Table 2.

Table 2

Comparison table amendments to Safety Rules

Current edition	Proposed changes
3. Coiled tubing installations	3. Coiled tubing and snubbing installation
	3.4. Snubbing installations are intended for carrying out works on development, capital and current repair of oil and gas wells without silencing with excessive pressure on the wellhead. 3.5. The site preparation, installation and operation of the snubbing installations must be carried out in accordance with the technical specifications and operating instructions of the manufacturer. Before starting work, the unit must be equipped with: the necessary documentation, staff equipment and tools, instructions for safe operation. Work with the use of snubbing equipment is performed by personnel who have undergone training in accordance with the established procedure. 3.6. Requirements for the snubbing unit: a) the snubbing unit must be equipped with emergency and work equipment; b) before the start of work in the precursors are installed dies, which correspond to the size of the column of pipes located in the well; c) after the installation of emergency and work-

Current edition	Proposed changes
	ing pretensioners, they are pressed for pressure in accordance with the plan of work, but not higher than the worker; d) the unit must be staffed with a complete set of tools for repair of the pre-installation and installation in general; e) Before and after the well repair works, a revision of all units and aggregates shall be carried out.
12.4 Before the elevator is installed on the well, the well must be silenced. The silencing must be carried out with a solution with a density that meets the requirements of subclause 3.3.3 of clause 3.3 of chapter 3 of section V of these Rules. All wells where formation pressure exceeding hydrostatic and wells where conditions are fountaining or gas and oil manifesting (according to the performed calculations) at reservoir pressures lower than hydrostatic.	12.4 Before the elevator is installed on the well, the well must be silenced. The silencing must be carried out with a solution with a density that meets the requirements of subclause 3.3.3 of clause 3.3 of chapter 3 of section V of these Regulations. All wells where formation pressure exceeding hydrostatic and wells where conditions are fountaining or gas and oil manifesting (according to the performed calculations) at reservoir pressures lower than hydrostatic. If the snubbing system is used, the well is not subject to preliminary silencing. A special sealing unit (isolating plug, etc.) is installed in the NCP tubular space.
12.10 Before dismantling wellhead fittings, the pressure in the pipe and annulus should be reduced to atmospheric pressure. Wells equipped with a shut-off valve, those do not foresee a previous shutdown by workplan, must be stopped, pressurized to atmospheric, and last for at least three hours. Dismantling wellhead fittings is carried out after the visually established discontinuation of gas from the well and checking the constancy of the level of the fluid in it.	12.10 Before dismantling wellhead fittings the pressure in the pipe and annulus should be reduced to atmospheric pressure. Wells equipped with a shut-off valve, those do not foresee a previous shutdown by work-plan, must be stopped, pressurized to atmospheric, and last for at least three hours. Well, that develops by capital and current repair is carried out with the use of the snubbing unit, and work plan does not foresee a previous shutdown, after installing the sealed knife, must be stopped, pressurized to atmospheric, and last for at least three hours. Dismantling wellhead fittings is carried out after the visually established discontinuation of gas from the well and checking the constancy of the level of the fluid in it.

It is worth exploring the production mastering possibility by domestic machine building to provide the domestic market equipment for the implementation of snubbing technologies, that, provided availability of the necessary design, technological documentation and financing for production development, could increase the economic effect of snubbing technology for the state.

Conclusions. Summarizing the above, it should be noted that the application in Ukraine of up-to-date technologies of carrying out works in wells is possible after provided appropriate changes to Safety Rules for wells` operation, technical adaptation of domestic wells to ensure their usage for snubbing technology, training and certification of technical personnel, who will work these technologies.

In turn, rapid and widespread introduction of snubbing technologies will have a tangible effect in oil and gas extraction, in the social sphere and in the national economy as a whole.

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APPLICATIONS OF REMOTE SENSING AND GIS IN MINERAL EXPLORATION-A RESOURCE-SAVING TECHNOLOGY

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Mineral exploration is a task that we need to approach with maximum information. Missing out on rare metals and minerals can easily occur, and the process of searching for them is a costly risk. That is one of the reasons for remote sensing in mineral exploration being so important. Remote sensing can be used to measure, variations in acoustic wave distributions, force distributions and also electromagnetic energy distributions. The latest progress in the field of remote sensing and origin of new computer software such as Geographical Information System (GIS), ENVIS (Environmental Information Software) have transformed the world and made life much easier for mineral explorers.

1.0 Introduction

"Remote Sensing is a technique in which data are acquired without any direct contact to derive information about objects or materials (targets) located on the earth surface or its atmosphere" (Lillisand, T. M and Keifer, R. W., 1979).

"Remote sensing is the art and science of acquiring information about the Earth's surface without actually being in contact with it. Reflected or emitted energy is sensed, recorded, processed, analyzed and applied to gather that information" (CCRS/CCT).

In general, remote sensing involves seven steps (figure 1):

Remote sensing is done using sensors primarily used to detect the reflected or emitted electromagnetic radiation (EMR). These sensors are installed on vehicles, called platforms and operate with the help of cameras and scanners.

Ranges in EMR are of the primary importance like visible region in the range of 0,4-0,7 μm (Blue: 0,4-0,5 μm, Green: 0,5-0,.6 μm, Red: 0,6-0,7 μm.). Other bands include Ultraviolet (UV) region (adjoins the blue end), Infrared (IR) region (adjoins the red end), Microwave region (longer wavelength intervals ranges from 0,1 to 100 cm, etc.).

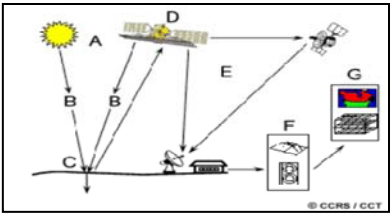

Figure 1. Steps in Remote Sensing (After CCRS/CCT)

Energy Source or Illumination (A)
Radiation and the Atmosphere (B)
Interaction with the Target (C)
Recording of Energy by the Sensor (D)
Transmission, Reception, and Processing (E)
Interpretation and Analysis (F)
Application (G)

The images generated in remote sensing are broadly utilized for mineral exploration in mapping the geological features and lineaments and identifying rocks by their spectral signature. Images are gathered either through synthetic aperture sensors or optical sensors. Synthetic aperture sensors can sense electromagnetic data by transmitting microwaves and receiving the reflected back waves from the Earth's surface while optical sensors measure the spectral data of sunlight reflected from the surface of the Earth.

Reconnaissance lithologic mapping is usually the initial step of mineral resource mapping. Aforementioned is complimented with structural mapping, as mineral deposits often occur along or adjacent to geologic structures, and mapping variation, as mineral deposits are usually linked with hydrothermal changes of the surrounding rocks. In addition to these, knowing the use of multi and hyperspectral remote sensing is crucial as multi and hyperspectral data can help distinguish and thematically map regions of exploration interest by using the discrete absorption features of most minerals. Eventually coming to the mineral exploration stage, GIS stands a useful tool in assimilating and analyzing various georeferenced available data in picking the best sites of mineral deposits.

The exploration of mineral has a very high-cost component and involve intensive labor work, except if they are highly automated. Nowadays, in developed nations, high-end machines perform services in all stages of detections and productions. In underdeveloped countries like Nigeria where technology is deficient, the work is unendurable and clumsy, and hence little exploitations are made. Various studies of mineral deposits showed that much of these countries natural resources are unexploited. Mineral resource mapping can change the entire situation using an integrated application of three geospatial technologies (Remote Sensing, GPS and GIS).

The study area is captured and digitally analyzed using any GIS software like ArcGIS and QGIS. GPS receiver is used to categorize ground sections for the collected data. Geometric corrections, supervised or unsupervised classification and setting quantitative relationships between spectral signatures and ground features that are indicators of appearance or absence of mineral deposits can also be done effectively using these tool.

The applications of aerial photo presentations on a variety of areas, viz, land cover mapping, soil and geologic mapping, zones of mineralisation, wetland mapping, wildlife ecology, archaeology, agriculture, forestry range, land management, water resources, urban and regional planning, environmental assessment, landform identification and evaluation is the real-time programme of various agencies.

With the help of remote sensing, faults, geological contacts and fractures are brought out clearly which help in prospecting mineralized areas. It has proven to be a valuable tool in exploring the mineral resources and separating the favorable areas from unfavorable areas. Remote sensing data provide the lithological, geomorphological and structural guides essential for understanding various parameters responsible for the localization of most of the ore deposits. (Rawashdeh, S.A 2007).

2.0 History and Development

It has been over seven decades that remote sensing technology has been in use but for mineral exploration it has gathered a rapid pace with the advent of firstly taking aerial photographs and doing compare-and-contrast, matching the photos with the findings of gold on the surface and looking for similar features to find gold, and then satellite imagery in the recent years after it became commercially feasible and available, the same compare-and-contrast method was used with satellite images also. Since 1972, satellite remote sensing, combined with other exploration techniques, has proved operational exploration and engineering cost saving and reduced exploration risks through better geological mapping. Land and ocean remote sensing satellite systems have significantly increased our ability to explore, prove, and manage energy and mineral properties globally. The use of remote sensing has been there since over a decade but more recently with the introduction and upgradation of GIS structural and lithological mapping have developed to a very high level, in all the mineralized areas. It has also been observed that new age sensors can map efficiently lithologic and stratigraphic units and identify alteration in rocks, soils, and vegetation cover indicative of undiscovered subsurface minerals. Mapping and monitoring of resource development can also be done using the same sensors. The use of satellite remote sensing data along improved geographic information systems (GIS) has grown substantially through evolving integration with other geophysical, geochemical, and geological data.

3.0 Exploration process using Remote Sensing and GIS

Remote Sensing, GPS and GIS serve as a tool for mineral resource monitoring, and mapping. Remote Sensing has some major advantages at gathering the information, that makes it a competent tool in mineral deposits exploration and extraction, then using GIS to develop geological maps of mines. Collecting and handling data in the field is just the initial step in coming up with an output used in exploration. The next step is to modify the data into information that is presentable and understandable through various image visualization techniques that include, Visual interpretation, Digital processing, Preprocessing, Image Enhancement, Transformations, Classification and Integration.

Remote sensing systems make use of spectral signatures of the features. Every material behaves differently to EMR depending upon their chemical and structural properties, the amount of EMR it reflects absorbs, transmits, emits and varies with wavelength. The amount of radiation from a material is plotted over a specific range of wavelengths. These points are connected and create a curve named as the material's *'spectral signature'*., It depends on the different properties like the material targeted, the incident energy (angle, intensity and wavelength) and this uniqueness of the reflected or emitted EMR used to detect and discriminate the objects or surface features.

Every object has a unique spectral signature, and similar objects share a similar spectral signature. Hence, one can identify the feature and its characteristics to a great extent with their spectral signatures. An object's spectral signature can be learned using remote sensing sensors data which is need to be analyzed manipulated and processed. Most of this is accomplished by altering spectral color bands of the image.

Spectral signatures form a Spectral Reflectance Curve is a graphical representation of the spectral response over different wavelengths of the electromagnetic spectrum. It provides an in-depth knowledge of spectral characteristics of different rocks or material and also useful to select a specific wavelength band for remote sensing data acquisition, as shown in figure 2. These curves are crucial to interpret and analyze an image obtained in single or numerous wavelengths. Spectral rationing can be carried out on selected bands to improve the spectral resolution of an image using sensors like Landsat 4-5 Thematic Mapper (TM) on the EM spectrum. The quotient images are then aligned and converted into formats like RGB, which is a color composites format. Single TM bands can also be chosen for color composites to increase the classes of the surface materials spectrally as logged by the digital number (DN) values.

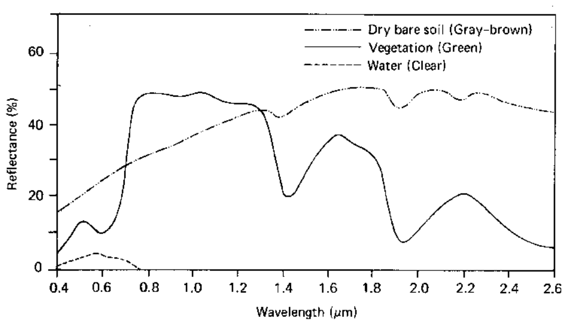

Figure 2. Typical spectral reflectance curves for vegetation, soil and water (After Lillesand et al., 2004)

4.0 Processes involved in mineral exploration

Mineral targeting consists of the utilization of geological traits that control their emplacement. Finding these geological controls is usually viewed within the framework of the parameters that are linked with the depositional and post-depositional processes that have influenced them. Exploration methods are typically designed to take care of the peculiarities of all observable surface geological evidence that can be used to recognize areas of likely mineralization. The use of remote sensing is intended at digitally analyzing and manipulating the information in the imagery, which can be directly associated with the surface processes like weathering and alterations that are connected with mineral deposits. All the geologically associated information generated from the imagery from the database for a GIS-based extraction of the most encouraging areas. Specific attributes that are deemed useful for exploration of the Geology and Mineral Potential is the rock outcrops and lineaments concentration. Aforementioned is a direct outcome of the likely mineralized area as well as vegetation cover designating healthy plant growth as a cause of soil fertility as a result of soil enrichment with minerals. In a situation where the exposures are reduced, mapping usually involves indirect methods such as; Structural evidence, Geochemical evidence, Inference from geological mapping and Geophysical evidence.

4.1 Location of the Study Area

A piece of detailed information regarding the Land Cover area and Latitudes-Longitudes of the area under study. Longitudes and Latitudes are taken and recorded. Remote sensing ensures the availability and rapid access to real-time geospatial data.

4.2 Digital Image Processing (DIP)

The images are firstly pre-processed before the input of the images used. The supplied/input images used can be arranged via various agencies collecting the remote sensing data of that particular area. In countries, government organisations can provide the required data for many cases. The next step involves the preparation of the dataset related to the area. Satellite orbit information can be used to improve the imagery Ground Control Points (GCPs), if the Digital Elevation Model (DEM) for the area is not present. Imagery rectification can be done to present already geocoded SPOT Multispectral data or Landsat MSS(image -to- image geocoding) using the Universal Transverse Mercator(UTM) coordinate system, Clarke 1880 spheroid, or as required in a region depending upon the location and properties. After developing the images of data needed (training sites), supervised classification (or unsupervised classification) can be carried out to recognize the arrangement of the distribution of the various spectral classes in the images.

Lineaments play critical roles in controlling mineralization. The study of lineaments and their interpretation is necessary for the geological presumptions that have relevance and links to mineral exploration targets (Odeyemi, 1999). Thus, the images are sensibly studied for such features and subsequently subjected to Lineament Density Analysis (LDA) using a pixel size of our requirements. The lineament density is used to aggregate and calculate the total lengths per square kilometer using the appropriate Algorithm. This involves digitization of linear and curvilinear features as segments from properly filtered ETM bands, e.g. considering LanSat7 Bands 5 and 4 are best suited for picking linear features. Enhancement involves not only edge enhancement but also directional filtering. The procedures made the structural features adequately visible for digitization. As these features are digitized, they are saved as layers. These layers go through the overlay process with other segments such as geological structures maps, roads, rivers and other layers for lineaments identification that is then interpreted as geologically related (Short 2001). Any Remote Sensing and GIS software can be used for the lineament density analysis. Utilizing a pixel size of a specific area, the lineaments per square kilometer is automatically calculated whereas the attribute of each segment such as length and orientation and serial number are automatically created and stored in the vector attribute cells. Using the excel functions like histogram, the lineaments can also be automatically aggregated to be displayed in a rose plot.

Lineament density analysis is a typical job in most geological applications of structural controls to mineralization. Zones of intersections and trends are usually sorted after in explorations because it is premised on the fact that mineralization is structurally controlled.

Digital Image Processing is divided into two parts. Digital Image Enhancement where Color Composites, Contrast Stretching, Filtering and Edge Enhancement, Density Slicing, Thresholding, IHS, Time Composite and Synergic Images are done and Digital Image Processing and extraction where Supervised Classification, Unsupervised Classification, Fuzzy Classification, Image Transformation, Principal Component Analysis, etc. are carried out. Some of the image processing software like ERDAS Imagine, ILWIS, ENVI, ArcGIS, MATLAB, etc. are very useful for Digital Image Processing.

5.0 Conclusion

The applicability of geospatial technology helps in mineral exploration by presenting extensive land cover information about mineral a mapping. The spectral properties of soil cover can be known and also the tectonic information delivered about by the distribution of the lineaments features which complement the selection of the promising areas for comprehensive mapping. The benefits of geospatial technology, especially remote sensing, have introduced new extents into the study and understanding of earth's processes and even vegetation cover, which has direct relationships with the deposits. A necessary pre-requisite to joining in these opportunities is the building of various indigenous capacities and competencies for the development and utilization of science and technology (ARCSSTE-E 2007).

The main advancement in mineral exploration is the ability to integrate various forms of data with the help of computers. Noted drill results can be easily integrated with topographic maps, structural maps, air photos, and many other details like ore grade data. Data synthesis can considerably improve the precision and effectiveness of a mineral exploration process.

Remote sensing is a mainstay in mineral exploration and for a good reason. Advancements in data processing and remote sensing technology will proceed to allow explorers to take more calculated gambles and assess their mining progress and mineral exploration with a sense of certainty.

It can be inferred that the various computer hardware and software are along with the latest technologies of remote sensing and information technology, are highly valuable and essential for the understanding and assessment of the present or current status of the existing geological resources in any part of the world.

Use of computer models have also been useful in the interpretation of the past and present data as well as predicting the future status of these resources in any region which help in implementing the scientific methods for the management and conservation of and accurate exploration of these mineral resources.

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INNOVATIVE RESOURCE-SAVING METHOD OF EXPLOSIVE DESTRUCTION OF COMPLEX-STRUCTURAL FERRUGINOUS QUARTZITES

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Abstract.
Massifs of iron ore quartzites are complex folded structures, dissected, as a rule, by three systems of mutually perpendicular cracks. The layers of iron-rich layers (ore quartzite), crumpled in the so-called flaking folds, often alternate with practically barren rocks (shale, low ore magnetite quartzite, etc.). At the same time, the rocks differ significantly both in their mineralogical composition and internal structure, and in their strength properties. When drilling blastholes in areas where iron-rich areas of ferruginous quartzites coexist with barren rocks, borehole explosive charges cross contact zones when the contact plane is horizontal or at an angle to the horizontal plane. As a result, after the explosion, a dilution of iron-rich ore from waste rock is observed. Thus, the complex structure of ferruginous quartzites, namely, the interbedding of iron-rich quartzites with their barren varieties led to the creation of new resource-saving methods for the explosive destruction of ferruginous quartzites.

Introduction

Ferruginous quartzites developed as iron ore raw materials differ significantly in their structure and strength characteristics. Strong quartzites with high iron content are often in contact with less durable barren quartzites or shales. With the explosive destruction of such areas of iron-ore massifs in the contact zones, there is a dilution of minerals with low, and often with practically zero, iron content. Dilution of minerals, in this case, iron ore can be reduced by applying for crushing rocks in quarries special constructions vertical borehole charges based on reducing the dynamic loads in the contact zones. As a result, the mixing of the rock with the ore component and the oreless rocks is significantly reduced, which contributes to improving the quality of the explosive preparation of the rock mass used as ore raw materials. Based on the use of a new wellbore explosive charge design, we have developed a new resource-saving method for the explosive destruction of ferruginous quartzites of complex structure, the essence of which, as well as the results of industrial testing in the conditions of Krivbass iron ore quarries, are given below.

Methods of research.

When developing a new resource-saving method of explosive destruction of ferruginous quartzites of a complex structure, standard methods were used to determine the strength properties of rocks, as well as light optical microscopy methods to establish the granulometric characteristics of quartzites destroyed by the explosion.

Experimental studies of physic-mechanical properties and structural features of iron ore rocks

For studies of the physic-mechanical properties and the specific surface energy of rocks destruction of the iron ore deposit (Krivoi Rog), samples were taken, which are described in Table 1. The study of the strength characteristics of rocks, such as density, uniaxial compression strength, speed of longitudinal and transverse waves, Poisson’s ratio and Young's modulus was carried out in accordance with current standards [1-5] on models made from selected geological exploration cores.

For each test series, 10 cubic models with a 40±2 mm rib and a cylindrical shape with a height equal to the core diameter (core diameter 55-57 mm) were prepared.

The edges of the samples were processed with grinding powder, while their curvature did not exceed 0,05 mm.

Table 1

Information on samples of iron ore (Krivyi Rih), selected for the study of physical and mechanical properties and specific surface energy of their destruction

Type of rock	Sampling point	Characteristics of the rock
Magnetite quartzite thin-layer	Central MPP, quarry №3, geological exploration core, diameter 43 mm	Thin-coarse, fine-grained, thin-banded texture, durable
Iron ore quartzite is fine-grained	Ingulets MPP geological exploration core, diameter 43 mm	Medium, massive, fine-grained

The prepared rock samples were tested for uniaxial compression on a hydraulic press PR-500 using an approved method for determining the tensile strength for uniaxial compression [3]. Pressure control was carried out with a manometer, and transverse strain measurement was measured with displacement strain gauges. The information obtained in the automatic mode arrived at the computer complex for the corresponding processing and construction of the diagram "compression stress - strain (longitudinal or transverse)".

The test results were also entered in tabular form into a computer database. The tensile strength of the sample was calculated by the formula, MPa

$$\sigma_{comp}=P_{\max}/S \tag{1}$$

where $S_0=a^2$ is the cross-sectional area of the cubic form sample, cm^2;

- is the average size of the sample face, cm, and the cross-sectional area of the sample is cylindrical (core) - $S_0=\pi d^2/4 \sim 0.785d^2$,

- sample diameter, cm.

The study of the rocks acoustic properties -the propagation velocity of longitudinal and transverse waves -was carried out on a stand developed in IGTM NAS of Ukraine (Fig. 1, [4]).

Fig. 1 - General view of the stand to determine the propagation velocity of longitudinal and transverse waves with an assembly module for positioning a rock sample with sensors

The stand includes a portable pulsed ultrasonic device with piston-type piezo-transducers in a protective casing made of axially polarized ceramics, oscillographic indication and automated processing of the received oscillograms of longitudinal and transverse waves. When using this device, the processing of test results was carried out using a validated method for determining the propagation velocity of longitudinal and transverse waves [5]. The contact of the piezoelectric transducer with the rock sample was provided by lubricating the surface with technical petroleum jelly of a high degree of cleaning.

Measurements of the velocity of propagation of elastic waves were carried out on three pairs of sample faces, then their average value was determined. The obtained average calculated values of the velocity of propagation of elastic waves with an accuracy of no more than 10 % and a degree of reliability of 0.95 were achieved by testing at least 10 samples for each type of rock.

Young's modulus based on the measured values of the propagation velocity of elastic waves in a rock sample was determined by the formula [4]

$$E = \rho C^2_p, \tag{2}$$

where ρ is the density of the rock sample, kg/cm^3;

C_p - longitudinal wave velocity, m/s.

The results of studies of physical and mechanical properties of rocks are given in Table.2.

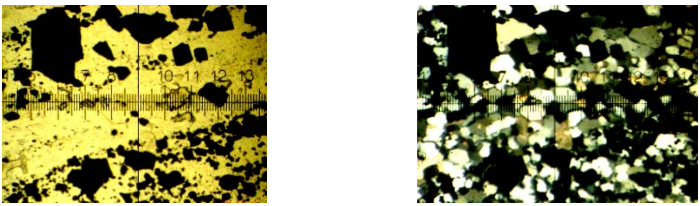
Table 2

Basic physical and mechanical properties of rocks of the iron ores deposit

Type of rock	Density, $\rho \cdot 10^{-3}$, kg/m^3	Compressive strength, σ_{comp}, MPa	Longitudinal wave speed C_p, m/s	Poisson's ratio, ν	Young's module, E, MPa
Ferruginous quartzites fine-grained	2,10	85,5	3600,0	0,21	45416
Magnetite quartzite thin-layer	3,0	300,0	5600	0,39	100500

From the prepared sections of rocks of different genesis produced thin sections, polished sections on a grinding machine. The study of the structural features of ferruginous quartzites using a polarizing microscope in transparent petrographic thin sections (see Table 1) made it possible to establish the following.

The texture of ferruginous quartzite is layered, fine-grained. The size of mineral aggregates and individual grains: strips of aggregates of quartz grains – strip thickness 300-400 microns; individual grains of quartz in the strip -30-100 microns: magnetite grain stripes - no solid stripes, individual magnetite grains, ranging in size from 20 to 300 microns are bordered with fine quartz grains (20-50 microns), the width of such a combined strip is 300-00 microns (like pure quartz stripes (Fig. 2).



a *b*

Fig. 2 – The structure of thin-strip magnenite quartzite in transmitted (*a*) and polarized (*b*) light – Kryvyi Rih, Central MPP: (dark grains –- magnetite; light – quartz), micrograph of a transparent section, magnification 150×

Mineralogical composition. Quartz and magnetite approximately equal in the 1:1 ratio make up 95 % of the minerals of quartzite. In the field of view of the microscope, there are individual transparent grains of ferrocalcite (not more than 0,5-1,0 %). Structure under the microscope is thin-striped. Secondary changes are absent.

Estimated changes specific fracture surface energy of rock samples of iron ore deposits in their dynamic loading

Experimental studies in polygon conditions to assess the nature of the destruction and the specific energy of formation of a new surface during the destruction of rock samples during the explosion of explosive charges on models were carried out according to the modeling methods developed by us [6, 7]. The research methodology provided for two series of experiments: evaluation of the specific surface energy of rock samples destruction by an explosion perpendicular to the layering; assessment of the specific surface energy of rock samples destruction by an explosion parallel to the layering.

In accordance with the research methodology, rock samples were taken (geological exploration cores with a diameter of 55-57 mm) were carried out on the ledges of quarries outside the zone of influence of dynamic (explosive) loads, and their preparation was carried out in a laboratory.

To determine the specific surface energy of explosion damage in the prepared models, an explosive cavity was drilled into the center of one of the faces with a diamond drill at 2/3 of the height of its edge, 5-6 mm in diameter. The prepared samples (control and experimental) were destroyed, under polygon conditions, by a dynamic load of high intensity (explosive destruction of models) in a special explosion chamber (Fig. 3 and 4).

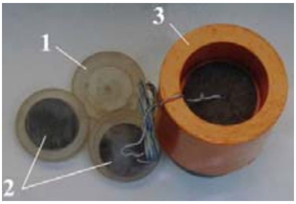


Fig. 3 – General view of the conductor, rock sample and the explosion chamber: *1* - conductor; *2* - rock sample; *3* - blast chamber

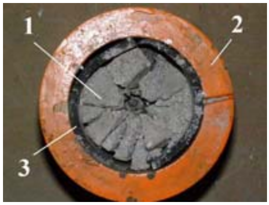


Fig. 4 – Rock sample in the explosion chamber after destruction: *1* - sample; *2* - blast chamber; *3* - damping gasket.

The mass of explosive charge in all series of experiments was $150 \cdot 10^{-6}$ kg, the specific consumption of explosives was 2,42 kg/m^3. Characteristics of the studied samples are given in Table 3.

Table 3

Characteristics of rock samples

	The average size of the bream rock sample, h_{av}, cm	Volume, V, cm^3	Mass, m, g	Density, ρ, kg/cm^3	Output surface of the sample, S_0, cm^2
Ore quartzite	4,1	69,0	210,0	2,83	101,0
Magnetite quartzite thin-layer	4,0	64,0	229,5	3,57	96,0

The estimation of the change in the specific surface energy of the destruction of rocks models under the action of the explosive energy of the cylindrical charge of the explosive was performed by sieve analysis according to known methods [8] using a set of laboratory sieve-type with a hole size of 12.0; 10.0; 7.0; 5,0; 3.0; 2.0; 1.0; 0,50 and 0,25 mm.

The newly formed surface of the destroyed samples, after their separation into fractions for determining the particle size distribution, was calculated by the formula [8]

$$S_н = \frac{6}{\rho}\sum_{i=1}^{n}\frac{m_i}{d_i} - S_0, \qquad (3)$$

where ρ is the density of rock samples (g/cm^3); m_i, d_i –- mass (g) and diameter (cm) of the average piece of the *i*-th fraction, respectively; S_0 is the initial surface of the sample (cm^2), and the diameter of the middle piece was determined by the formula

$$d_{cp} = \sum_{i=1}^{i} w_i d_i \,, \tag{4}$$

where w_i=m_i/m is the content of the i-th fraction or the *i*-th piece, in fractions of a unit; mi is the mass of the i-th fraction, g; m is the total mass of all fractions, g; d_i - the average size of a piece of the *i*-th fraction, cm.

The energy intensity of destruction for the formation of a unit of new surface is estimated by the value of the specific surface energy (γ). When samples of rocks were destroyed by an explosion of an explosive charge of mass *M* and heat of explosion *Q* (kJ/kg), the energy expended to form a unit of the newly formed surface – γ_n was calculated using the formula

$$\gamma_n = MQ/S_{new}, \tag{5}$$

where S_{new} is a new newly formed surface during an explosion, cm^2;

- the mass of high blasting explosives (of the ten type) - 150·10^{-6} kg;
- the heat of explosion of this explosive 5908 J/kg.

The degree of rock samples fragmentation was determined by the formula

$$K_f = h_{\mathrm{av}}/d_{av}, \tag{6}$$

where h_{av} - the averaged size of the edges of the sample model, cm;

$_{av}$, - the diameter of the middle piece.

Experimental studies on the assessment of energy costs for the destruction of rock samples by an explosion on the formation of a new surface and the nature of their fragmentation were carried out in polygon conditions at the Rybalsky granite quarry (Dnepr). A high blasting explosive (PETN) placed in a cylindrical cartridge or a piece of a detonating cord (12 g/m) connected to an initiator and installed in a charging cavity was used as an explosive.

The prepared rock models were placed in an explosion chamber, the inner surface of which was lined with a rubber gasket to prevent secondary crushing of the fragments of the models destroyed by the explosion against the walls of the explosion chamber (see Fig.4 and 5).

The assessment of the particle size distribution of the destroyed model was carried out according to the following main indicators: the nature of the destruction of the model as a whole, the diameter of the middle piece and the area of the newly formed surface. In this regard, the amount of energy expended on the destruction of the medium can be indicative only with the same loading parameters and sizes of the test specimens.

Based on the calculated values of the newly formed surface of rock samples destroyed by an explosion of an explosive charge, the following indicators were determined: the specific surface energy of destruction, the diameter of the middle piece and the degree of crushing of rock samples by formulas (3)-(5). The results of experimental studies on the assessment of particle size and energy characteristics of rock samples destroyed by dynamic loads are given in Table 4, and the cumulative curves of particle size distribution under various loading conditions are shown in Fig. 6.

Table 4

Granulometric and energy characteristics of rocks samples destroyed during explosive loading

Type of rock	The average size of the bream rock sample, h_{av}, cm	Area of newly formed surface, cm^2, S_n	Average size of piece, dsp cm	Specific surface energy of destruction, J/cm^2	Degree of crushing sample $K_{cr} = h_{av}/d_{\text{av}}$
Ferruginous quartzite (ore quartzite)	4,1	1484,3	0,56	0,60	7,32
	4,0	1767,3	0,72	0,50	5,55
	4,3	1353,0	0,54	0,66	8,0
Magnetite quartzite thin-layer	3,94	916,3	0,93	0,96	4,23
	4,0	1113,4	0,82	0,80	4,90

Experimental investigations of the influence of dynamic loads on the change of the ferruginous quartzites structure

In order to study and evaluate structural changes in anisotropic ferruginous quartzites under dynamic loads (explosion, impact), studies were conducted in the field conditions of the Rybalsky quarry (Dnepr). Assessment of structural changes in ferruginous quartzites (ore quartzite and thin-strip magnetite quartzite, see Table 1), destroyed by dynamic loads, was carried out on models from natural materials.

The products of destruction after dynamic loading of models (explosion) were scattered on laboratory sieves with cell sizes of 400, 315, 160, 100 and 50 microns and the mass of each fraction was determined. The size distribution of the smallest fraction (0-100 microns) was established using a polarization microscope equipped with an integration table for quantitative measurements. For the automatic construction of cumulative curves from microgranulometric measurement data, programs in the BASIC language were used. The data were processed by the method of approximation of experimental curves and approximated by a two-parameter dependence

$$C = D/(a + bD), \tag{7}$$

where C is the total fraction of particles in the fraction,%; D - particle size, microns; a and b are parameters that have specific values for each diameter of the charging cavity) with a reliability p of the order of 98-99 %.

To get more information about the nature of the rocks destruction, according to granulometric data (measurement tables, composition curve), we determined such parameters as the weighted average grain size d_{av}, their median size Q_{50} (i.e. the average size is 50 % of the 0-100 micron fraction) and quartiles Q_{25} and Q_{75}, from which the crushing uniformity coefficient S_0 was established by the ratio $S_0=(Q_{75}/Q_{25})^{0.5}$.

The weighted average particle size was calculated by the formula

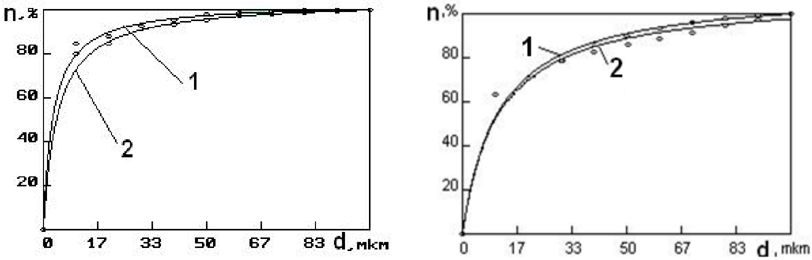
$$\bar{d} = \frac{1}{N}\sum_{i=1}^{n} d_i f_i \tag{8}$$

where f_i is the frequency of the i-th fraction; di is the size of particles in the i-th in the fraction in the range of 0-100 microns with a step of 10 microns; N is the number of particles observed under the microscope.

From the values of quartiles Q_{25} and Q_{75}, the particle sorting coefficient by size, characterizing in our case the uniformity of the medium crushing at the “explosive-rock” contact, was then calculated from relation (7)

$$S_0 = (Q_{75}/Q_{25})^{0.5}. \qquad (9)$$

The cumulative curves of the particle the fine fraction distribution are shown in Fig.5, and the results of calculations of the fine particle-size characteristics fraction for ferruginous quartzites destroyed by impact and explosive loads are given in Table. 5.



a *b*

Fig. 5 - Cumulative curves of fine particle fraction (0-100 microns) granulometric composition of ore (*a*) and magnetite thin-layer (*b*) quartzite (Kryviy Rih) destroyed by dynamic loads of high intensity: 1 - explosion; 2 - impact

Table 5

Granulometric characteristics of fine fractions of quartzite containing iron, destroyed by shock loads

Rocks	Average particle size, d_{av} microns	Quartile measure, microns			The coefficient sorting particles (crushing uniformity) S_0
		Q_{50}	Q_{25}	Q_{75}’	
Ore quartzite Novo-kryvorozhsky MPP	66,71	34,49	12,7	63,75	2,30
Magnetite quartz-ite thin-band Central MPP	59,27	31,35	10,01	58,01	2,41

Analysis of the cumulative curves (Fig. 5, *b*) fraction composition shown within experimental error of their identity, characterizing thereby almost complete similarity, at least at the micro level of fracture quartzite when exposed to explosive and shock loads.

The data in the table indicate that impact loads provide less intensive crushing than explosive loads on the rock. The uniformity of the crushing of fine particles formed mainly in the zone of contact of the sample with the target during impact destruction and at the "explosive-rock" boundary under explosive impact was evaluated by the coefficient of crushing uniformity S_0 (formula (9)). For rocks, S_0 varies in the range of values 1,5-3, and the closer S_0 is to the value 1,5, the more evenly the fractured material is fragmented.

A microscopic study of finely divided fractions (0-100 microns) of ferruginous quartzites formed during shock and explosive effects, it was found that they consist of acute-angle quartz fragments (90-99 %) and 1-10 % ore mineral – magnetite (Fig. 6). This fact can be explained by the high density of structural defects in quartz, observed in transparent petrographical thin sections in the form of numerous subparallel strips consisting of gas bubbles and representing nothing but "healed" microcracks (Fig. 7).

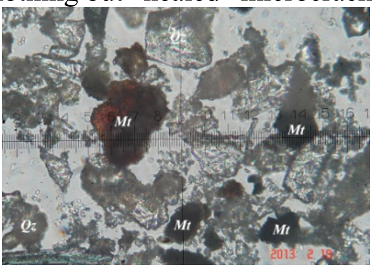


Fig. 6 – Microphotography of the stalled fragments of quartz particles of quartz (*Qz*) and magnetite (*Mt*) in products of explosive destruction of ferruginous quartzite, an increase of 240× (1 distribution of the measuring scale is 5 microns)

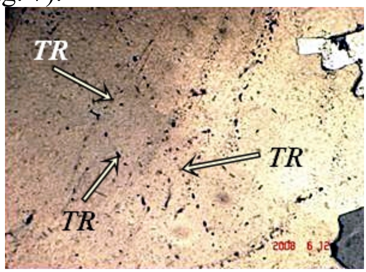


Fig. 7 – Microphotography of quartz in thin-laced magnetite quartzite (Kryviy Rih). Increase 600×. In quartz it is clearly visible "counted" microcracks *TR* (strips of bubbles of gas)

Moreover, the destruction of this rock by explosive loads occurs mainly due to numerous defects in quartz grains, and newly formed cracks in quartzite inherit the planes of the greatest weakening of structural bonds, i.e. natural microcracks and interspecific boundaries. It is also established that the character of the cumulative curve of granulometric composition (see Fig. 5*a*,*b*) depends largely on the density of defects in the structure of quartzites, rather than on the original sizes of mineral grains.

Experimental studies of the destruction of the solid medium by the explosion energy in the contact zone of rocks of varying strength

Blasting operations at mining enterprises largely determine the effectiveness of subsequent technological processes. One of the effective methods of controlling the action of an explosion, and, consequently, crushing of rocks, is the correct choice of the design of the explosive charge and its location in the rock massif in contact zones of various strengths. To study the destructive effect of an explosion of elongated cylindrical explosive charges on models, a method of experimental research has been developed.

The purpose of the experimental studies was a comparative assessment of the results of the destructive effect of an explosion of elongated cylindrical explosive charges in contact zones of various strengths on models. For carrying two series of experimental studies on simulation of explosive destruction of the solid medium with the contact zones of different strength were manufactured model of sand-cement mixture cubic shape with edge 150 and 200 mm. For this, sand-cement dough was placed in a metal form in layers. One layer was formed from a mixture of equal proportions: 1:1- sand +400M cement and another: 1:1- granite chips +400 M cement with the addition of 0,5 liters of water.

When manufacturing models for two series of experiments, cylindrical cavities with a diameter of 8-10 mm were formed with inserts about their axis to a depth of 85 and 105 mm to accommodate elongated cylindrical explosive charges in them (Fig. 8). For one series, one cavity was formed in the center of the model with a rib size of 150 mm, and for the other with a rib size of 200 mm, two charging cavities were formed.

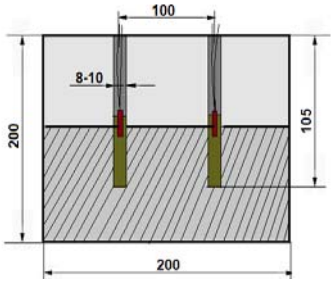


Fig. 8 - Scheme of model with location of charging cavities in it

Before forming the elongated cylindrical explosive charges, the paper cartridges with an outer diameter of 0,95 in diameter of the charging cavity were manufactured in the model. In prepared cylindrical cartridges, a highly explosive substance (PENT) weighing 1,0 g or detonating cord segments was placed for all series of experiments. Specific consumption was 0,3 kg/m^3. The quartz sand of 0,25 mm fraction was used as a catch. For initiating charges, we used segments of the waveguides - the Impulse Non-electric Starting System, the PRIMA ERA, 0,8 m in length, connected to the detonator plug. An explosive device carried out blasting charges. For the evaluation of the physical and mechanical properties of the destructible medium, samples were produced simultaneously with the production of basic models for determining the density ρ, the velocity of longitudinal waves C_p and the strength for uniaxial compression of model material in accordance with current standards [1-5].

Experimental investigations of the destructive effect of the explosion of elongated cylindrical charges on prepared sand-cement models were carried out under the conditions of the polygon conditions at the "Rybalsky" quarry (Dnipro). The models were placed in a thick-walled metal rubberized explosive chamber.

After each explosion, the granulometric composition of the destroyed models was carried out using the sieve analysis method with help of a set of laboratory sieves with apertures of the following sizes 0.25; 0.5; 1.0; 2.0; 3.0; 5,0; 7.0; 10.0; 12.0; 16.0; 20.0; 26.0; 30.0; 40.0; 50.0; 60.0; 70.0; 80.0 mm. When processing the particle size distribution, the total mass of the model destroyed by the explosion, the content of fines, the content of large fractions and the diameter of an average piece were determined.

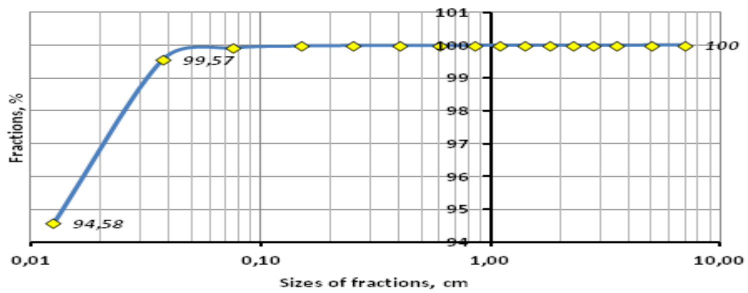
In the case of explosive destruction of relatively large models, as practice has shown, crushing products consist of a large number of fine particles (0-200 μm) and several dozen large fragments (several cm). In this case, the nature of the explosive destruction of models is best assessed in a straight section of the cumulative particle size distribution curve by the value of tg α

$$tg\alpha = P_{(0,0375)} - P_{(0,0125)}/(0{,}0375-0{,}0125), \quad (10)$$

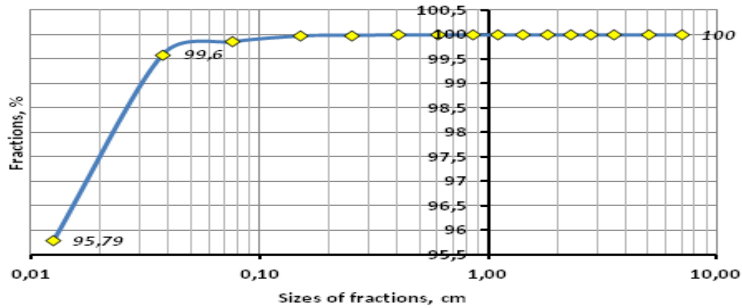
where P (0.0375) is the number of particles in the size range 0-0,0375 cm; P (0,0125) - the number of particles in the size range 0-0,0125 cm; (0,0375-0,0125) – size difference, cm.

The greater the value of tgα, the more evenly the model is fragmented.

In fig. 10 shows the cumulative curves of the particle size distribution of the products of explosive destruction of sand-cement models, destroyed by elongated cylindrical explosive charges, variously oriented with respect to layering.



a



b

Fig. 9 - Cumulative curves of granulometric composition of sand-cement models destroyed by explosion: *a* - solid, *b* - layered

Analysis of the cumulative curves obtained as a result of processing the data on the sieving of fragments of models destroyed by the explosion showed the following. In continuous sand-cement models, a more uniform crushing of the material is observed (tgα=199,6).

Judging by the magnitude of tgα, the presence of stratification in models reduces the uniformity of its fragmentation by explosion energy. In this case, in the case of the application of a breaking load parallel to the stratification, the crushing uniformity is slightly higher (tgα=170,4) than in the case when the breaking load is directed perpendicular to the layers (tgα=153,4).

Analysis of the experimental results showed that the explosion of elongated explosive charges arranged parallel to the layering of the model (loading of the medium occurs perpendicular to the layering) increases the yield of large fractions (5-7 cm). With the arrangement of elongated explosive charges at an angle of 90º with respect to the layers in the model (loading of the medium – parallel to the layering), the proportion of coarse fractions decreases on average by 10-15%. At the same time, when the medium is loaded perpendicular to the layering, the diameter of the middle piece decreases by about the same amount (10%).

Thus, a layered medium undoubtedly has a significant effect on the nature of its destruction by an explosion. This circumstance was taken into account when developing a resource-saving method of explosive destruction of ferruginous quartzites of complex structure, for which the presence of stratification is very characteristic.

Development, industrial inspection and evaluation of the breaking method of hard rocks on quarries ore deposits with blasting

Testing of the developed method was carried out under the conditions of the Petrovsky and Gleyevatsky open pits of Central Mining Plant, the Institute of Geotechnical Mechanics of the National Academy of Sciences of Ukraine, together with the concern ‘SouthOre’. According to its geological structure, the deposits of ferruginous quartzites, to which Gleyevatskoe and Petrovskoye belong, are located within the Ukrainian shield and lie in a narrow strip from Krivoi Rog to Kremenchug in the form of an extended folded zone within the so-called Krivoy Rog-Kremenchug synclinorium [9].

Industrial experiments were conducted on the selected block number 1, horizon -60 m, Petrovsky pit, in two stages. The first one using the base passport of blasting operations, and the second according to the developed recommendations. The block was swept over the 6×6 m grid of the location of the well charges with the basic method of blasting without taking into account the anisotropy coefficient. Taking into account the anisotropy coefficient of 1,3 for ferruginous quartzites, the grid was adjusted: the distance between the rows of wells - 5,2 m, in a row - 6,8 m. The height of the ledge was on average 16 m, the depth of the boreholes for the solid charge design was 19 m, the overdrill was 3 m, and for the design of a charge of variable cross section, 17 m, the overdrill was 1,0 m. The boreholes were drilled with a diameter of 0,25 m, the average charge mass in the borehole was 0,750 tons, the specific consumption was 1,4 kg/m^3, the type of emulsion explosive used — Anemix, Ukrainit M or Prima ERA. The charge initiation in the borehole was carried out by an intermittent detonator, equipped with a patronized emulson explosive and a capsule detonator connected to the NONEL waveguide, "Impulse" or "Prima-Era" (Fig. 10, 11).

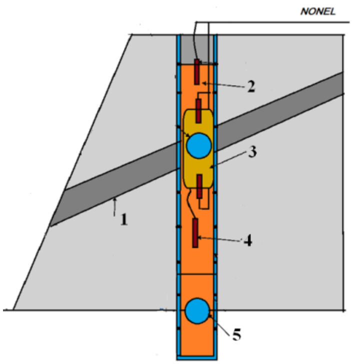


Fig. 10 – Borehole charge design: 1 – contact zone; 2 – explosive; 3 – cumulative charge section; 4 – detonator; 5 – spherical inserts

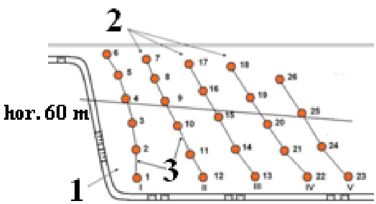


Fig. 11 – Scheme of location and commutation of borehole charges on the block: 1 – block; 2 – boreholes; 3 – commutation of borehole charges;

The design of the recommended borehole charge are shown in Fig. 10. The commutation of the downhole charges on the block is diagonal (Fig. 11).

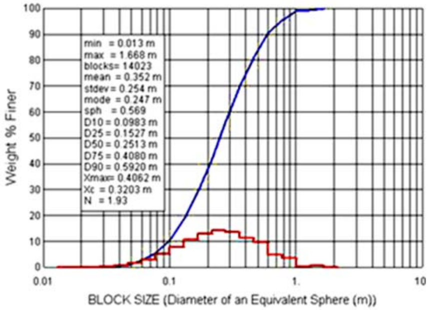


Fig. 12 - Granulometric composition of the destroyed by the explosion of the rock mass, Petrovsky quarry, Central MMP, mountains. - 60 m (mass explosion on September 27, 2007)

At the experimental and control sites, an assessment was made of the quality of crushed rock mass along the diameter of the middle piece. After measuring the granulometric composition by the method of the oblique photo of planimetry [8], the data was processed using standard programs. As a result, it was arranged that the fraction 170-470 mm - about 50 % is the most homogeneous, and 98.9% through a sieve with a cell size of 1200 mm. The average size of the fraction is 352 mm (Fig. 12).

Conclusions

Analysis of the results of industrial experiments showed that the use of modified parameters of drilling and blasting operations using a special combined charge design with a spherical insert of cumulative action reduces the dilution of the mineral in the zone of its contact with waste rock by 50%.

In the course of an industrial inspection of the contradictory method of subterranean destruction in the period 2012-2014, on the granite quarries of the Private Joint-Stock Company “Ukragrovzry-vprom”, according to the developed recommendations, was repelled about 600 thousand m^3 of rock mass. It has been established that while maintaining the quality of blasting rock mining, the use of the proposed technology makes it possible to reduce the specific consumption of explosives and the volume of drilling work on average up to 15% while maintaining the design elevation of the bank bottom. Saving industrial explosives per borehole amounted to 40 kg.

During 2014-2018 years, the implementation of improved parameters of drilling operations, which used to reflect about 1000 thousand m^3 of mining mass with an expected economic effect of 1430 thousand UAH.

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SHORT NUMERICAL ANALYSIS OF THE POSSIBILITIES OF THE PLASTIC ROCKS MADE STRESSES INFLUENCE ON THE PIT SHAFT NEAR THE WASTE DUMP

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Summary. Using the code based on the finite difference method FLAC2D (Itasca C.G.) numerical simulations of rock mass behaviour near the two shafts under the plastic layers influence in rock mass under longwall mining. The model took into account the influence of the load resulting from the waste dumps nearby located. The purpose of the numerical analyses were to confirm the forecasts, carried out with analytical methods. The locations of potential damage of the shafts were determined, which were initially determined by analytical calculations.

Keywords: stability of shafts, numerical modelling, FLAC2D, pressure of plastic layers, environmental protection, post-mining waste, stability of the massif, cohesion and angle of internal friction of rocks and soils

1. Introduction.

Shafts are the basic working that provide useful mineral deposits. During exploitation of coal beds they perform transport and ventilation functions. Their essential role for the technological sequence and safety of the mine causes, that the mining shafts require special attention.

In 1955, Sałustowicz wrote: "the main factors determining the load on the underground mining shaft are: the depth of excavation and the properties of the rocks through which the shaft passes. In compact and strong rocks, this pressure at lower depths is equal to zero; its occurrence starts only from the depth at which the effort of the material has its critical value. […] In the case of a plastic rock mass, the pressure on the shaft lining depends on the depth at which the plastic layer is deposited and on the strength properties of the rocks. In the case of plastic rock layers after the shaft lining is made,

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their displacement can take many years and cause excessive pressure on the shaft lining. This excessive pressure of plastic rock layers can damage the shaft lining."

There are two theses in this citation. The first is that the process of formatting high pressure on the shaft lining in the case of plastic rock layers is a process that evolves over time. The second thesis is that the result of this process may be excessive pressure on the shaft lining. It can be assumed that the notion “excessive” of professor Sałustowicz understood a pressure greater than the predicted pressure determined by analytical methods.

The task of determining the pressure acting on the shaft lining in the rock mass, in which there are plastic rock layers, is more complicated in the case when large-sized earth structures, acting on the rock mass, are created near the shafts.

Using the FLAC2D (Itasca Consulting Group) code based on the Finite Difference Method, numerical simulations of the layered rock mass with plastic rock layers were carried out. The purpose of numerical analyses was to locate or confirm the location of zones of increased stress in the rock mass acting on the shaft lining. Simulations were carried out for two different shafts located a short distance to the post-mining waste dump. The numerical calculations also take into account the influence of the dump load on the shafts linings.

2. Characteristics of shafts linings and dumping ground

2.1. Characteristics of shafts linings

Numerical simulations were carried out for two different shafts lining located at the same distance from the waste dump. These were:

The S1 shaft with a depth of 907.4 m has a diameter of 8,0 m and is located on the main site. It is a mine shaft, two-compartment. The shaft head is made up to a depth of 8.0m, its thickness in the section from the framework to 6,0 m is 2,2 m, while below it the thickness decreases to 1,6 m.

The mining shaft head was made of brick and concrete. Below the shaft head on the section from 8,0 m to 16,5m, the thickness of the shaft lining is 0,9 m, including 0,55m of solid-grade brick 35 and 0.35m of C16/20 concrete. At a depth of 17,0 m, a curb of C20 /25 concrete is made. Below the curb to the depth of 70.0m, the shaft lining is made as a double-layer with a waterproofing foil. The preliminary shaft lining is made of concrete panels of 1.0m × 0.5m × 0.25 m, after which a concrete can with a thickness of 0.5m was made. The final shaft lining in this section was made using the sliding method from the bottom to up of C16 /20 concrete. The thickness of the final shaft lining is 0,5 m. At the depth of 70,0-71,5 m, second can was made. Below the second can, the shaft lining is made of concrete. The thickness of the shaft lining on the section of the Carboniferous rock ranges from 0,35 to 0,60 m.

The S2 shaft has a depth of 849,5 m and a diameter of 7.5m. It is a downcast output-material shaft, two-compartment, equipped with four mine cages, used to go to work, leave the workers and transport materials in rail cars. The head of the shaft is made up to a depth of 15m and the wall thickness of the head made of solid brick 35 with waterproofing foil and concrete C 16/20 is 1.0m. The head is ended with a can. Below the head of the shaft on the section from the depth of 15,2m to 28,0m, the thickness of the shaft lining is about 0,85m, including about 0.50m made of solid-brick 35 and 0,35m of C20/25 concrete. At a depth of 28,0 to 29,5 m, the curb is made of concrete panels and C20/25 concrete.

Below the can to the depth of 70.0m, the shaft lining is made as a double-layer with a waterproofing foil. The prefabrication is made of concrete panels measuring 1,0 m×0,5 m×0,25 m on cement mortar, followed by a 0,15m thick concrete sealing gasket. The thickness of the final shaft lining made of C16/20 concrete is 0,35 m. At the depth of 70,0-71,5 m, another can was made. Below the second can, the shaft lining is made of C16/20 and C20/25 concrete. The thickness of the shaft lining on the section of the Carboniferous rocks is from 0,4 to 0,6 m.

The shafts are in one line and are located at the same distance from the waste dump. Using the maps and geological cross-sections, the same geological structure of the soil and rock layers in the case of both shafts was assumed. The cross-sections (size) of the post-mining waste dump and shafts were different.

Shafts were modeled on the base of cross-sections of shafts S1 and S2.

2.2. Characteristics of the post-mining waste dump

The waste dump is located in the upland, where there are hills crossed by valleys. Terrain is from 255 to 275 m above sea level. The area is generally small slope in the south and is drained by drainage ditches. It is surrounded from the northern side by meadows, forests and a few single-family buildings, western meadows and wasteland with a few single-family buildings. From the southern side, the dumping ground is limited by a street, behind which there is a railway siding and objects of the main mine plant, from the eastern side by meadows and wasteland. There are larger agglomerations of residential housing in this direction.

The post-mining waste dump covers an area of about 65 hectares and is mainly made of gangue from coal enrichment, i.e. claystones, mudstones, loess silts and sandstones. This material is characterized by diversified graining. This allows for a high density ratio $I_s \geq 0.95$ and a filtration coefficient of $10^{-5} \div 10^{-6}$m/s. Also filtering waste is used in the building of the waste dump. These are filtered and pressed sludge from the processes of coal enrichment in the treatment plant. Filtering waste is used to make an insulating layer laid directly in the dump bed. The southern slope of the waste dump is about 200 m away from the two main shafts S1 and S2 (Fig. 2.1).

The height of the waste dump is about 50m. The waste dump is divided into two working levels (Fig. 2.2).

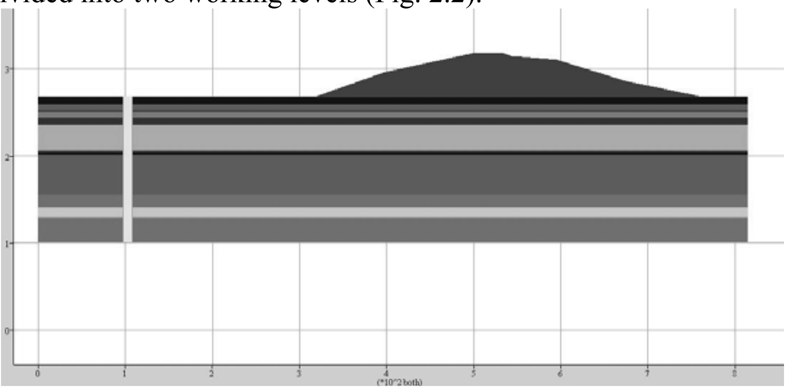


Fig. 2.1. Numerical model of the rock mass and the cross-section of the dumping ground near the S1 shaft

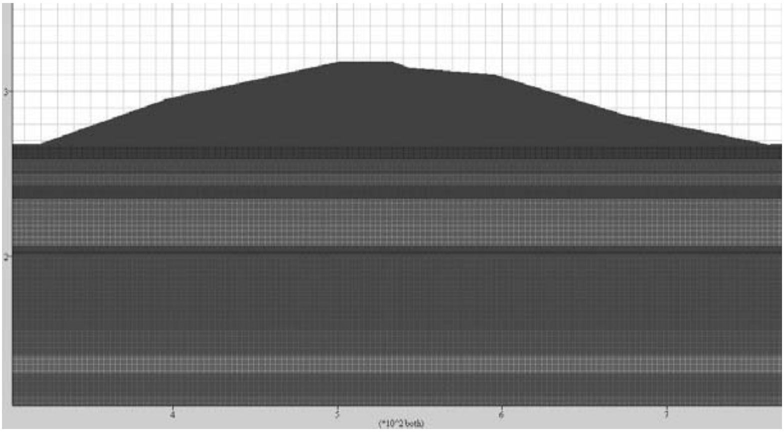


Fig. 2.2. Numerical 2D-model of waste dump with subsurface layers of soils and rocks

The numerical model of the dumping ground, up to a height of about 50m and a length of up to about 400m on the basis of a land development project and technical reclamation using gangue was made . The current map of the surface was also used.

3. Numerical model of rock mass

The simplified rock mass model was built using a code based on the Finite Difference Method FLAC2D v.5,0 (Itasca C.G.). The program has been repeatedly described and is used, also in three-dimensional version (FLAC3D) for solving research and practical tasks in the field of geoengineering, geomechanics and geotechnics (e.g. Cundall and Board, 1988; Cała and Flisiak J., 2003; Itasca CG, 2005; Kwaśniewski and Wang , 1994, Tomiczek, 1995, etc.). Based on data from exploratory boreholes, and information from the geological and surveyor mine department, 16 layers of soil and rocks were described, for which strength and deformational properties were added (Table 1).

Table 3.1

Basic soils and rocks parameters

Depth of the floor H_s [m]	Thickness of stratum m_w [m]	Stratum	Angle of internal friction φ[°]	Cohesion c [MPa]	Elasticity modulus E [Pa]
1	1	Dust	14	0.018	36e6
9	8	Finely grained sand	33	0.001	15e6
11	2	Dusty sand	25	0.005	30e6
16	5	Fine-grained and dusty sand	27	0.003	24e6
18	2	Sandy dust	16	0.020	20e6
25	7	Dusty clay	16	0.028	26e6
33	8	Fine- and coarse-grained sand	29	0.002	48e6
62	29	Gray and dusty clay	12	0.026	30e6
65	3	Dust	18	0.020	38e6
67	2	Dusty sand	25	0.007	32e6
113	46	Dusty silt	12	0.030	27e6
122	9	Weathered mudstone	33	3.5	7e9
124	2	Fine-grained sandstone	39	6.4	12e9
128	4	Grey mud-stone	31	4.0	8e9
139	11	Fine-grained sandstone	40	6.5	12e9
153	14	Grey mud-stone	32	4.2	14e9

The modified Coulomb-Mohr failure criterion was assigned to soils, rocks and shaft lining. The volume density σ was equal to from 1700 kg/m^3 to 2700 kg/m^3.

The failure envelope of this criterion in the plane (σ_1,σ_3) is determined by the equation (Fig. 3.1; Itasca, 2005)

$$f^s = \sigma_1 - \sigma_3 N_\varphi + 2c\sqrt{N_\varphi}$$

The softening function for tensile stresses

$$f^t = \sigma^t - \sigma_3$$

where

φ - angle of internal friction,
c - cohesion,
σ^t - tensile strength, and

$$N_\varphi = \frac{1 + \sin\varphi}{1 - \sin\varphi}$$

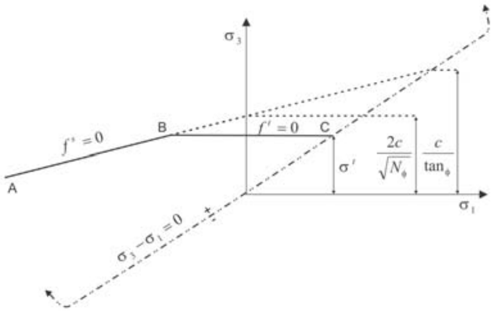


Fig. 3.1. Graphical interpretation of the Coulomb-Mohr failure criterion in the FLAC2D code (Itasca, 2005)

This strength criterion is one of several implemented in the FLAC2D program based on the Finite Difference Method and successfully used to solve tasks in the field of geomechanics and geotechnics.[2]

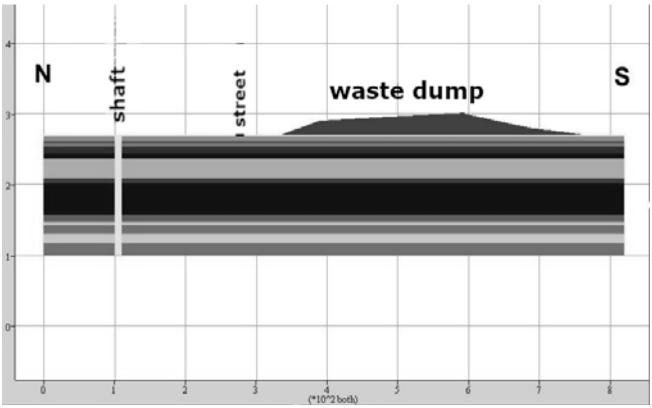


Fig. 3.2. Numerical rock mass model with distinguished strata of soils and rocks (S2 shaft)

[2] The FLAC2D code has already been described many times (e.g. [4-5]), therefore its detailed characterisation have been omitted.

In aim to analyse the influence of the waste dump on the stability of the S1 and S2 shafts, 2D structural and physical rock mass models with a length of 820m (X) and a height of 220m (Y) were built; 50m is the height of the dump, and 170m is the depth from the outset (e.g. Fig. 3.2). The model has been divided into approximately 45000 finite elements in approximately 705000 model grid points.

In a 2D numerical model, displacement boundary conditions were make an assumption. It has been assumed that grid points on the vertical - side walls of the model can displace in the vertical direction (in y direction, on the left and right or, otherwise, north and south). The grid points located on the bottom wall of the model are not able to move neither in the horizontal direction x nor in the vertical direction y. The remaining grid points were able to displace freely in any direction. The top wall of the model was unloaded.

By assigning the primary stress state in the model, it was assumed that the vertical stress in the rock mass was resulted from mass forces and is determined by the bulk density of soils and rocks in the modeled rock mass.

4. Results of numerical simulation

In aim to present the results of the numerical simulations, selected maps of stress distributions were presented in the region of one of the shafts. The maps of shaft S1 were only presented, because the stress and displacement distribution of soils and rocks in the vicinity of both shafts was similar and determined by the location of layers.

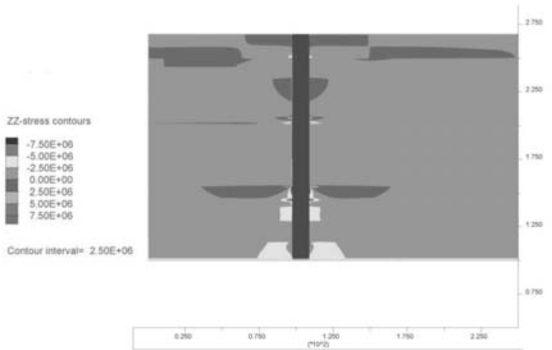


Fig. 4.1. Distribution of vertical stresses σ_z in the vicinity of the shaft

Values of vertical stresses σ_z are less than 5.0MPa (absolute values) and are compressive stresses (light grey colour, Fig. 4.1). These are (of course) the values of stresses expected at depths up to 200 m. However, in the vicinity of plastic layers, there are also tensile stress zones of small values, reaching about 3,0 MPa (dark grey) and compression values of slightly higher values than those characteristic for the depth of rock run, up to about 6,0 MPa (light colour). All compressive or tensile stresses concentration zones are located in the vicinity of plastic layers.

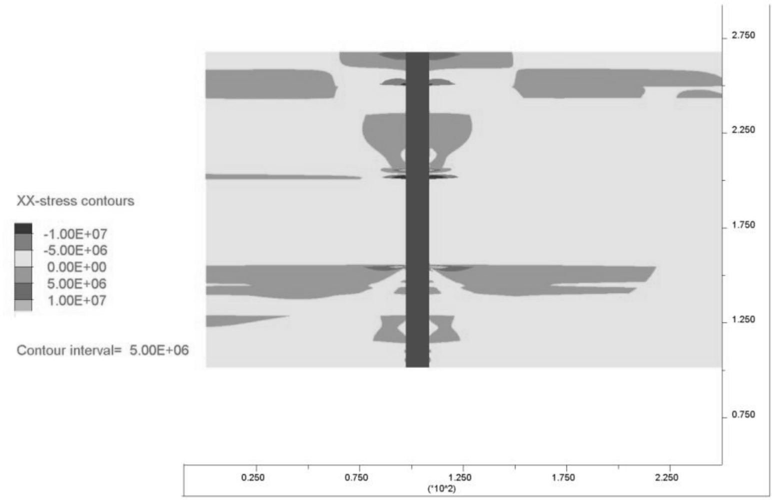
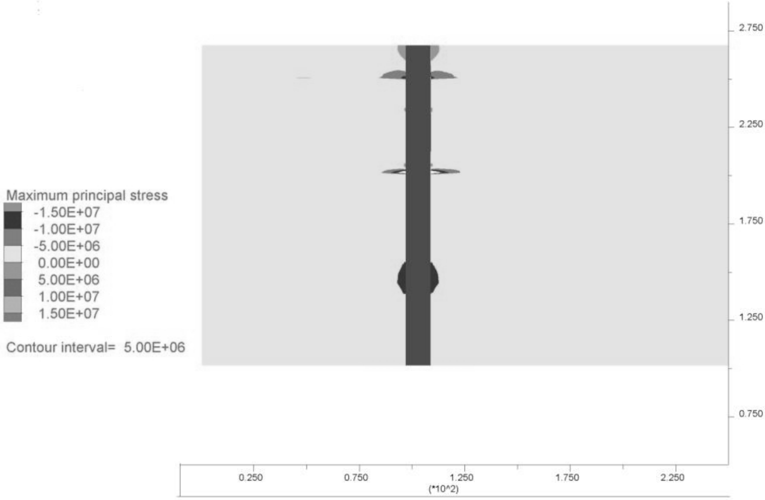


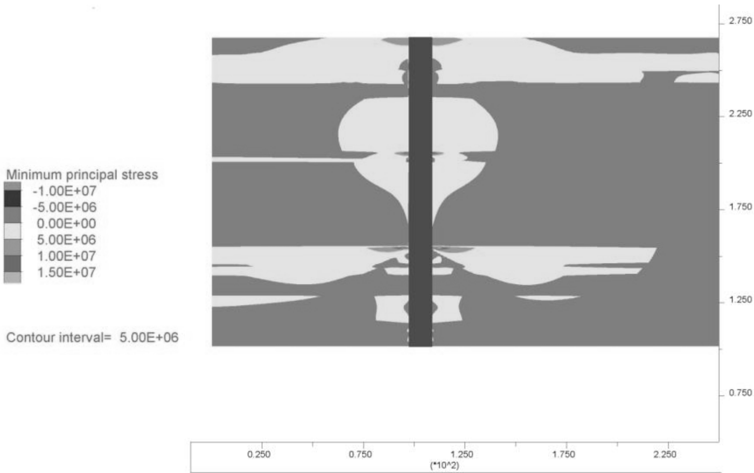
Fig. 4.2. Distribution of horizontal stresses σ_x in the vicinity of the shaft

Similarly, in the case of horizontal stresses σ_x, the concentration zones of compressive or tensile stresses are located in the vicinity of plastic layers (Fig. 4.2). Compressive stresses, generally smaller than 5.0MPa (light color, absolute values) locally reach as much as 10.0MPa (dark color) and tensile stresses also usually lower than 5.0MPa (light gray color).

The distribution of the principal stresses σ_1 and σ_3 is a confirmation of the location of compressive and tensile stresses concentration zones (Fig. 4.3). And if the absolute values of stress do not exceed 5,0 MPa, then at the interface between the plastic layers and the shaft lining, there located zones in which the values increase to about 10,0 MPa. This phenomenon is particularly clearly seen in Figure 4,4, where the areas of the maximum concentration of the principal stresses are marked.



a



b

Fig. 4.3. Distributions of the maximum σ_1 *a* and minimum σ_3 principal stresses *b*

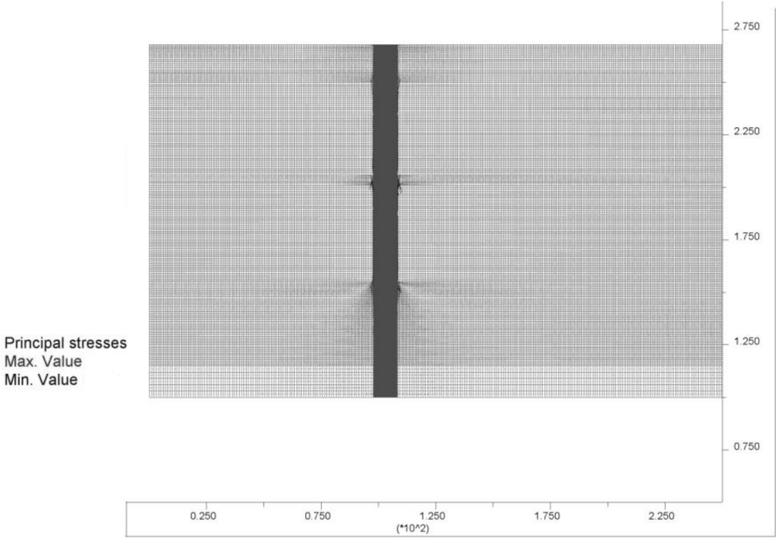


Fig. 4.4. The concentration zones of maximum and minimum principal stresses σ_1 and σ_3.

Numerical simulations carried out on 2D rock mass and waste dump models in the region of S1 (and S2) shaft clearly show that the zones of stress concentration in the vicinity of the shaft tube are located in three different places:

- the first zone is located at a depth of 0 to -30 m from the outset. Its formation is probably related to the pressure of weakly concise subsurface layers of soils,

- in the depth from about -50 to -70 m from the outset there is a second zone. Its origin is related to sand layers and dusty sand located at this depth,

- a third zone is formatted at a depth of about 115 m. According to geological profiles, at this depth, there is a layer of dusty silt lying on the layers of mudstone and sandstone.

5. Summary

Using the program based on the Finite Difference Method FLAC2D (Itasca C.G.), numerical simulations of rock mass behaviour in the vicinity of two S1 and S2 shafts located in a layered rock mass with soft soils and rock were carried out.

The model also takes into account the influence of the load resulting from the waste dump located within a distance of about 200 m.

The purpose of numerical analyses were to confirm and verify predictions, made by analytical methods, of potential damage of shaft lining.

The analysis of fields and stress distributions confirmed that zones of stress concentration near the shaft tube are located in three different zones, in the vicinity of the locations of plastic layers, or located on solid-soft rock contact zones.

The simulations carried out have mainly a qualitative nature. However, they are a supplement to the analytical calculations indicating the shaft lining areas requiring special attention of the technical inspectors.

Subsequent technical inspections confirmed the location of potential damage sites to the shaft pipe. Concrete spalling and cracking were found.

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CAUSES AND MECHANISM OF ROCK BREAKING

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Objective is to determine reasons and development of deformation signs of rock breaking.

Methodology – The paper represents laboratory experimental data concerning rock deformation and breaking in volumetric stress state in terms of the plant of unequal-component volumetric uniaxial compression designed by DonPhTI of the NAS of Ukraine. Conditions of the development of resonance phenomena, while rock deforming and breaking, have been analyzed.

Results – It has been demonstrated that rock is a classic auxetics in which elastic parameters change their value and sign in the process of mechanical loading. Characteristic deformations, in terms of which elastic characteristics change their value, are quantified and constant for all the materials. It has been shown that deformation increment is with alternating signs that highlights wave nature of the deformation. During deformation, four characteristic stages have been singled out that's is peculiar for all the rocks irrespective of the type of stress state and value of extensive pressure.Sign of breaking is represented by resonant increase in the amplitude of increment of linear and shearing deformations at stage three due to double vortex-wave resonance in terms of velocity, structural dimensions, and frequency. While comparing as for the values of shearing modulus and volumetric compression (velocity of proper oscillation of rock particles and sound velocity within the medium), shock wave and dynamic breaking take place. If there is no velocity resonance, then "slow" flow is observed.

Scientific novelty – For the first time, it has been demonstrated that elastic characteristics of rocks are not the material constants but they are structurally sensitive characteristics of deformation resistance with changes in their value and sign while mechanical loading; shearing deformations are of rotational nature. Process of rock deformation includes four characteristic stages. From the viewpoint of breaking, stage three is the most interesting one – resonant increase in the increment of maximum, minimum, and shearing deformation. When modulus of volumetric compression is equal to a shearing modulus, anti-resonance and shock wave take place. If there is no shock wave, then the material just flows.

Practical implications – resonant increase in the amplitude of increment of maximum, minimum, and shearing deformation is the criterion of breaking in general and the criterion of dynamic breaking in particular; in practice, it can be used as a sign or prognostic criterion of breaking.
Keywords: increment of deformation, moduli of elasticity, spiral-vortex wave, anti-resonance, breaking.

Introduction

This was determined by history that the problem to predict material breaking in general and rock breaking in particular was solved by developing so-called theories of strength. Nonavailability of generally acknowledged reliable criteria and signs of breaking shows that the method has appeared to be dead end since the problem is still topical. Traditional methods to interpret experimental data obtained while testing rocks for their strength in the form of "stress-strain" diagrams have run their course. Thus, main objective of the study is to search for the reasons and signs of rock breaking under conditions of volumetric unequal-component compression taking into consideration the fact that energy exchange between the interaction objects is of wave nature and all the structural components of the planet (layers and blocks of different scale levels) are resonators.

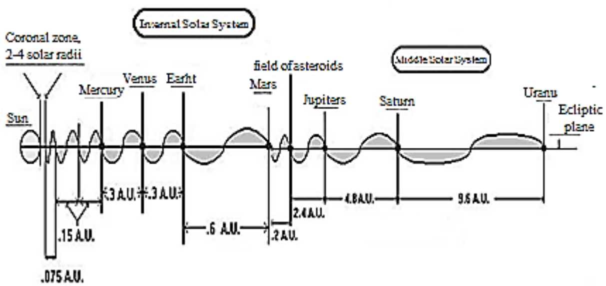
Main idea of the research is to analyze changes in the amplitude of increment of deformations and values of elastic parameters in the process of mechanical loading. Thus, scientific and research tasks included laboratory experiments which involved rock testing under conditions of unequal-component volumetric compression along with the recording ofbothloading incrementin terms of three axes and amplitude of increment of linear, shearing, and volumetric deformations. Since ratio of stresses increment to deformations increment is characterized by physical idea of elastic moduli, next task is to analyze their changes while loading. Apart from laboratory experiments and nonconventional representation of their results, the task is to analyze modern ideas concerning resonance phenomena in rock mass as the authors are sure that resonance is the basic reason for rock breaking. Consequently, logics of the research material representation is in the fact that, at the beginning, short overview of modern ideas concerning resonance phenomena is given followed by the representation of the obtained experimental material which proves the resonance reason of breaking.

Analysis of recent studies and publications.

According to recent theoretical and experimental data, energy exchange between the interaction objects is of wave nature [1-5]. All the Earth's structural components (rock layers and blocks of various levels) are resonators which exciting wave sources are represented by complex Universe source of wave energy with T=160 min period (f=0,104 MHz frequency); that source consists of the reactive component entering the Earth orthogonally to its surface from all the sides and active spiral-vortex one. Reactive component stipulates changes in volume and results in pulsation and spheroidal vibrations of the Earth as well as all its structural elements. Spiral-vortex component results in the planet rotation and toroidal vibrations (being transversal to the reactive component)

Spiral-vortex wave generated in a resonator may be polarized linearly or around a circle in terms of meeting certain amplitude and phase conditions depending upon the conditions of wave field excitation, available boundaries, and their characteristics. Polarization direction may be changed for the opposite one due to characteristic spatial dimension of a wave field. Linear polarization is observed within the areas of that change. On those grounds, ecliptic plane may be considered to be a huge polarization plane where planets of the Solar System move along elliptic orbits (Fig.1,2). Orbits of external plants are divisible by the diameter of the Sun and wave length.

a



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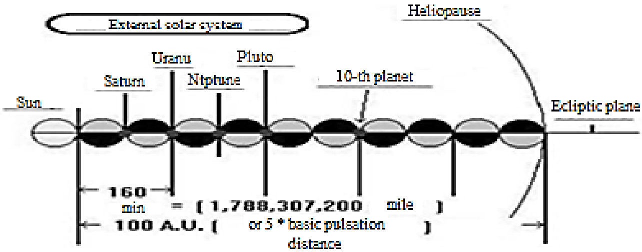


Fig. 1 –Stoneking resonance effecting the arrangement of planets:
a –internal and middle Solar System, only expansion wave is shown (planets are arranged in nodes); *b* – external Solar System:
-compression wave; - expansion wave

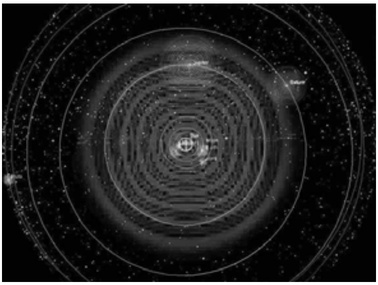


Fig. 2 –Model of hyperspatially resonating Solar System according to David Wilcock

If we agree with resonant structure of the Solar System with planets-resonators, then resonant period of their vibrations may be determined by equating centrifugal force of rotational (vortex) motion to gravitation force

$$T_n = 2\pi\sqrt{R^2/\gamma M},$$

where R is radius of a planet; T_n is period of its vibration as zero source of wave processes; M is mass of a planet; γ is gravitational constant.

In terms of the Solar System resonance, length of resonating chain is

$$L_{cn}=n\lambda/4,$$

where λ=46,32 mln km is length of the wave propagating within the polarization plane (ecliptic plane); L_{cn} is distance of a planet from the Sun, in case of the Earth L_{cn}=149,5-150,15 mln km; n=1,3,5,7 etc. are odd harmonics (resonance within transversal waves takes place only in terms of odd harmonics). In terms of the Earth, n=13.

We will obtain T_n=84,38 min for the Earth as a solid sphere.

Velocity of transversal wave propagation in the Earth interior will be, km/s

$$V_s = D/T_n = 12700 / 84{,}38 \cdot 60 = 2{,}51$$

In case of pulsation of the planet surface as a wave source of zero order

$$R^2/MT_n^2 = (const)_n,$$

where R is radius of a planet; M is its mass; T_n=160 min is pulsation period of the planet surface which is constant for all the objects of the Solar System.

For planet rotation

$$R^3/MT_{вр}^2 = (const)_{вр},$$

where $T_{вр}$ is period of the planet rotation.

Rotation periods of planets as a source of zero order vibrations and periods of their rotation around axes are interconnected

$$T_{вр}=nT_n,$$

where n is number connecting those periods (harmonic order at which vibrations of pulsation and rotation resonate).

For the Earth, n=9, $T_{вр}$=24 hours.

Universal minute constant is proposed for all the discrete systems of macro- and microlevels

$$K = \frac{4\pi^2 R^3}{MT_n^2} = \frac{3\pi}{\rho T^2} = \gamma,$$

where ρ is density of a planet-resonator; in case of the Earth, average density is ρ=5520 kg/m^3; γ is gravity constant, $\gamma=6.67\cdot10^{-11}$ m^3/kg·s^2;

In terms of the Earth as a gravity resonator in stable state, we can set down

$$3\pi / \gamma = \rho T_3^2 = const$$

In this case, T_3 is period of main proper vibration of the Earth as a gravitational resonator.

Hence, T_3=5058,1, s=84,3 min.

Velocity of pulsations propagation in the Earth interior will be, km/s

$$V_p = 2D/T_2 = 2 \div 127000 / 84{,}3 \cdot 60 = 5{,}02$$

As we can see, periods of pulsation waves and toroidal vibrations coincide showing their synchronism and phase balance. Generally, they are propagated together.

Diameter of the earth's sphere as a space object with the atmosphere and all other mantles (Fig.4) may be determined with the help of zero pulsation period being D_0=24096 km.

Proper vibrations of the Earth are gravitational; they are bearing ones, as if wrapping all the types of interaction of structural elements of the Earth, modeling infinite amount of sequences of those types of wave energy which are accumulated in the Earth's substance - from nuclear to gravitational.

According to [6,7], spiral-vortex field, coming from the Sun and interacting with the Earth's substance, is modeled by means of low frequencies of proper gravitational vibrations (Fig.3) to form spherical or toroidal solitons; under the effect of background spiral-wave field, it moves in those forms with alternating velocities along the alternating directions (Fig.4).

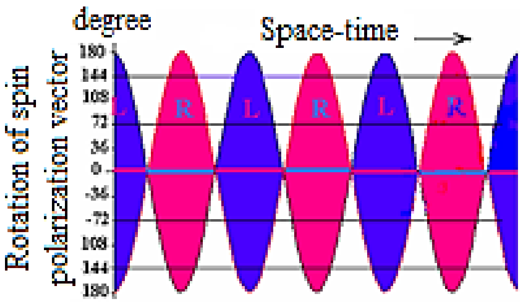


Fig.3 - Modulation constituent of a vortex field in terms of its 100% proper low-frequency modulation

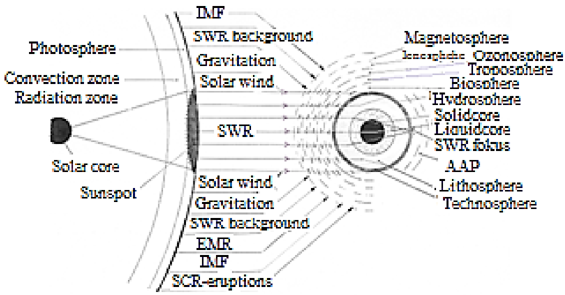


Fig.4 - Scheme of the formation of directed flow of spiral-vortex radiation (SVR) from a big sun spot

In the Earth interior, wave interaction with any sphere resonator is also divided into two motions: active vortex motion and reactive pulsing one. A sphere or block resonator is considered as a secondary radiator (radiating transversal proper vibrations). General energy of a source-resonator depends upon vibration amplitudes and phases being the function of excitation frequency and parameters of the resonator itself. Source-resonator system is the system with one degree of freedom which has one resonant and one antiresonant frequency.

According to [8], resonance is observed not only within reactive constituent which may be interpreted as a longitudinal wave; it is half-wave occurring within all the harmonics and characterized by complete wave transmission from the source without reflection and

Anti-resonance is observed only within the active (transversal) constituent; it is of wave nature, occurs only in terms of odd harmonics and characterized by partial reflection from the boundary line and transformation of the rest of energy into proper transversal vibrations of a layer-resonator propagating along the layer boundaries not going beyond them. A phenomenon of anti-resonance was named as the effect of acoustic resonant absorption.

Thus, longitudinal and transversal constituents of a wave may be observed separately only in terms of resonance (when transversal constituent is equal to zero) and anti-resonance (when there is no longitudinal constituent).

In terms of resonance, primary wave field is strengthened, thoughthere is rather quick attenuation (exponentially). In terms of anti-resonance, primary field decays, though there is a considerable growth in the amplitude of proper spiral-vortex vibrations.

Resonance frequency is connected with characteristic dimension of the object-resonator

$$f_{mx} = \frac{V_p}{2\ell} k,$$

where V_p is velocity of longitudinal constituent of a volumetric wave;

ℓ is characteristic dimension of a layer, block (thickness in case of a layer);

k is harmonic order.

Anti-resonance frequency is determined as follows

$$f_0 = (V_s / \ell) k,$$

where V_s is velocity of transversal constituent of a volumetric wave.

Series of resonance and series of anti-resonance frequencies arise for systems with a set of degrees of freedom.

If resonator is located far from the source, there will be no changes in the source radiation. However, in this case, wave field of the resonator itself may be not only strengthen but weakened as well. Field strengthening results in the development of the repulsion force. On the contrary, if there is certain field decay near the resonator, then the forces of resonator attraction by a source arise. As for a compensating action of the resonator, amplitude of its vibrations is close to the amplitude of the source vibration; in terms of the phase, they are shifted by an angle close to π (180°). Inversely, if resonator strengthens the source radiation, then the amplitude of resonator vibrations increases sharply. In this case, we speak about strengthening action of a resonator.

In the context of simultaneous action of active and passive (resonator) radiator, for areas of strengthening and weakening which alternate, value of total potential is as follows [1,2]

$$\varphi = \varphi_1 \left[1 + \frac{\rho \omega S^2}{iZ(\omega) 4\pi d} \right],$$

where φ_1 is potential developed by one source; S is area of the resonator surface; $Z(\omega)$ is mechanical impedance (acoustic resistance) of a resonator.

It should be mentioned that mechanical impedance is of complex nature (it has both active R and reactive X constituents that depend upon the operating mode of a resonator)

$$Z(\omega)=R+iX.$$

Alternating areas of strengthening and weakening for a "source-resonator" system with a set of degrees of freedom are determined basing upon following condition

$if\left[1+\dfrac{\rho\omega S^2}{iZ(\omega)4\pi d}\right]>1$ - system operates in a strengthening mode;

$if\left[1+\dfrac{\rho\omega S^2}{iZ(\omega)4\pi d}\right]<1$ if< 1 - weakening area is being implemented;

$if\left[1+\dfrac{\rho\omega S^2}{iZ(\omega)4\pi d}\right]=1$ if<1 - there is the alternation of strengthening and weakening areas transition.

Impedance (resistance) of a resonator at which those transitions are possible are called boundary ones $Z_{гр}(\omega)$. The latter equation is the basis for the value of boundary impedances

$$Z_{гр1}=\pm\infty \text{and} Z_{гр2}=1/2\ Z_{\text{ap}}.$$

Here, Z_{ap} is the input resistance (impedance) of a resonator in terms of which it is set for anti-resonance.

$$Z_{ap}=\frac{i\rho CS(kr_0)^2}{4\pi d}=\frac{i\sqrt{\rho G}S(kr_0)^2}{4\pi d}, ,$$

where r_{o} is effective (characteristic) resonator dimension;

ρ is medium density;

ω is resonance frequency;

D is distance between a source and resonator;

K is wave number;

C is phase velocity of the wave in the medium, $C=\sqrt{K/\rho}$;

G is shearing modulus of a resonator.

A system operates with strengthening when modulus of volumetric compression goes through infinite distance K=∞ and change its sign or decreases; a system operates with weakening when shearing modulus experiences decrease by 4 times and wave resistance will be equal to a half of anti-resonance one. When moduli of shearing and volumetric compression are equal, a shock wave arises as well as considerable growth in the amplitude of transversal (vortex) vibrations and destruction in dynamic mode.

Papers [9,10] demonstrate that nowadays there are four types of resonance:

Velocity and frequency resonance - excitation of one set of frequencies from a possible spectrum by a vibration system;

Structural and wave velocity and dimension resonance - excitation of proper structural elements by a vibration system when characteristic dimensions of a structural element are close or divisible by the system dimension;

Double vortex-wave resonance in terms of velocity, dimensions, frequency, and parameters of vortex structures resulting in sharp changes in the interaction between a structural element and the medium with generation of new dispergating waves and vortex structures;

Structural resonance - resonance interaction through a medium of several asymmetric systems with boundary lines or vortex structures.

All geo- and gas-dynamic phenomena in rock mass are connected with resonance types three and four. Consequently, it is expedient to carry out their experimental studies and field observations.

Statement of the problem.

The problem is to study changes in the increments of rock deformations and elastic parameters to determine resonance phenomena in the process of its deformation and breaking.

Research methods.

To achieve the objective, laboratory studies were involved to analyze rock behaviour under conditions of unequal-component volumetric compression in the unequal-component triaxial compression plant designed by DonPhTI of the NAS of Ukraine; in addition, a method to represent the obtained experimental results has been changed.

Statement of the material and the results.

To a large extent, information content of the experimental studies is determined by the type of experiment equipment, number of parameters recorded during the experiment, methodology of the obtained data processing, and a technique of data representation and analysis.

Experiments dealing with rock testing in terms of unequal-component compression ($\sigma_1 > \sigma_2 > \sigma_3$) have been carried out on the plant of unequal-component triaxial compression designed by DonPhTI of the NAS of Ukraine (Fig. 5).

Fig. 5 - Unequal-component triaxial compression plant designed by DonPhTI of the NAS of Ukraine (UCTCP)

Irrespective of the fact that the mentioned plant is a unique one, the way of the research results representation over the years of its operation as well as the results interpretation have not allowed revealing completely the breaking process physics yet.

Traditionally, studies of the rock behaviour in terms of unequal-component compression are represented in the form of diagrams: "spherical stress tensor - relative volumetric deformation", "octahedral stress tangential - relative octahedral tangential (shearing) deformation", "maximum stress - maximum deformation" etc. Those diagrams help determine elastic parameters (modulus of shearing compression, shearing modulus, coefficient of transversal deformation), characteristic deformations, in terms of which elastic parameters experience jump-like changes, regularities of rock strength changes due to the type of stress-strain state etc. However, neither absolute loading value nor relative deformation value can be a breaking criterion in general case even for one and the same type of stress state or one and the same spherical stress tensor. Strength limit and boundary relative deformation depend upon numerous factors which effect is sometimes not even considered.

For instance, Fig. 6 and 7 show following diagrams: “spherical stress tensor - relative volumetric deformation” and “octahedral stress tangential - relative octahedral tangential (shearing) deformation”.

a *b* *c*

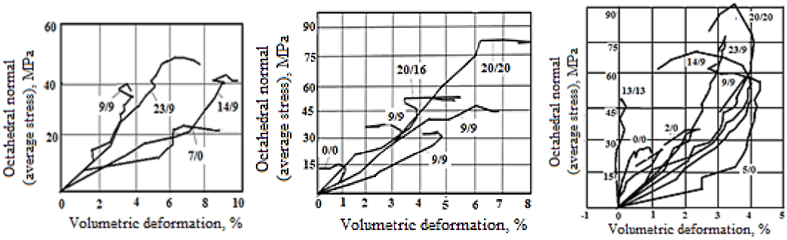


Fig.6 – “Octahedral normal stress (spherical tensor) – volumetric deformation” dependence for volumetric stress state (values of intermediate and minimal stress are shown after the slash): *a* - for coal; *b* - for argillites; *c* - for sandstones

The diagrams reflect clearly the changes of relative deformations in the process of loading. They demonstrate properly the points of inflexions with jump-like changes in elasticity moduli; that indicates the available electronic and structural-phase transitions in minerals making up the rocks.

Fig.6 shows well the structural-phase transitions of the first and third order. First order transitions occur with the decrease in volume in terms of constant pressure while third order transitions (so-called critical transitions) occur with the increase in volume in terms of growing pressure. Jump-like changes of uniform compression modulus in Fig.6 and shearing modulus in Fig.7 prove structural-phase transitions of the first and third order.

a *b* *c*

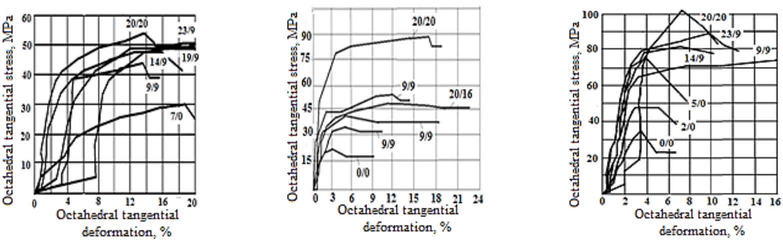


Fig. 7 - **Fig.6** - “Octahedral tangential stress – octahedral shearing deformation” dependence for volumetric stress state (values of intermediate and minimal stress are shown after the slash): *a* - for coal; *b* - for argillites; *c* - for sandstones

“Average stress - average deformation” diagrams (Fig.6) show clearly that in the process of loading, modulus of volumetric compression growth consistently; on the threshold of dilatancy (compression limit), it passes through the bifurcation point (+∞) and changes its sign (volume decrease is replaced by volume increase).

That point is called structural-phase transition of the third order or critical transition.

In certain cases, we can observe rock strengthening (so-called cold-hardening) when numerous fractures formed due to the volume increase in terms of structural-phase transition prevent each other from propagation.

On the contrary, shearing modulus (Fig.7) at the beginning of compression has rather high values (including negative ones) while passing through the bifurcation point (-∞), changes its sign for positive one, and decreases continuously down to the moment of antiresonance.

Fig.8 represents “maximum stress - maximum deformation” diagrams for sandstones and coal.

a *b*

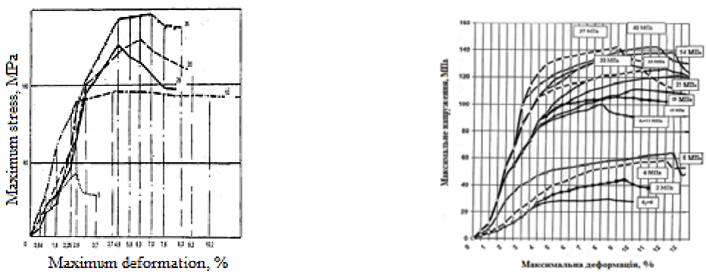


Fig. 8 - “Maximum stress – maximum deformation” diagrams for sandstones, if $\sigma_1>\sigma_2=\sigma_3$ (*a*) and coal in terms of different values of minimum σ_3 and intermediate σ_2 compression stresses (*b*)

Diagrams in Fig.8 help determine rock strength, moduli of shearing and decay, characteristic maximum linear deformations in terms of which the moduli experience jump-like changes (Fig.8*a*). Traditionally, diagrams are drawn to be smoothed but inflexion points (especially in rocks) are so clear that the smoothing results in the loss of physical sense. It is easy to note that both coal and sandstones demonstrate six-seven characteristic points of diagram inflexion points.

Characteristic deformations at which elasticity moduli experience jump-like changes complement a uniform series 0.64; 1.0; 1.5; 1.8; 2.25; 2.6; 3.0; 3.7; 4.6; 5.8; 6.3; 7.0; 8.3; 9.7; 11.2; 12.7; 13.7; 14.5; 16.3% which comply with the universal manifestation of structural harmony, golden section law, and may be represented as follows

$$\gamma_N = \left(\frac{1}{\Phi}\right)^{\ell/2+1/4}; \quad s_N = \Phi\left(\frac{1}{\Phi}\right)^{\ell/2+1/4}; \quad \varepsilon_N = \Phi^2\left(\frac{1}{\Phi}\right)^{\ell/2+1/4};$$

where γ_N, s_N, ε_N are shearing, vortex (rotational), and normal (linear) deformations respectively;

Φ=1б618 is Phidias number (golden number);

ℓ=1,2,3… is any whole quantum number.

In this context, conventional approach to the interpretation of the obtained experimental data has run its course; thus, first of all, it is proposed to study increment of all the three deformations while loading, their changes and effect upon elastic parameters of rocks since elasticity moduli in their essence are the ratio of loads increment to the deformations increment.

Data on deformations while testing prismatic rock samples were recorded with the help of mechanical gauges of clock type with the accuracy of 10^{-6} m. Strain gauges are not capable of recording breaking deformations up to 10-20%.

Maximum external load being formed on the sample is about 400 MPa.

Loading deformation degree for corresponding and synchronous record was 2 MPa. To obtain reliable results, each experiment involved at least three tests.

Fig. 9,10 demonstrate data concerning changes in the increment of volumetric, shearing, and linear relative deformations in the process of coal sample loading in terms of different types of stress state. Similar dependences are shown in Fig. 11,12 for sandstone.

The data are given twice: dependence of the deformation increment upon absolute value of maximum compression stress and upon relative value (current value related to the breaking one). That makes it possible to evaluate clearly similarity of the dependences for all rocks and all types of stress state.

a *b*

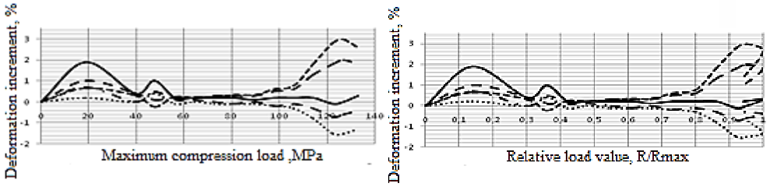


Fig. 9 - Changes in the increment of volumetric, shearing, and linear deformations in the process of coal sample loading, if $\sigma_1 > \sigma_2 = \sigma_3 = 20$ MPa, depending on absolute value of maximum compression stress (*a*) and relative value of maximum load (related to breaking one) (*b*)

increment volum deformation, increment shift deformation,

ε_1, ε_2, ε_3

a *b*

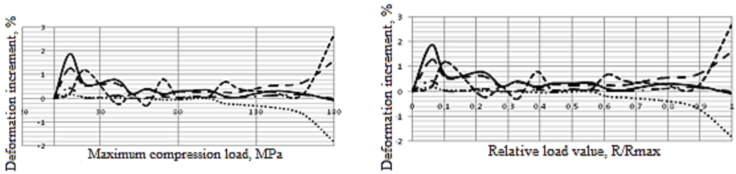


Fig. 10 - Changes in the increment of volumetric, shearing, and linear deformations in the process of coal sample loading, if

increment volum deformation, increment shift deformation,

ε_1, ε_2, ε_3

a *b*

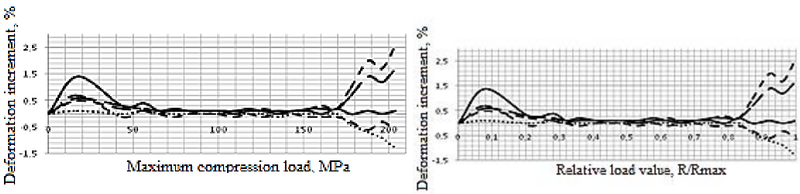


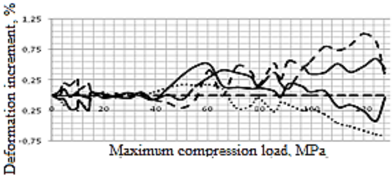
Fig.11 - Changes in the increment of volumetric, shearing, and linear deformations in the process of coal sample loading, if

($\sigma_{\perp}(2)$=23 MPa; ($\sigma_{\perp}(3)$=9 MPa

increment volum deformation, increment shift deformation,

ε_1, ε_2, ε_3

a *b*



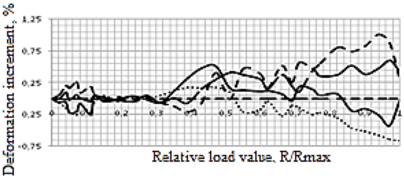


Fig.12 - Changes in the increment of volumetric, shearing, and linear deformations in the process of coal sample loading, if $\sigma_1 > \sigma_2 = \sigma_2 =$ 13 MPa
——— increment volum deformation, -------- increment shift deformation,
— — — — ε_1, —— ▪ —— ▪ ε_2, ''''''''' ε_3

While analyzing the data, it should be stated that the process of rock deformation in terms of mechanical loading under conditions of unequal-component triaxial compression may be divided into four stages:

1 - intensive decrease in the volume at the initial stage of deformation practically without changes in the geometry (in terms of loading up to 0.3-0.4 of breaking ones);

2 - changes in the volumetric deformation increment to zero (reaching at threshold of dilatancy or compression limit), fluctuations in the increment of the rest of deformation with the amplitude being not more than 0.5% (in terms of loads up to 0,6-0,75 of breaking ones);

3 - sharp increase in the increment of shearing, minimum, and maximum linear deformations up to several percents (in terms of loads up to 0,85-0,99 of breaking ones), increase in rock volume;

4 - inversion of the increment of all the deformation types and dynamic rock breaking with sharp load drop (it is observed right before breaking in terms of 0.99 value of breaking load). If there is no load drop, then elastic vibrating shearing process (flow) is observed.

Stage three, when amplitude of the increment of linear and shearing deformations increases by several times (from tenths of a per cent up to several per cents), indicates the availability of resonance phenomena. Transition of breaking into dynamic one or into flow is connected with the resonance type. Consider changes in elastic parameters while loading to define correspondence of the resonance type to theoretical models.

Fig.13 and 14 represent data concerning changes in modulus of uniform compression, shearing modulus, and coefficient of transversal deformation for the abovementioned samples.

a *b*

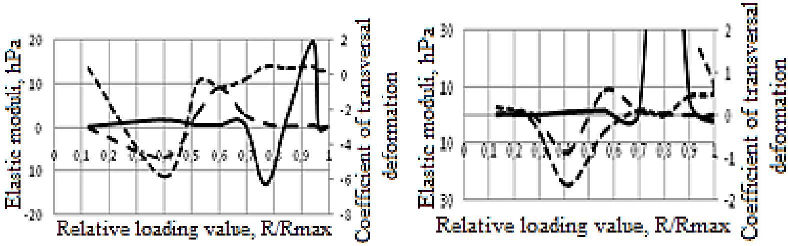


Fig. 13 - Changes in elastic parameters while loading in terms of coal:
a - if $\sigma_1>\sigma_2>\sigma_3$ ($\sigma_\perp(2)$=23 MPa; ($\sigma_\perp(3)$=9 MPa
b- if ($\sigma_\perp(2)$=30 MPa; ($\sigma_\perp(3)$=20 MPa
——— *K*, — — — *G*, -------- *V*

a *b*

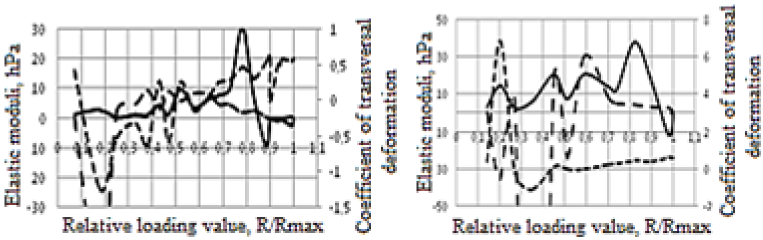


Fig. 14 - Changes in elastic parameters while loading in terms of sandstone: *a* - if *b* - if ($\sigma_\perp(2)$=35 MPa; ($\sigma_\perp(3)$=20 MPa
——— *K*, — — — *G*, -------- *V*

The data show that both coal and sandstone are classic auxetics having negative coefficient of transversal deformation. Modulus of shearing to the compression limit experiences periodical changeswhile passing through the bifurcation point (-∞) in sandstones with the decrease to zero after reaching the compression limit. Modulus of volumetric compression changes periodically to the compression limit with sharp growth at the compression limit often passing through the bifurcation point (+∞); after that point, it decreases sharply taking on negative values.

While loading, the moduli become equal several times (velocity resonance occurs).However, before the compression limit, it happens in terms of antiphase change of moduli; resonance in the monochromator mode (flow under the action of volumetric constituent of the deformation field) takes place. Only after bifurcation of uniform compression modulus, both moduli become equal in terms of co-phased decrease; anti-resonance in the acoustic resonant absorption mode takes place with sharp increase in the amplitude of deformation amplitude (by the value of quality factor). When shearing modulus and volumetric compression modulus are equal (velocity of the substance particles motion is higher than the sound velocity), a shock wave and dynamic breaking are observed. If there is no shock wave, then the breaking is in the form of flow.

Conclusions.

Thus, double resonance in terms of velocity, dimensions, and frequency is the main reason of rock disintegration. First, resonance takes place within the monochromator frequency (modulus of volumetric compression grows infinitely); then, the flow appears. After that, shock wave and dynamic breaking is observed on the anti-resonance frequency when shearing modulus decreases relative to the resonance one by 4 times, and velocity of the constituents motion in spiral-vortex wave is higher than the sound velocity within the medium. Since the difference between the resonance and anti-resonance frequencies is about 15%, possible transition offlow into dynamic breaking takes place with certain time delay.

Resonance increase by the order of amplitude of the increments of maximum and minimum linear deformations is the sign and criterion of rock breaking.

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DETERMINING EXPERIMENTALLY PARAMETERS OF ROCK MASS OSCILLATION EQUATION BY APPLYING LAGRANGE'S THEOREM FOR THE OPEN PIT VELIKI KRIVELJ LOCATION

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Abstract

Application of the energy of explosive represents a very important discipline of contemporary mining technology. The energy of explosive per its form has a destructive character, but today the application of this energy has been improved. In mining practice mass blasting as the manner of exploitation is much more applied. In that way the quantity of blasted mass of rock material increases and the production expenses decrease. However, the use of great quantities of explosive brings about increase of negative blasting effects. By negative blasting effects we understand seismic effect of blasting, the effect of the air blast wave, sound effect, scattering of blasted rock mass, etc. To estimate, to control and plan seismic effects of blasting it is necessary to determine rock mass oscillation equation. One of the most frequently used is the equation of M.A. Sadovskii, which defines the change of rock mass oscillation velocity depending on the distance from blasting point, quantity of explosive, conditions for performing blasting and geological characteristics of the rock mass. In this paper analysis of the method for determining parameters of the rock mass oscillation equation has been done and that according to the equation suggested by the Russian professor M.A. Sadovskii. Experimental researches were performed on copper ore at open pit Veliki Krivelj, Bor district, eastern Serbia. To determine parameters in the Sadovskii equation, besides the usual model – Method of Least Squares one more model has been presented applying Lagrange's theorem. Then, it has been stated that both models can be used to calculate rock mass oscillation velocity.

Key words: working environment, blasting, seismic effect, oscillation velocity, rock mass oscillation equation.

1. Introduction

In mining, explosives are first of all used to extract raw mineral materials from hard rock. Effects of blasting may be divided into two categories, and those are: profitable and not profitable work. Profitable work is manifested in the form of crumbling and crushing rock material in a limited zone around explosive material. Work that is not profitable is a phenomenon known as seismic effect of explosion. Work that is not profitable is connected to elastic/lateral movement, i.e. oscillation of rock mass particles within a large space around the point where the explosion happens. That is the phenomenon manifested and felt as impact, i.e. rock mass vibration. When talking about negative effects of blasting besides seismic effect of blasting we also mean the activity of the air blast wave, sound effects, scattering of blasted rock mass, etc. [1-6]. Due to those reasons greater attention is being paid to study these phenomena, striving to reduce them to allowed limits.

By applying rock mass oscillation equation when blasting, there is possibility to determine in advance the for every blasting rock mass oscillation velocity, i.e. the scope of impact. Thus, blasting, as regards seismic effects, is in control [7].

As a link between rock mass oscillation velocity and basic parameters that influence its value, most often is used the M.A. Sadovskii equation, where the oscillation velocity v is given as

$$v=K\cdot R^{-n} \tag{1}$$

where R represents reduced distance, and parameters K and n are conditioned by rock mass characteristics and blasting conditions. Thus, v is a decreasing and convex function of the changeable R.

2. Rock mass oscillation equation

To establish the correlation between the oscillation velocity and three basic parameters affecting its size: the explosive quantity, properties of rock material and the distance, there have been developed several mathematical models in the world. One of the most frequently used models, i.e. equations, is the equation of Sadovskii defining the equation on velocity alteration of rock mass oscillation depending on the distance, the explosive quantity, and the way of blasting [8]. The equation defined in this way offers the possibility to determine the seismic effect of blasting towards a facility or a settlement, whereby the connection, between the velocity of rock mass oscillation and consequences that can affect facilities, is used [9].

The equation of M.A Sadovskii is given in the form

$$v = K\left(r/\sqrt[3]{Q}\right)^{-n} \tag{2}$$

where there are

v - velocity of rock mass oscillation [cm/s];

K - coefficient conditioned by rock mass characteristics and blasting conditions, where the amount of explosive is given by way of the volume. K is determined by terrain surveying;

n - exponent, conditioned by rock mass properties and mining conditions and determined by field measurements as well;

r - distance from the blast site to the monitoring point [m];

Q - amount of explosive [kg]

R - reduced distance, expressed as $R = r/\sqrt[3]{Q}$.

The Sadovskii equation is determined based on test blasting for the concrete working environment.

2.1. Models of determination of rock mass oscillation equation parameters

There are two parameters K and n in the equation (1) which should be determined for the specific work environment and by particular blasting conditions.

2.2.1. Model 1 - determining parameters by applying the Least Square Methods

The Least Square Method is mainly used to obtain the parameters K and n which represents a common model [10].

2.2.2. Model 2 - determining parameters by applying Lagrange's theorem

In the rock mass velocity oscillation equation (1)

$$v = K \cdot R^{-n} \tag{1}$$

parameter n may be determined by approximations by applying Lagrange's theorem [11].

If we differentiate rock mass oscillation velocity equation (1) per R we will obtain

$$v' = -n \cdot K \cdot R^{-n-1} \quad i.e.\ \ v' = -\frac{n}{R} \cdot K \cdot R^{-n} \tag{3}$$

Having in mind (1), the equation (3) is reduced to

$$v' = -\frac{n \cdot v}{R} \tag{4}$$

From (4) we find

$$n - =-v'\frac{R}{v}\ . \tag{5}$$

If we find derivative v' in a point R_c, then from equation (5) we can determine n. The value of derivative v' may be determined by one of formulas for numerical differentiation. For that purpose, Stirling's formula for the case of equidistant values for the changeable R is often applied. In spite of that it is necessary that step h be a small number, although in the examples of mass blasting in mining is not the case.

Earlier researches have shown that values of parameter n are mostly in the interval from 1 to 3, most often in the interval from 1 to 2.

To determine parameter n we will apply Lagrange's theorem which is worded: if the function $f(x)$ is continuous at $[a,b]$ and differentiable at the interval (a,b), then there is at least one c, $a<c<b$, for which is

$$\frac{f(b-f(a)}{b-a} = f'(c)\ . \tag{6}$$

In our case, since the oscillation velocity is continuous, decreasing and convex function at $[R_1,R_s]$ and differentiable at the interval of (R_1,R_s), for this function Lagrange's theorem is valid

$$\frac{v(R_s) - v(R_1)}{R_s - R_1} = v'(R_c) \quad R_1 < R_c < R_s \tag{7}$$

Having in mind formulae (1) and (3), formula (7) is reduced to

$$\frac{k \cdot R_s^{-n} - k \cdot R_1^{-n}}{R_s - R_1} = -k \cdot n \cdot R_c^{-n-1} \tag{8}$$

i.e. to

$$\frac{R_s^{-n} - R_1^{-n}}{R_s - R_1} = -n \cdot R_c^{-n-1} \quad R_1 < R_c < R_s \tag{9}$$

From equation (9) we get

$$R_c = \left(\frac{n \cdot (R_s - R_1)}{R_1^{-n} - R_s^{-n}} \right)^{\frac{1}{n+1}} \qquad R_1 < R_c < R_s \tag{10}$$

In practice it may be assumed that R_1 is the least reduced distance, and R_s the greatest reduced distance that had been monitored during blasting i.e. measuring.

In formula (7) values $v(R_s)$, $v(R_1)$, R_s and R_1 are taken from the table of experimental data, thus we have

$$\frac{v_s - v_1}{R_s - R_1} = v'(R_c) \tag{11}$$

Thus, we find derivative $v'(R_c)$ at the point R_c, where R_c is given by way of help of formula (10).

In practice, formula (7) is used so that values $v(R_s)$ and $v(R_1)$ are taken from the tables of obtained experimental data for adequate reduced distances R_s and R_1, so that for R_s is taken the greatest reduced distance, and for R_1 the least reduced distance for each monitored table.

To find values of $v'(R_c)$, in accordance with formula (7) we will use formula

$$v'(R_c) = \frac{v_s - v_1}{R_s - R_1} \tag{12}$$

where we have:

v_s - registered rock mass oscillation velocity for the greatest reduced distance R_s,

v_1 - registered rock mass oscillation velocity for the least reduced distance R_1.

In this way we get $v'(R_c)$, and then R_c is determined according to formula (10). In formula (10) for determination of value for R_c appears parameter n. For the initial approximatively value of parameter n we will assume that it is: $n=n_0=1{,}5$.

Now, according to formula (10), for $n=1{,}5$ we have

$$R_c = \left(\frac{1{,}5 \cdot (R_s - R_1)}{R_1^{-1{,}5} - R_s^{-1{,}5}} \right)^{\frac{1}{2{,}5}} \tag{13}$$

where the obtained value R_c is between values R_j and R_{j+1} given in the table of experimental data, and which means: $R_j<R_c<R_{j+1}$.

With thus determined value for R_c we are to determine value $v_c=v(R_c)$, where it is: $v_j<v_c<v_{j+1}$

To find value for $v_c=v(R_c)$ we will use interpolation formula

$$v_c = v_j \frac{v_{j+1}-v_j}{R_{j+1}-R_j}\cdot(R_c-R_j) \tag{14}$$

With thus found value for v_c, in accord with formula (5), we get parameter n as

$$n = -v'\cdot\frac{R_c}{v_c} \tag{15}$$

- it will be marked as n_1, thus according to the formula (7) we have

$$n_1 = -v'\cdot\frac{R_c}{v_c} \tag{16}$$

Now, with thus found value n_1 we will use formula (10) to determine new values for R_c, where instead of n we put n_1, and at the same time we proceed with the previous action for finding new approximate values n_2 of parameter n.

In practice, after several iterations satisfactory value of parameter n can be achieved.

Value of parameter K we get from formula (1) where it is

$$K=v\cdot R^n \tag{17}$$

In equation (17) for n we take value that was obtained as previously shown.

Thus, data for pairs are also used (R_m, v_m)=1,2,...,N, from the table of experimental data, so we take

$$\begin{aligned} &K_1 - v_1\cdot R_1^n \\ &K_2 - v_2\cdot R_2^n \\ &\dots \\ &K_N - v_N\cdot R_N^n \end{aligned} \tag{18}$$

for parameter K is taken its arithmetic mean, i.e.

$$K = \frac{K_1+K_2+\dots+K_N}{N} \tag{19}$$

Thus, we have determined the model for solving rock mass oscillation equation, applying

$$v=K\cdot R^{-n}$$

that is valid for the given environment where the blasting was executed.

3. Defining statistical criteria

For the above-mentioned models 1 and 2, based on experimental data, we have obtained equations, which make possible to determine the oscillation velocities of the rock mass v depending on the reduced distance R. In order to assess the degree of connection between v and R we have used the curved line dependency index ρ [12,13].

The evaluation of the relationship degree of two variables [10] to values of the curved line dependency index ρ is given in the following review:

$0{,}0<\rho<0{,}2$ none or highly poor correlation,

$0{,}2<\rho<0{,}4$ poor correlation,

$0{,}4<\rho<0{,}7$ significant correlation,

$0{,}7<\rho<1{,}0$ strong or highly strong correlation.

As a convenience measure of the obtained functional relationship for the given experimental data, the criterion "3S" was also used [14]. This criterion uses squares of differences between the obtained experimental data and the calculated ones for oscillation velocities of v. If those differences are one after another $\varepsilon_1,\varepsilon_2,...\varepsilon_N$, then it is

$$S=\sqrt{\frac{\varepsilon_1^2+\varepsilon_2^2+...+\varepsilon_N^2}{N}} \tag{20}$$

According to this criterion, for the evaluation of convenience of the obtained functional correlation, the following relations are valid:

- if it is $\left|\varepsilon_{max}\right| \geq 3S$, the obtained functional correlation is rejected as unfavourable,

if it is $\left|\varepsilon_{max}\right| < 3S$, the functional correlation is accepted as a good one.

4.0. Methodology of research

4.1. General characteristics of the work environment in the open pit Veliki Krivelj

In the open pit Veliki Krivelj, Bor district, eastern Serbia, we have performed experimental investigations on copper ore [15]. The basic physical and mechanical properties of the rock mass were determined on samples in the laboratory. By examining the physical and mechanical properties of the work environment, the following values were obtained:

- compressive strength, σ_s=31 [MPa]
- bending strength, σ_s=5,1 [MPa]
- tensile strength, σ_1=3,4 [MPa]
- cohesion, C=7,5 [MPa]
- strength coefficient, f=3
- volume density, $\gamma = 24\,[\text{kn/m}^3]$
- angle of internal friction, φ=52 [°]
- porosity, p=3,5 [%]
- velocity of longitudinal waves, c_p=2,4 [km/s]
- velocity of transverse waves, c_s=1,4 [km/s]

4.2. Method of blasting in the open pit Veliki Krivelj

The measurements of shocks resulting from blasting in the open pit Veliki Krivelj were performed during blasting using half-second electric detonators. Delay time of initiation between boreholes was 0.5[s], which led to ten explosions and appropriate rock mass oscillation velocities.

As explosive we used powdered ammonium nitrate for general purposes Amoneks-I manufactured by Trayal from Kruševac. The holes were arranged in a single line while one cartridge of explosive was placed in each hole. Diameter of explosive cartridge was 28 [mm], cartridge length of 0.15 [m], and cartridge weight of 0.1 [kg]. Electric capacitor was used for initiation of the explosive.

During experimental investigations performed in the open pit Veliki Krivelj, the following blasting parameters were specified:

depth of borehole: 0б5 [m],

weight of explosive charge per borehole: 0,10 [kg],

number of boreholes: 10,

distance between boreholes: 1,0 [m],

distance between measuring point and the first borehole: 5,0 [m], delay time between initiation of boreholes: 0,5 [s].

Figure 3 shows a record of rock mass oscillation velocity for blasting in the open pit Veliki Krivelj.

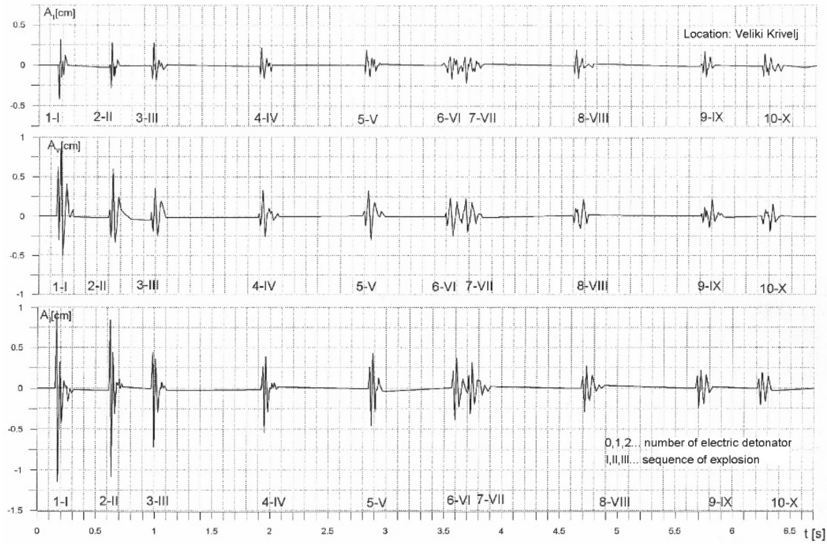

Figure 3. Record of rock mass oscillation velocity for blasting for 1st experiment

From the record can be seen maximal values for all three-rock mass oscillation velocity components. On the basis of that maximal, i.e. resulting rock mass oscillation velocity v_{res} can be defined, applying the formula

$$v_{res} = \sqrt{v_t^2 + v_v^2 + v_l^2} \tag{21}$$

where we have:

v_t - rock mass oscillation velocity horizontal transversal component;

v_v - rock mass oscillation velocity vertical component;

v_l - rock mass oscillation velocity horizontal longitudinal component.

4.2.1 Calculation for parameters of rock mass oscillation equation in the open pit Veliki Krivelj - 1st experiment

The values of distance from the blasting point to the point of observation r, amount of explosive Q, calculated values of reduced distances R, registered values of rock mass oscillation velocities by components v_t, v_v, v_l and resulting rock mass oscillation velocities v_{res} for a total of ten explosions (blasting 1-1st experiment), are given in Table 1.

Table 1

Review of blasting parameters and measurement results - 1st experiment

No	r[m]	Q[kg]	R	v_t[cm/s]	v_v[cm/s]	v_l[cm/s]	v_{res} [cm/s]
1	5,0	0,1	10,7722	0,4200	0,9300	1,1400	1,5300
2	6,0	0,1	12,9266	0,3000	0,6000	1,0500	1,2460
3	7,0	0,1	15,0810	0,2400	0,4000	0,6450	0,7960
4	8,0	0,1	17,2355	0,2000	0,3750	0,5000	0,6562
5	9,0	0,1	19,3899	0,1800	0,4000	0,4750	0,6465
6	10,0	0,1	21,5443	0,1500	0,3000	0,4000	0,5220
7	11,0	0,1	23,6988	0,1900	0,2700	0,3300	0,4668
8	12,0	0,1	25,8532	0,1900	0,2100	0,2900	0,4053
9	13,0	0,1	28,0077	0,1500	0,2150	0,2500	0,3622
10	14,0	0,1	30,1621	0,1400	0,2250	0,2000	0,3320

Based on data given in Table 1, the rock mass oscillation equation is calculated by the formula (1) - by the models 1 and 2. The calculation of the curve was carried out for values of reduced distances from R=10,7722 to R=30,1621 . Thus, curve parameters were calculated enabling us to determine the equation of rock mass oscillation in the form of:

Model 1

$$v_1 = 49{,}4477 \cdot R^{-1{,}470} \qquad (22)$$

Graphic review of the rock mass oscillation equation is shown in figure 2.

Model 2

$$v_2 = 83{,}6601 \cdot R^{-1{,}6540} \qquad (23)$$

On the basis of the obtained equations for rock mass oscillation (22) and (23) it is possible to calculate values of rock mass oscillation velocities for the corresponding reduced distances for models 1 and 2.

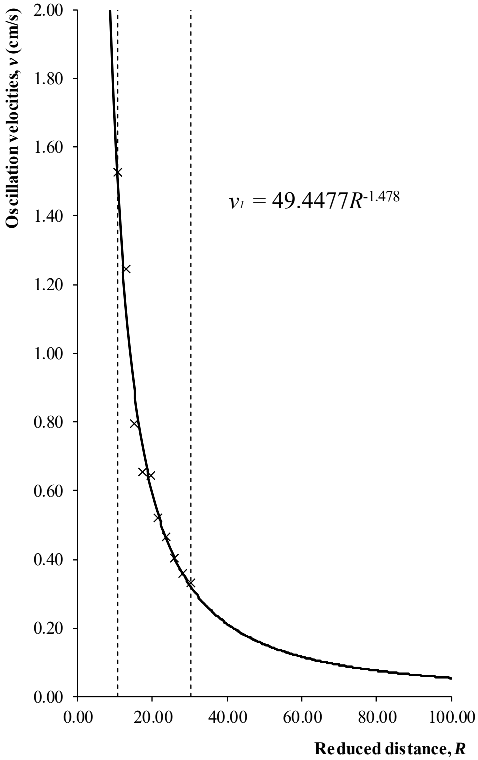

Figure 2. Graphic review of oscillation velocity and reduced distance for 1st experiment

Review of reduced distances R, registered rock mass oscillation velocities v_r, calculated rock mass oscillation velocities v_{i1}, v_{i2}, as well as the difference between registered and calculated oscillation velocities for models 1 and 2 are given in Table 2.

Table 2

Review of recorded and calculated rock mass oscillation velocities for models 1 and 2-1st experiment

No	R	v_r [cm/s]	v_{i1} [cm/s]	v_{i2}[cm/s]	v_r-v_{i1}	v_r-v_{i2}
1	10,7722	1,5300	1,4737	1,6409	0,0563	-0,1109
2	12,9266	1,2460	1,1256	1,2137	0,1204	0,0323
3	15,0810	0,7960	0,8962	0,9406	-0,1002	-0,1446
4	17,2355	0,6562	0,7357	0,7542	-0,0795	-0,0980

No	R	v_r [cm/s]	v_{i1} [cm/s]	v_{i2}[cm/s]	v_r-v_{i1}	v_r-v_{i2}
5	19,3899	0,6465	0,6182	0,6207	0,0283	0,0258
6	21,5443	0,5220	0,5290	0,5214	-0,0070	0,0006
7	23,6988	0,4668	0,4595	0,4454	0,0073	0,0214
8	25,8532	0,4053	0,4041	0,3857	0,0012	0,0196
9	28,0077	0,3622	0,3590	0,3378	0,0032	0,0244
10	30,1621	0,3320	0,3217	0,2989	0,0103	0,0331

Based on the data in Table 2, a statistical analysis was carried out and the following values were obtained:

for model 1:

The curved line dependency index ρ_1 between the reduced distance R and v rock mass oscillation velocity is: ρ_1=0,9880 (there is a strong correlation between R and v , given in the formula (22)).

The maximum difference between the recorded and calculated rock mass oscillation velocities $\varepsilon_{max}=\max|\varepsilon_i|$, is

$$\varepsilon_{max1}=0{,}1204,\ S_1=0{,}0592,\ 3S_1=0{,}1776.$$

Since $\varepsilon_{max1}<3S_1$, supposed functional relationship is accepted as a good one.

for model 2:

The curved line dependency index ρ_2 between the reduced distance and rock mass oscillation velocityv is:

ρ_2 =0,9833 (there is a strong correlation between R and v, given in the formula (23)).

The maximum difference between the recorded and calculated rock mass oscillation velocities $\varepsilon_{max}=\max|\varepsilon_i|$, is:

$$\varepsilon_{max2}=0{,}1446,\ S_2=0{,}0686,\ 3S_1=0{,}2058$$

Since $\varepsilon_{max2}<3S_2$, supposed functional relationship is accepted as a good one.

4.2.2 Calculation for parameters of the rock mass oscillation equation in open pit Veliki Krivelj – 2nd experiment

The values of distance from the blasting point to the point of observation r, amount of explosive Q, calculated values of reduced distances R, registered values of rock mass oscillation velocities by components v_t,v_v,v_l and resulting rock mass oscillation velocities v_{res}

for a total of ten explosions (blasting 2-2nd experiment), are given in Table 3.

Table 3

Review of blasting parameters and measurement results – 2nd experiment

No	r [m]	Q [kg]	R	v_t [cm/s]	v_v [cm/s]	v_i [cm/s]	v_{res} [cm/s]
1	15,0	0,1	32,3165	0,1300	0,2150	0,2100	0,3275
2	16,0	0,1	34,4710	0,1050	0,1500	0,2200	0,2862
3	17,0	0,1	36,6254	0,1000	0,1175	0,1200	0,1955
4	18,0	0,1	38,7798	0,1050	0,0950	0,1300	0,1922
5	19,0	0,1	40,9343	0,0950	0,1000	0,1200	0,1828
6	20,0	0,1	43,0887	0,0800	0,1300	0,1250	0,1973
7	21,0	0,1	45,2431	0,0700	0,1250	0,1200	0,1869
8	22,0	0,1	47,3976	0,0600	0,1050	0,1200	0,1704
9	23,0	0,1	49,5520	0,0550	0,1100	0,1100	0,1650
10	24,0	0,1	51,7064	0,0550	0,1050	0,1000	0,1551

Based on data given in Table 3, the rock mass oscillation equation is calculated by the formula (1) - by the models 1 and 2. The calculation of the curve was carried out for values of reduced distances from R=32,3165 to R=51,7064. Thus, curve parameters were calculated enabling us to determine the equation of rock mass oscillation in the form of:

Model 1

$$v_1=30,0381\cdot R^{-1,3445} \tag{25}$$

Graphic review of the rock mass oscillation equation is shown in figure 3.

Model 2

$$v_2=291,2993\cdot R^{-1,9516} \tag{25}$$

On the basis of the obtained equations for rock mass oscillation (24) and (25) it is possible to calculate values of rock mass oscillation velocities for the corresponding reduced distances for models 1 and 2.

Review of reduced distances R, registered rock mass oscillation velocities v_r, calculated rock mass oscillation velocities v_{i1}, v_{i2}, as well as the difference between registered and calculated oscillation velocities for models 1 and 2 are given in Table 4.

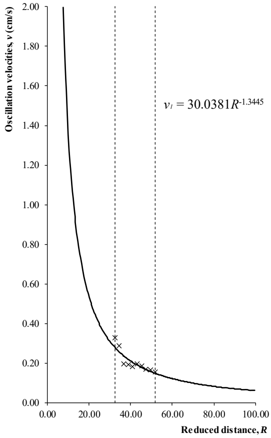

Figure 3. Graphic review of oscillation velocity and reduced distance for 2nd experiment

Table 4

Review of registered and calculated rock mass oscillation velocities for models 1 and 2-2nd experiment

No	R	v_r [cm/s]	v_{i1} [cm/s]	v_{i1}[cm/s]	v_r-v_{i1r}	v_r-v_{i2r}
1	32,3165	0,3275	0,2807	0,3300	0,0468	-0,0025
2	34,4710	0,2862	0,2574	0,2910	0,0288	-0,0048
3	36,6254	0,1955	0,2372	0,2585	-0,0417	-0,0630
4	38,7798	0,1922	0,2197	0,2312	-0,0275	-0,0390
5	40,9343	0,1828	0,2043	0,2081	-0,0215	-0,0253
6	43,0887	0,1973	0,1907	0,1882	0,0066	0,0091
7	45,2431	0,1869	0,1786	0,1711	0,0083	0,0158
8	47,3976	0,1704	0,1677	0,1563	0,0027	0,0141
9	49,5520	0,1650	0,1580	0,1433	0,0070	0,0217
10	51,7064	0,1551	0,1492	0,1319	0,0059	0,0232

Based on the data in Table 4, a statistical analysis was carried out and the following values were obtained:

for model 1:

The curved line dependency index ρ_1 between the reduced distance R and rock mass oscillation velocity v is:

ρ_1=0,8828 (there is a strong correlation between R and v, given in the formula (24)).

The maximum difference between the recorded and calculated rock mass oscillation velocities $\varepsilon_{max1} = \max|\varepsilon_i|$, is:

ε_{max1}=0,0468, S_2=0,0249, $3S_1$=0,0746

Since $\varepsilon_{max1} < 3S_1$, supposed functional relationship is accepted as a good one.

for model 2:

The curved line dependency index ρ_2 between the reduced distance R and rock mass oscillation velociti v is

ρ_2=0,8515 (there is a strong correlation between R and v, given in the formula (25)).

The maximum difference between the recorded and calculated rock mass oscillation velocities $\varepsilon_{max} = |\varepsilon_i|$, is

ε_{max1}=0,630, S_2=0,0277, $3S_1$=0,0832

Since $\varepsilon_{max2} < 3S_2$, supposed functional relationship is accepted as a good one.

4.2.3 Calculation of parameters for rock mass oscillation equation in open pit Veliki Krivelj - 3rd experiment

The values of distance from the blasting point to the point of observation r, amount of explosive Q, calculated values of reduced distances R, registered values of rock mass oscillation velocities by components v_t, v_v, v_l and resulting rock mass oscillation velocities v^{res} for a total of ten explosions (blasting 3-3rd experiment), are given in Table 5.

Table 5

Review of blasting parameters and measurement results - 3rd experiment

No	r [m]	Q [kg]	R	v_t [cm/s]	v_v [cm/s]	v_l [cm/s]	v_{res} [cm/s]
1	25,0	0,1	53,8609	0,0950	0,1600	0,1520	0,2403
2	26,0	0,1	56,0153	0,0830	0,1230	0,1300	0,1973
3	27,0	0,1	58,1697	0,0890	0,1220	0,1400	0,2059
4	28,0	0,1	60,3242	0,0880	0,0930	0,1150	0,1721
5	29,0	0,1	62,4786	0,0720	0,1000	0,1220	0,1734
6	30,0	0,1	64,6330	0,0680	0,1160	0,1400	0,1941
7	31,0	0,1	66,7875	0,0590	0,0700	0,1150	0,1470
8	32,0	0,1	68,9419	0,0660	0,0880	0,1050	0,1521
9	33,0	0,1	71,0963	0,0750	0,1000	0,1240	0,1761
10	34,0	0,1	73,2508	0,0650	0,1050	0,1100	0,1654

Based on data given in Table 5, the rock mass oscillation equation is calculated by the formula (1) - by the models 1 and 2. The calculation of the curve was carried out for values of reduced distances from R=53,8609 to R=73,2508. Thus, curve parameters were calculated enabling us to determine the equation of rock mass oscillation in the form of:

Model 1

$$v_1 = 16{,}9145R^{-1.0947} \tag{26}$$

Graphic review of the rock mass oscillation equation is shown in figure 4.

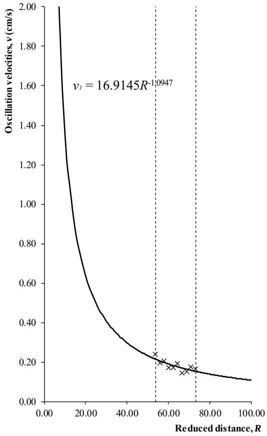

Figure 4. Graphic review of oscillation velocity and reduced distance for 3rd experiment

Model 2

$$v_2 = 54{,}9857R^{-1.3779} \tag{27}$$

On the basis of the obtained equations for rock mass oscillation (26) and (27) it is possible to calculate values of rock mass oscillation velocities for the corresponding reduced distances for models 1 and 2.

Review of reduced distances R, registered rock mass oscillation velocities v_r, calculated rock mass oscillation velocities v_{i1},v_{i2}, as well as the difference between registered and calculated oscillation velocities for models 1 and 2 are given in Table 6.

Table 6

Review of registered and calculated rock mass oscillation velocities for models 1 and 2 - 3rd experiment

No	R	v_r[cm/s]	v_{i1} [cm/s]	v_{i2}[cm/s]	v_r-v_{i1}	v_r-v_{i2}
1	53,8609	0,2403	0,2153	0,2263	0,0250	0,0140
2	56,0153	0,1973	0,2062	0,2144	-0,0089	-0,0171
3	58,1697	0,2059	0,1979	0,2036	0,0080	0,0023
4	60,3242	0,1721	0,1902	0,1936	-0,0181	-0,0215
5	62,4786	0,1734	0,1830	0,1845	-0,0096	-0,0111
6	64,6330	0,1941	0,1763	0,1760	0,0178	0,0181
7	66,7875	0,1470	0,1701	0,1683	-0,0231	-0,0213
8	68,9419	0,1521	0,1643	0,1611	-0,0122	-0,0090
9	71,0963	0,1761	0,1589	0,1544	0,0172	0,0217
10	73,2508	0,1654	0,1538	0,1482	0,0116	0,0172

Based on the data in Table 6, a statistical analysis was carried out and the following values were obtained:

for model 1:

The curved line dependency index ρ_1 between the reduced distance *R* and rock mass oscillation velocity v is:

ρ_1=0,7895 (there is a strong correlation between *R* and *v*, given in the formula (26)).

The maximum difference between the recorded and calculated rock mass oscillation velocities $\varepsilon_{\max} = \max|\varepsilon_i|$, is:

$$\varepsilon_{\max 1}=0,0250,\ S_1=0,0162\ \ 3S_1=0485$$

Since $\varepsilon_{\max 1}<3S_1$, supposed functional relationship is accepted as a good one.

for model 2:

The curved line dependency index ρ_2 between the reduced distance R and rock mass oscillation velocityv is:

ρ_2=0,7813 (there is a strong correlation between *R* and *v*, given in the formula (27)).

The maximum difference between the recorded and calculated rock mass oscillation velocities $\varepsilon_{\max} = \max|\varepsilon_i|$, is:

$$\varepsilon_{\max 2}=0,0217,\ S_2=0,0165\ 3S_2=0494$$

Since $\varepsilon_{\max 2}<3S_2$, supposed functional relationship is accepted as a good one.

4.2.4 Calculation for parameters of rock mass oscillation equation in open pit Veliki Krivelj - 4th experiment

The values of distance from the blasting point to the point of observation r, amount of explosive *Q*, calculated values of reduced distances *R*, registered values of rock mass oscillation velocities by components v_t,v_v,v_l and resulting rock mass oscillation velocities v_{res} for a total of ten explosions (blasting 4-4th experiment), are given in Table 7.

Table 7

Review of blasting parameters and measurement results – 4th experiment

No	r[m]	Q[kg]	R	v_t [cm/s]	v_v [cm/s]	v_i [cm/s]	v_{res} [cm/s]
1	35,0	0,1	75,4052	0,0430	0,0950	0,0810	0,1320
2	36,0	0,1	77,5596	0,0300	0,0800	0,0650	0,1074
3	37,0	0,1	79,7141	0,0220	0,0820	0,0640	0,1063
4	38,0	0,1	81,8685	0,0200	0,0670	0,0500	0,0860
5	39,0	0,1	84,0230	0,0210	0,0680	0,0610	0,0937
6	40,0	0,1	86,1774	0,0240	0,0800	0,0760	0,1129
7	41,0	0,1	88,3318	0,0220	0,0510	0,0510	0,0754
8	42,0	0,1	90,4863	0,0260	0,0580	0,0570	0,0854
9	43,0	0,1	92,6407	0,0330	0,0650	0,0680	0,0997
10	44,0	0,1	94,7951	0,0340	0,0700	0,0680	0,1033

Based on data given in Table 7, the rock mass oscillation equation is calculated by the formula (1) - by the models 1 and 2. The calculation of the curve was carried out for values of reduced distances from *R*=75,4052 do *R*=94,7951. Thus, curve parameters were calculated enabling us to determine the equation of rock mass oscillation in the form of:

Model 1

$$v_1=8{,}3666\cdot R^{-0{,}9989} \qquad (28)$$

Graphic review of the rock mass oscillation equation is given in figure 5.

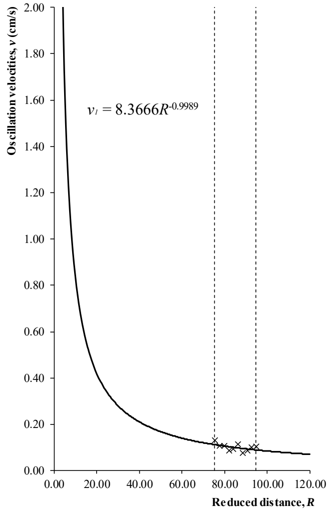

Figure 5. Graphic review of oscillation velocity and reduced distance for 4th experiment

Model 2

$$v_2=29{,}0823\cdot R^{-1{,}2774} \quad (29)$$

On the basis of the obtained equations for rock mass oscillation (28) and (29) it is possible to calculate values of rock mass oscillation velocities for the corresponding reduced distances for models 1 and 2.

Review of reduced distances R, registered rock mass oscillation velocities v_r, calculated rock mass oscillation velocities v_{i1},v_{i2}, as well as the difference between registered and calculated oscillation velocities for models 1 and 2 are given in Table 8.

Based on the data in Table 8, a statistical analysis was carried out and the following values were obtained:

for model 1:

The curved line dependency index ρ_1 between the reduced distance R and rock mass oscillation velocity v is:

ρ_1=0,5334 (there is a significant correlation between R and , given in the formula (28)).

The maximum difference between the recorded and calculated rock mass oscillation velocities ε_{max}=max$|\varepsilon_i|$, is

$$\varepsilon_{max1}=0,0205,\ S_1=0,0130,\ 3S_1=0,0389$$

Since $\varepsilon_{max1}<3S_1$, supposed functional relationship is accepted as a good one.

Table 8

Review of registered and calculated rock mass oscillation velocities for models 1 and 2 - 4th experiment

No	R	v_r[cm/s]	v_{i1} [cm/s]	v_{i2}[cm/s]	v_r-v_{i1}	v_r-v_{i2}
1	75.4052	0,1320	0,1115	0,1163	0,0205	0,0157
2	77.5596	0,1074	0,1084	0,1122	-0,0010	-0,0048
3	79.7141	0,1063	0,1055	0,1083	0,0008	-0,0020
4	81.8685	0,0860	0,1027	0,1047	-0,0167	-0,0187
5	84.0230	0,0937	0,1001	0,1013	-0,0064	-0,0076
6	86.1774	0,1129	0,0976	0,0980	0,0153	0,0149
7	88.3318	0,0754	0,0952	0,0950	-0,0198	-0,0196
8	90.4863	0,0854	0,0929	0,0921	-0,0075	-0,0067
9	92.6407	0,0997	0,0908	0,0894	0,0069	0,0103
10	94,7951	0,1033	0,0887	0,0868	0,0146	0,0165

for model 2:

ρ_1=0,5334 (there is *a* significant correlation between R and , given in the formula (28)).

The maximum difference between the recorded and calculated rock mass oscillation velocities ε_{max}=max$|\varepsilon_i|$, is

$$\varepsilon_{max1}=0,0205,\ S_1=0,0130,\ 3S_1=0,0389$$

Since $\varepsilon_{max1}<3S_1$, supposed functional relationship is accepted as a

The curved line dependency index ρ_2 between the reduced distance R and rock mass oscillation velocity v is:

ρ_2 =0,5215 (there is a significant correlation between R and v, given in the formula (29)).

The maximum difference between the recorded and calculated rock mass oscillation velocities ε_{max}=max$|\varepsilon_i|$, is:

$$\varepsilon_{max2}=0,0187,\ S_2=0,0131,\ 3S_2=0,0392.$$

Since $\varepsilon_{max2}<3S_2$, supposed functional relationship is accepted as a good one.

5. Conclusion

As a link between rock mass oscillation velocity and basic parameters that influence its value the most frequently is used the equation of M.A. Sadovskii. Rock mass oscillation velocity according to equation of Sadovskii is determined based on test blasts for the concrete working environment. Experimental researches have been performed at open pit Veliki Krivelj in eastern Serbia. Specificities of the working environment have been expressed by parameters K and n, that has been conditioned by characteristics of the rock mass and blasting conditions.

In this study parameters K and n in the Sadovskii equation have been determined by way of two models in the given working environment. First model represents the usual method of the Least Squares, and the second model has been derived by applying Lagrange's theorem. Thus, we obtained the adequate functions which represent rock mass oscillation velocity depending on reduced distance. Based on calculated reduced distances, registered rock mass oscillation velocities, calculated rock mass oscillated velocities as well as differences between registered and calculated velocities, statistical analysis was performed calculated adequate indexes of the curved line correlation show that between reduced distance and rock mass oscillation velocity there exist a very strong and significant curved line relationship expressed by obtained functions. As for the estimation of the degree of adequacy and exactness of the selected curve line, besides the mentioned criterion of the curved line dependency index, criterion "3S" has also been taken into consideration. According to this criterion functional relationship is accepted as a good one.

Comparing values of the registered rock mass oscillation velocities with corresponding calculated values, we see that they have approximately similar values. Based on the values obtained by statistical analysis we conclude that both models can be used to calculate rock mass oscillation velocities.

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DEVELOPMENT OF MANAGEMENT CONTROL METHODS OF RESPONSIBLE ENGINEERS OF THE STRUCTURES IN DEEP QUARRIES

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Abstract

Purpose. Develop ways to control the state boards of slopes and responsible engineering structures in the deep quarries for increasing efficiency and safety conduction of open development of fields of useful fossil based on creating single system of geomechanical monitoring and management by natural and mining technical factors.

Methods. When performing work there are used numbers of complex methods of study, including scientific generalizations, experimental research in the laboratory and industry conditions with using graphic analytical and mathematical modeling, as well as mathematical statistics and correlation analyze of developed management methods of responsible engineering structures in the quarries.

Findings. Worked out management control method of slopes of boards and responsible engineering structures is implemented in the quarry of Muruntau Central oremanagement of state enterprise Navoi mining and metallurgy plant. Results of implementing scientific results allowed to increase efficiency and safety conduction of mining works, ensure support of massif quarry and located on it complex of high angle conveyor in the stable condition for full period of treatment.

Originality. By graphical analytical method it is established to measure the radius of slipping line of high wall boards depending on the coupling coefficient, specific weight and angle of intenal friction of massif of rock mass. Determined the mass of elementary blasthole of drilling and blasting during preventive presplitting in the deep quarry. Worked out monitoring system taking into account effects of explosive and technological loads in the complex of cyclone flowing technology – ore with high angle conveyor. Worked out complex calculation method of voltage and deformation condition of board of deep quarry and construction of high angle conveyor, allowing to ensure safety conduction of mining works. Reducing level of impact of mass explosion in the deep quarries on the protected facilities and engineering structures is achieved through the development of seismic safety technology of conducting drilling and blasting operations and determination of their effective parameters taking into account accecsable velocity of massif oscillation, type of blasting materials and distance from center of explosion to the protected facility. Determined the mass of blasthole charges of blasting materials on one slackening and recommended explosion schemes and slackening intervals of reference data, ensuring reduction of level of seismic oscillation and increasing the storage of near edge zone massif and engineering structures in the deep quarries.

Practical implications. Worked out seismic technologies of drilling and blasting and determined their effective parametes, ensuring minimization of dynamical load, appearing during the work of mining and technological equipment in the deep quarries. Worked out calculation methods of effective parametes of drilling and blasting taking into account physical and mechanical and mining and technological features of massif of deep quarries. Worked out engineering calculation method of effective paramets of drilling and blasting, reducing seismic impact of massif explosion during combined system of developing mineral deposits. Worked out engineering calculation of mass of blasthole charges of drilling and blasting on the one slackeeing a nd recommended blasting schemes and slackening intervals of reference data ensuring reduction of the level of seismic oscillation and increasing storage of near edge zone massif and responsible engineering structures in the deep quarries.

Keywords: deep quarry, management of the state of the slopes of the sides, responsible engineering structures, improving the efficiency and safety of open mining, geomechanical monitoring system, natural and mining factors.

1. Introduction

The main tendencies in the development of open method of mining are to attract to the development of minings with amended mining and geological conditions and increase of scale of quarries. The annual output of minerals in the world is about 20 billion tons. In open pits up to 90% brown and 20% coal, 70% of ores of metals, 95% of non-metallic minerals are mined. The basic directions of the increase of efficiency of mineral resources exploitation associated with the improvement of technique and technology of mining operations in deep horizons of open pits with the use of cyclic-flow technology (CFT).

The Republic of Uzbekistan has implemented large-scale policies for effective development of mining and metallurgical industry. Along these lines, particularly, in the deep horizons of open pits heavy duty conveyor systems, road and rail transport are being used, hard-to-recover and complex structural ores are being involved into the development in large amounts.

On a global scale with the aim of improving the unified system of geomechanical monitoring, and management of natural and mining-technical factors in deep open pit mines, special attention is given to improving the methods of state management of pit walls and benches and responsible engineering structures. From this point, the vision the following directions of scientific researches deserve special attention: the deepening of theoretical research of processes of fore pit walls massif destruction of deep opencasts on the basis of improvement of the unified system of geomechanical monitoring; a more precise definition of the impact of mining-technical factors on the preservation of fore pit walls massif and responsible engineering structures, and also control their condition in deep pits; the deepening of experimental studies of the deformation of the edge of the massif under the action of mass explosions, taking into account regional factors; development of effective methods of management of technology of drilling and blasting in pre-controlling the area and as responsible engineering structures of deep pits; effective use of different vehicles in separate processing zones and stages of development of mineral deposits in open pits. Thereby, further development of the application of CFT with SIC using technical and organizational means of achieving design solutions to enhance ways to manage the condition of walls and ledges` slope, as well as the critical engineering structures in deep pits to ensure their long-term safety and trouble-free operation is an important scientific problem of subsoil use.

2. The main part of the article

During investiogation stability condition of boards of Muruntau quarry it is established that the level of block structure at the territory of far fields from critical. Determined separate areas of boards that requires high attention during conduction rock mass in the quarry on whci can be started the development of local deformation that leads to the mixing supports and may damage to the work of high angle conveyor. For the purpose of more reliable control over mechanical condition of near edge zone massif at the territory of cyclone flowing technology – ore with high angle conveyor established additional seismic Delta Geon station that expanded the possibility of acting geodynamical monitoring system. Dispite the fact that stable evaluation and calculation parameters of board of Muruntau quarry in the area of placing high angle conveyor has difficulties due to various mass capability of rock mass render resistance distribution of seismic waves in the different direction from explosion and earthquake.

Rocks mass massif destroyed by constant external effect and not in the natural condition. Quarry board especially is complicated with siltstones and carbonaceous mica-quartz composition with lenses and mica-quartz composition with lenses and interbedded shales of different composition. As a result both board area and cyclone flowing technology complex- ore with high angle conveyor locating in the same from epicenter of expecting explosions and earthquake, in the different level can be subjected to long seismic effect. Complexity in the evaluation of geodynamical processes appearing in the near edge zone massif, calls the necessity in the production of of special research for concrete condition of of reviwed quarry for the purpose of substantion of high angle conveyor resistance on which is is placed. Control methods of slopes and their calculation parameters for increasing effective and safety open developed fields. Famous methods of designing paramaters of drilling and blasting does not fully meet criteries and limitations for ensuring safety of ersponsible engineering structures from seismic effect of explosion.

So, inspite of number of researches, the problem of ensuring stability of slopes in their limited condition with responsible engineering structures due to difficulties and high difference mining and technological and hydrogeological conditions of deposits for powerfull quarries have not been completely solved.

Determined splitting lines of near edge zones rock mass by graphic and analytical methods in the quarries. It is established that measuring radious of splitting lines of slope boards depending on coupling coefficient, specific weight and angle of internal friction of of rock mass, as well as angle of slope of boards of deep quarries. Based on received values worked out program of calculating opetimal parameters of stability of near edge zone of mass rock.

As a result of conducted investigations, worked out block-scheme of sequence of calculating parameters and construction of splitting lines of board massif of rocks for determining optimal parameters of competence of board massif of rocks

It is investigated the regularity of changing mass of elementary blasthole charges, consisting of several spherical charges of blasting materials furing preventive crack appearance in the deep quarry depending on coefficient, taken into account durability limit, frac rate and Puasson rock rate as well as depth of well, the length of stem and distance between contour blasthole charges, based on which engineering calculation program is developed.

There is a formula of calculation of mass of elementary of blasthole carges of drilling and blasting taking into account friction coefficient of rock mass during preventive presplitting, kg

$$Q = \frac{\varepsilon K a^{2} \sigma_{p}^{1,5} \left(L - l_{заб} \right) (1 - 2\mu)}{K_{тр} \sqrt{E}}, \qquad (1)$$

where ε - acoustic indicator of friction of rock mass; ε=1,6·10^{-16}м$^{-1}$; K=1,7-1,85 coefficient taking into account strength limit of rock mass (lower edge for small hard rocks, upper - for more hard); a – the distance between countoouring blasthole charges of drilling and blasting, m; $\sigma_{р}$ - firmness of rocks for blasting during dynamical mode of load, MPa; L - the depth of hole, м; $l_{заб}$ - the length of stemming of blasthole charges of drilling and blasting, m; μ - Puasson coefficient of rock mass; **Ошибка! Источник ссылки не найден.** - coefficient taking anto account friction of massif; E - elasticity module of rock mass, MPa.

Received values shows that with increasing coefficient, taken into account durability limit of blasted rocks from 1,7 to 1,85 mass of elementary blasthole charge of blasting materials is being increased on dependence and accordingly consists 2,96 and 3,23 kg. With increasing distance between blasthole charges of blasting materials from 2,5 to 4 m is being increased on parabolic dependence and accordingly consists 2,1 va 5.3 kg.

Also, it is established with increasing the depth of contour of blasthole charges from 12 to 18 m, the mass of elementary charges will be increased on line dependence and accordingly consists 3,0 and 5,2 kg.

With increasing frac rate of rocks from 1,06 to 1,2, the mass of elementary charges will be decreased on hyperbolic dependence and accordingly consists from 0,3 to 0,36 mass of elementary charges will be decreased on line dependence and consists 3,05 and 2,16 kg.

Experimental investigations established destruction mechanism of contoured massif with using construction of charge with inert gas. It is established that during explosion asymmetric mass destruction happens and impact of explosion to the secured massif will be reduced at the expense of energy absorption during reinstallation of inert stemming.

In order to evaluate the effect of mss energy of blasthole charges of drilling and blasting on the condition of near edge zone masiif of lower horizons carried out complex researches with using seismic explosration methid.

To solve these issues theare used seismic receivers CMB-130 and seismic statitons ACM-12-OB; the record of signals was performed by oscilloscope H-700. Excitation of elastic waves in the middle was carried out hit of sledgehammer with weight 6 kg through metallic gasket. Moment of a wave to the seismic receiver is determined according to the first performance of seismogram.

In the second step there are explored measuring law of countoured vilation of rock mass with the surface of high wall and boards of quarries. Drilling countoured holes was carried out by drilling machines СБШ-250МН and СБУ-125. To ensure exact orientation space of holes drilling machines have been equipped qith orientarors which countoured based on aviation hyro compass.

Due to the water content of the massif and a limited range of blasting materials during production of experimental research water proof emulsion blasting materias were used which are produced in the plants of the Republic of Uzbekistan. Initiation of blasthole charges was carried out intermediate detonator patronized of ЭВВ brand Nobelit-216Z. Mounting explosion network was carried out detonating cord brand ДШЭ-12. Explosion of countouring blastholes and blast hole fragmentations in the near edge zone of massif was carried our separately. Contouring blasthole charges of blasting materials blasted instantly. The explosion of blasthole charges of blasting materials in the near edge zone was carried out cross rows and according to diagonal scheme. Slackeing was provided with using pyrotechnic retarder КЗДШ -69 with interval of slackening 10, 35 and 50 ms.

Blasthole charges consisting of ЭВВ in linen sleeve with radial clearance have been placed in contouring holes. 60 and 80 m hose has been used. The hose was dipped into the well; one end of hose was tied with load. The second end of the hose was attached to the cross bar at the well mouth. After this, the well was filled with emulsion explosive charges through the hose. The intermediate detonator Nobelit-216Z was placed on the side of the well and was squeezed against the charge under its own weight.

In the third step, effective parameters of drilling and blasting operations of countering explosion was determined in the production condition. The main complex of research was carried out in the sectors No 1 and 2 in the Muruntau quarry.

In order to get screen gapping holes have been drilled under angle of 60° to horizons «with overlap». The distance between contouring holes was 3,0 m, herewith small resistance line amoung contouring blasthole charges was 2,6 m.

Fragmentation of rocks in the near edge zone in the sector №1 was carried out with the help of blast hole charges with using ЭВВ in the inclined (α_c=75°) holes. Distance from close blasthole charges to screen gapping in the upper part is is 6,8 m and in the lower part is 2,5 m.

Specific flow of blasting materials in the contoring zone was used from the condition of qualitative fragmentation of rock mass based on professional working experience of this enterprise.

Fragmentation of rock mass in the contouring zone in the section № 2 was carried in two step. In the first step the loosening of rocks of upper layer with capacity of 8 m was carried out during reduced specific flow of blasting materials. Simultaneously explosion of blasthole charges was carried out for alignment of contouring zone in the sector №1. Small resistance line from blasthole charges to screen gapping is 4,5 m.

Experimental and industrial research on evaluating deformation of contoured mass under the action explosion energy of contoured blasthole charges of blasting materials. Observance over deformation of massif was carried out on deep benchmakrs. Observance on plugged benchmarks on the surface was carried out to compare and evaluate possibility of using data on deformation of surface layer.

Deep benchmarks had the following structure. The benchmarks made of P-65 rail, was run into the well with 250 mm diameter. Guides were welded into the bottom of the rail, allowing it to make alignment of the rail inside the well and at the same time serve as an "anchor" for the benchmark. After centering, the lower part of the benchmark with a length of 2,0 m was covered with grade 300 concrete. Marks in the form of a cross were made at the upper end of the benchmarks using chisel. In the future, the measurements were made on them.

Researches established that, in the formation process of screening gap during the explosion of contoured blasthole charge includes three steps: development of crack network (fragmentation of rocks) inside the interwell barrier, expansion of the gap and ripping fragmented rocks, partly collapse as a result of unloading massif.

It is established that opening screening gap at the expense of closing cracks in adjacent massif will rise proportional quadrate of destruction power zone during refilling it fragmented rocks and maintained only when filling it with fractured rocks. Cracked interstice of filler does not depend on parameters of drilling and blasting operations and makes up 30%, whereas the maximal width of zone of crack development in interwell barrier consists of 0,7-0,8d_{well} (d_{well} - well diameter). It is established that changing the size of zone with destructed inter block connection in the contoured mass will rise, together with the mass of contour of blasthole charges of blasting materials coming to the single load surface of proportional stage 0,25 and substantially depends on mass destruction of previous explosions.

Investigations established that the power of upper part of high wall with strong destruction zone due to explosion of blasthole charge of above located horizons is equal to 3-4 m and is considered to be potentially unstable, medium velocity of distribution of longitudinal waves in this zone is approximately equal to 450 m/s. Effective depth of subgrade drilling of blasting charges is 6-7d_{ch} (wher d_{ch} - charge diameter of blasting materials).

The parameters of vibration appearing in the rock mass, during operation of processing equipment open-cast were established. Researched established that vibration appearing in the rock mass, during operation of excavator carries cyclic character. Maximal value of vibration level of massif appears during unloading rocks to the carriage body of dump truck from excavator grab. When processing boards reserves during conducting loading works near the complex engineering structures and location of underground rock productions, it is necessary to take into account the factor of cyclic vibration, appearing during operation mining machinery and it is necessary to work out measures on limitation of the intensity of conducting mining operations in this region.

Dynamic properties of high angle conveyor during the influence appearing in the conditions of Muruntau quarry was analyzed. Obtained results show that main forms are inflection of framed structure in the flatness of slope. Compounded calculation model of oscillation support of high angle conveyor construction and East board allows carrying out complex calculation on determining voltage deformation condition of East board and construction of high angle conveyor and resonance mode during dynamic loads, appearing during conducting mining operations in the Muruntau quarry.

According to this technology, core wells are drilled at the place where the ledge height equal 15 meters and explosives according to the passport of performing drilling and blasting operations for this quarry. At a distance of 1 m from boundary contour of the quarry, small diameter wells with 110 mm diameter are drilled along the contour array at a distance of 3 m from each other in order to form constructions from "elbows" with head angle of 45° and manufactured from 2 mm thick metal and welded to each another in the plantar part on both sides, so that their base rested upon the borehole wall.

Using a method of producing stable slopes of benches in quarries using borehole blasting charges of a small diameter with cumulative effect on the contour of the array provides a smooth separation of the array through its location, minimizing the seismic effect of mass explosion near the edge of the array and engineering structures, allowing better use of the explosion energy of main downhole explosive charges.

Optimal distance from outer screen border till protective site is calculated from the following equation

$$R_{э} \leq 5d_{n}\left[20\frac{\sqrt[3]{Q}}{C_{p}\cdot t_{n}}-1\right] \tag{2}$$

d_n - diameter of pre-splitting well or width of screen, m; Q - mass of simultaneously exploding charges, kg; C_p - the velocity of longitudinal wave, m/s; t_n - the durability of positive phase of predominant seismic oscillation, s.

The gap formed by the contour well explosives charges, provides wall stability for deep wells and essential engineering structures from the effects of massive explosions on the condition that the width of the shielding gap - h is greater than the displacement - A, particles of the surrounding array, that is $h \geq A$. The amount of displacement - A is defined by the formula

$$A=3{,}75\cdot10^{2{,}5}\cdot Q^{0{,}6}\,(g/p\cdot C_{\text{p}})^{0{,}5}R^{1{,}25}, \qquad (3)$$

where Q - the weight of the explosive charge, kg; R - the distance between the nearest borehole loosening and slotted wells, m; C_{p} - longitudinal wave propagation velocity in the rock, m/s; p - the density of the surrounding rock, kg/m^3; g - the acceleration of gravity, m/s^2.

In the calculation, the weight of the two neighboring borehole explosive charges is taken into account, which is defined by the formula

$$Q=1{,}41Q_{well}, \qquad (4)$$

where Q_{well} - the explosive charge weight in a single well, kg.

The width of the shielding gap - h is determined from the expression, sm

$$h = \pi d_w^2/4a\left(100/\sigma_{сж}\right)^{0{,}75\sqrt[3]{K}}, \qquad (5)$$

where d_{w} - well diameter, sm; a - the distance between the splitting wells deviated contour holes; σ_{comp} - the resistance of the rock into uniaxial compression, MPa; K - loading factor.

With the known properties of the quarry rock by the formula (3), determine the amount of solid particle displacement at different distances from the explosion or the mass of the downhole explosive charges to ensure displacement not exceeding the width of the gap calculated according to the formula (5).

The methods of blasting ensuring the safety of massif and engineering facilities in the quarries from the seismic forces of mass explosions. The generally accepted criterion for assessing the seismic force of the explosion is the velocity of the medium on the horizontal displacement component

$$U = K\left[\sqrt[3]{Q}/R\right]^n, \qquad (6)$$

where K - the coefficient depending on the ground conditions, ranging in most cases from the values of 200 and 350-500; n - index of seismic wave attenuation which is determined by the formula

$$n = 2 - \frac{\mu}{1-\mu}. \qquad (7)$$

Preservation of near edge massif and engineering facilities will be provided, if the deformation of the rock board and at the base of the protected structures, caused by the explosions force, will not go beyond the horizontal component of the velocity oscillations $U_{доп} \geq U_x$, which is dependent on the horizontal distance from the explosion (R) is described by the equation, sm/s

$$U_x = 450 \cdot \overline{R}^{-1.85}, \tag{8}$$

and evaluation of the acceptable fluctuations rate will be

$$\sigma_{add} = \rho \cdot C_a \cdot U_{add}, \tag{9}$$

where σ_{add} - to load at which the rock samples of the quarry are not destroyed by multiple dynamic effects, kgm/ sm^2 s^2; ρ - average density of rocks, kg/m^3; C_a - the average value of the velocity of longitudinal waves in m/s; U_{add} - mass velocity oscillations (displacements) of the array to be taken as a valid criterion for assessing the impact of seismic waves on the array.

Assuming the value of distance from the explosion site to the protected facility, you can determine the mass of seismic downhole explosive charges for the instantaneous and short-delay explosion (see table. 1).

Table 1

Seismic mass of explosive charges per one slowdown

R, m	30	40	50	60	80	100	120	150	200	300	400	500
Q_{ins}, kg	18	43	84	146	346	657	1170	2280	5400	18200	43200	84400
Q_{sd}, kg	12	29	57	98	232	452	780	1530	3620	12200	28900	56500

Identified the weight of downhole explosive charges per slowdown and recommended the delay interval of non-electrical systems for mid-blasting rocks between the rows of wells - 67 ms, between the wells in a row - 42 ms; for easily-blasted rocks, respectively, 109 and 67 ms and diagonal blasting scheme which bring down the level of seismic vibrations by 1,5-2 times, as well as improving the preservation of near edge massif and essential engineering structures in the deep pits from the influence of seismic mass explosion.

In order to determine the effective parameters of drilling and blasting at the combined development of deposits in terms of seismic safety, in addition to a fixed station, mounted on the surface of Muruntau quarry board, «Delta-Geon-2» stationary seismic station was additionally installed in the horizontal development of the mine M (mountains. +0,0 m), which makes the registration of seismic signals. As a result of research, received information about the reaction of the mine in M mountain ranges on external dynamic effect under high rock pressure for the first time. It was revealed that the nature of the seismic wave attenuation has got a significant difference for the surface and underground arrays, due to both structural features of the array, and its stress-strain state. It was established that for horizontal oscillations, the shift to quarry bowl has greater amplitude than the shift in the direction of the unbroken open development array. However, the maximum amplitude is specific to the vertical oscillations, which is explained by the effect known in seismology as effect of formation stress concentration zone near fracture zones and the interaction of the drooping and reflected waves from the fracture of seismic waves. But in this case the surface nonuniformity is self-generated contour.

Thus, based on the study received from seismograms, developed a method for calculating the parameters of the high pillar between the bottom of a deep pit and underground mines, determined its effectiveness parameters with a view of seismic influence of mass explosions leading to a deep quarry, on the basis of a charge weight per one delay, developed for mining applications of Muruntau 50 m.

It was established that the combined mining of the deposit blasting should be done separately from the specific usage of explosives not exceeding 1,2 kg/m^3, and the delay interval between the well charges in open and underground work should not exceed 35 ms.

The figure shows the scheme of single system of geomechanical monitoring in the Muruntau quarry, recommended for observing and assessment of rock mass condition through time and in space.

For the condition of Muruntau quarry when increasing its process depth for providing safety of mining operations, following recommendations are suggested: carrying out systematic instrumental observation for the purpose of studying size of deformation velocity of different board sections; in case of noticeable increase of deformation velocity of board section, carry out investigation of board massif structure for determining cracks or advantageous crack systems for formation of landslide orientation; inspection of performance of necessary appearance of landslide; loading upper side of boards of quarry that may be achieved by decreasing the angle of the slope.

It is established that changing boundary conditions due to deepening of quarry, causes redistribution of intension components and increase in the volume of rock mass located in unstable condition. According to results of geomechanical analysis, North board (dropping layers to the excavation side, presence of streak of carbonic and micaceous schist with low indicators of resistance of moving rocks) and South board (presence of a large Southern fracture) are referred as potential negative sections of board massif, on which probability of board deformation is increased.

Geodynamical monitoring system and controlling informational parameter in the process of conduction mining operations implemented in Muruntau quarry, allows in the stage of planning mining work and draw up reliable forecast of stability of boards, install place of possible deformation and take actions on preventing, thereby, ensuring high safety of mining works in the condition of continuous working production.

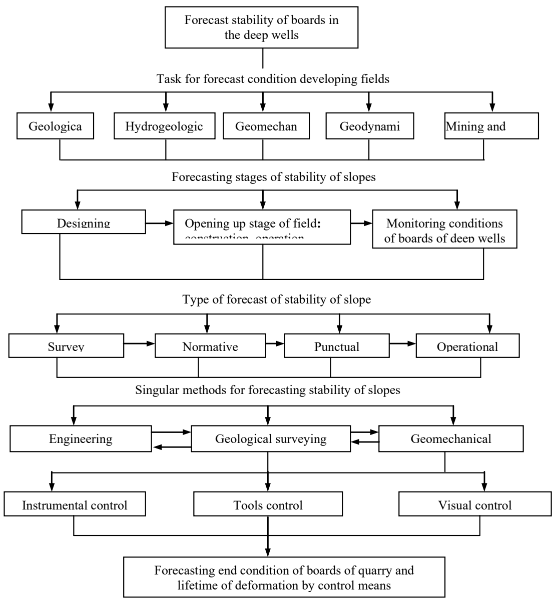

Fig. Scheme for integrated system of geomechanical monitoring in the Muruntau quarry

Main parameters of seismic effect of mass explosion in the mining production and difference between reactions of quarry rock mass in the dynamical effect on surface and in underground production were established. Analytical method for calculations of parameters of explosive operations, ensuring acceptable seismic load from mass explosion in underground rock production during transition to combined production fields was suggested.

Suggested scientifically-based technological and technical solutions on producing new control methods of deformation processes, allows to: lead the damage from local landslides to minimum and provide support of massif board of quarry located on high angle conveyor unit in stable conditions for full period of processing fields; optimizing quarry boundary due to implementation of rational parameters of slopes and boards and improve the depth of open production during ensuring safety condition of mining operations in deep horizons.

Results of implemented recommendations to the practice are indicated in the table 2.

Table 2

Results of implemented process on methods of controlling the conditions of essential engineering structures in the quarries

Solved tasks	Results of implementation
Mathematical modeling of board mass of quarry	– worked out mathematical model on determining splitting line in homogenous slope and computer programming (registration certificate); – established ways of reducing width of remained deformation zone during contour blasting: a) selection of explosive parameters, ensuring voltage limit in dropping wave of compression; b) selection of charge parameters ensuring to create screening gap with increased protection method.
Calculating stability of board massif of quarry in the location place of essential engineering structures	– developed computer programming on determining rational parameters of quarry boards taking in account seismic effect of earthquake; – worked out monitoring system on ensuring stable operation of conveyor ore transportation complex ore from high angle conveyor; –developed method of numerous calculation of voltage and deformation condition of quarry board, sequence and program for solving tasks.

Determination effect of mining and technological factors for the storage of board mass and engineering structures	– optimized parameters of drilling and blasting operations ensuring minimizing seismic effect of mass explosion in board massif and engineering structures; – recommended to take into consideration the factor of cyclic vibration appearing during operation of excavator and develop action on limiting intensity conduction of mining operations: during processing boards reserves; near to engineering structures; location of underground mining production.
Working out technical solution on seismic technology of drilling and blasting operation in the contour zone of quarry ensuring the storage of boards massifs and engineering structures	– determined calculation method of seismic weight of charges for one moderation during mass explosion. Recommended intervals of moderation and explosion scheme ensuring reducing the level of seismic oscillation in 1.5-2 times; – worked out monographic calculation methods of parameters of drilling and blasting; – suggested calculation method of drilling and blasting during combined process of fields.

Real economic effect from implemented recommendations is 336,644 mln. soums per year.

3. Conclusions

1. Using semigraphical method, change in the radius of the slip line of bench face, depending on the friction coefficient, specific weight and internal friction angle of the rock mass and the angle of the slope side of deep pits was identified. On the basis of these relationships, a program for calculating the optimal parameters of the stabile near-edge rock mass was developed.

2. Determined the weight of elementary downhole explosive charge during preliminary presplitting of deep career depending on factors, considering the strength, fracture and Poisson, and the depth of the well, the length of the stemming and the distance between the contour charges.

3. Long-term safety and trouble-free operation in essential engineering structures depending on the developed monitoring system that takes into account the impact of blasting and processing loads on conveyor ore transportation complex - high angle conveyor ore.

4. It has been established that during conducting the work near the pit walls and underground mining, the safety of mining operations will be increased due to the impact of dynamic loads generated

by excavation and loading equipment. The complex method of calculating the stress-strain condition of deep quarry board and construction of high angle conveyor, depending on the direction of the seismic wave with respect to its axis, which limits the intensity of mining operations.

5. Reduction of the impact of mass explosion in deep pits on the guarded facilities and engineering structures is achieved through the development of seismic safe technologies of drilling and blasting, and determine their effective parameters, taking into account the speed limit of the array fluctuations, type of the blasting charge, and the distance from the center of the explosion to the guarded facility.

6. The safety and effectiveness of the deep pits is determined by the methods used by management (technology and technical means) over the condition of essential engineering structures, over occurring deformation processes and the creation of a unified system of geomechanical monitoring of the quarry.

7. Determine the weight of the downhole explosive charges per slowdown and recommended the delay interval of non-electrical systems and as well as diagonal scheme of blasting rock mass, in order to bring down the level of seismic vibrations and increase the safety of massif and essential engineering structures in the deep pits from the seismic mass explosion effects.

8. Developed and industrially tested method of calculation of the effective parameters of drilling and blasting operations, taking into account the physical and mechanical, mining and processing properties of the array of deep pits.

9. Developed and industrially tested method of engineering calculation of the effective parameters of drilling and blasting operations, which reduce the seismic effect of mass explosions at a combined system of development of mineral deposits.

10. Developed and industrially tested method of engineering evaluation of the mass of downhole blasting charges per slowdown and recommended the delay interval of non-electrical systems, as well as the diagonal array of blasting scheme, to ensure reduction of seismic vibrations and increase in the safety of massif and essential engineering structures in deep pits.

11. The theoretical basis of stress-strain condition of deep quarry board and high angle conveyor design, depending on the direction of the seismic wave with respect to its axis, which limits the intensity of mining operations in the area, based on which developed a comprehensive methodology for their calculation.

12. Identified and experimentally tested the effectiveness of the parameters of the deep quarry boards located in a seismically hazardous area, allowing to reduce the risk of dangerous strains of near edge massif and to ensure safe conditions in the mine for the whole period of its operation.

The economic effect from the implementation of the developed recommendations and technology parameters was 336,644,000 soums per year.

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COMPOSITE MATERIALS FOR UNDERGROUND CONSTRUCTIONS

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Abstract

With the development of construction of underground facilities as well as new steps forward in the field of technology of materials and the development of composite materials, new modern materials have emerged and found their usage in support construction of underground facilities, among other things. Modern composite materials of support constructions are the subject of this research.

A review of bibliography, student books, scientific research papers, as well as technical documentation of certain mining companies has been carried out in order to get better acquianted with the characteristics of composite materials and the scope of their application in underground construction.

The outcome of this paper is a detailed overview of physical, mechanical, technical and economic characteristics of modern support constructions (a fiberglass anchor support combined with steel mesh and micro-reinforced shotcrete support) which were used in the realization of two projects of construction and support construction of underground facilities of the „Rudnik“ mine in the Republic of Serbia and one infrastructural facility (Dekani tunnel in Slovenia).

As can be concluded from the above mentioned examples, the introduction of composite materials (and the replacement of traditional support constructions with modern ones) is justified due to the increase of safety of the work force during construction of facilities, from an economic aspect, as well as the need to save time during installation of support construction and construction of underground facilities.

Introduction

The need for construction of underground mining facilities, as well as infrastructural ones, has been growing daily. The construction technology of underground facilities aims to maximize the use of mechanization and automatization of work operations during construction of facilities. Support construction of underground facilities is one of the most difficult and complicated work operations and it takes a significant amount of time within the construction cycle.

The size of the cross section of underground facilities has grown bigger over time, the main reasons for such growth being the use of self propelled diesel machinery of relatively big dimensions and the need for ventilation of deep mines. As the support construction of underground facilities is conditioned by the size and the shape of the cross section, the facilities themselves and some traditional types of support construction have been kept back over time and the advantage was given to modern support constructions made from modern materials, which meet both the technical conditions (Jovanović, 1995) such as construction strength (bearing capacity), stability, elasticity and structural durability and economic conditions and fatigue strength.

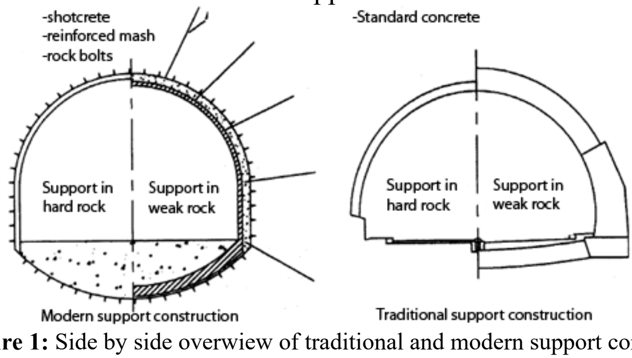
Previous observations primarily relate to support construction of underground mining facilities, while in the case of tunnels as main representatives of underground infrastructural facilities, support construction has always been different, to a certain extent. The main reason is the exploatation life of the facility which is, in the case of infrastructural needs significantly higher than underground needs and is planned for decades. The sizes of cross sections of underground facilities in mines with underground exploatation today are very similar to sizes of cross cuts of underground facilities in tunnels, and for that reason support constructions present today in tunnel construction are very similar to support constructions of modern mines with underground exploatation.

The materials used for support construction are also represented in any form of surface construction, bearing in mind that with support construction exploatation the needs have to be met, such as: applying high pressure, receiving of uneven load, resistance to specific atmospheric conditions (high humidity, aggressive atmospheric conditions) and water resistance (Jovanović, 1994)... Priority is given to materials that are lighter but meet the criteria of strength and bearing capacity. The most important and basic support construction materials are wood, steel, concrete and stone (Jovanović, 1994; Jovanović et al., 1990).) while recently various modern materials have been included (thermostable plastics, epoxy and synthetic resin and polymeric fibrous composites).

Support constructions

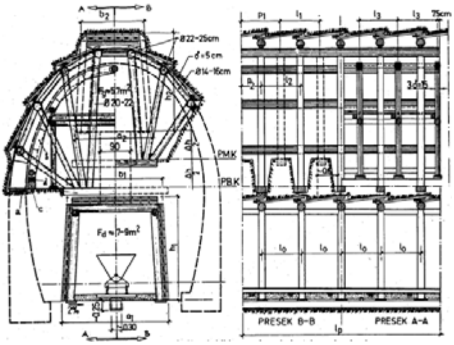
The main task of support construction is the maintenance of the underground facility during construction and usage, so as not to collapse and to protect the employed staff, as well as prevention of penetration of underground waters into the facility. (Antunović Kobliška, 1973.). Support construction is performed using support, i.e different forms of support construction. Support constructions are built into an excavated cross section of the underground facilities so as to stabilize the contour of the facility and should receive the load of the rock mass through which the facility has been built. In the previous chapter the main characteristics of support materials used for support construction have been given. The chosen support construction has to meet the technical and economic conditions, as well as fatigue strength.

Support constructions can be classified according to performance principle, constructive material, the type of underground facility, exploatation life, constructive features, way of dealing with underground pressure and special assignment (Antunović Kobliška, 1973.). Furthermore, support constructions can be divided into two groups (Tomanović, 2015.). Picture 1 shows the side-by-side overwiew of traditional and modern support construction.



Picture 1: Side by side overwiew of traditional and modern support construction. (Tomanović, 2015)

The group of traditional support constructions includes support constructions built during construction of underground facilities using traditional methods of construction (by drilling and low mechanized operations). The main characteristic of this type of support construction is the wood support construction (round or cut). As the construction technology itself, where high profile facilities are concerned, is such that excavation is performed in segments which are being broken out or excavated independently and support construction itself is done in independent segments in appropriate order. One example of this support construction is shown in picture 2.



Picture 2. Traditional support construction (Tomanović, 2015)

In addition to wood construction, steel constructions and concrete support constructions are also part of traditional support constructions. Traditional support construction, as has been stated previously, is connected to older construction technology of underground facilities which have been almost entirely pushed back where modern underground construction is concerned, whether it is connected with underground mining facilities or other types of facilities (infrastructural, hydrotechnical, communication etc.). Most often traditional support construction is constructed using only one type of constructive material (wood, steel or concrete).

Modern support constructions are built successively with excavation in phases that need to be coordinated with time during which excavation maintains stability. This type of support construction is included during support construction of high profile facilities regardless of the method used (mechanized construction using tunnel machines or classic drilling operations). The classic method of support construction by using steel frames in combination with longer steel or tin connections (which are used for reinforcing the support construction) has been substituted with support construction using micro-reinforced shotcrete with, if necessary, installation of radially arranged anchors. This type of support construction forms a „bearing ring“ of rock mass which combined with micro-reinforced shotcrete of high initial strength forms a secure support construction of an excavated facility. (Tomanović, 2015).

In practice during installation of micro-reinforced shotcrete, reinforcing mesh and anchors, continuous monitoring of deformation of the excavation contour is carried out. Support construction with NATM method (New Austrian Method) is based on usage of micro-reinforced shotcrete and represents a more rational choice compared to traditional armature since micro-reinforced shotcrete takes up significantly higher loads since the moment of installation compared to cast concrete armature. As an example, when constructing a support construction based on NATM method using solid rock mass, a layer of micro-reinforced shotcrete reinforced with reinforcing mesh is 10-15 cm thick, while for the same working environment cast concrete armature 30-50 cm thick is needed. (see picture 1).

Anchors represent a key bearing element with usage of NATM because they take up 70-80% of radial pressure of the rock mass. Anchors can be divided into passive and active (pre-stressed). In practice, there are different constructive types of anchors with wedge and split pin with expansion head, “Swelex“ anchors, SN anchors, IBO anchors, pre-stressed anchors (anchors with steel rope).

Micro-reinforced shotcrete is another type of support construction included in modern support constructions.

Composite materials

Modern materials consist of thermostable plastics, epoxy and synthetic resin and polymeric fibrous composites. Modern materials also include micro-reinforced shotcrete containing reinforcements in the form of fibers, particles, tiles, sheets or whiskers of different inorganic and organic origin. Composites are divided into metal, ceramic and polymeric composites. Roughly, they can be divided into three basic categories: composites reinforced by particles, composites reinforced by fibers and layered (laminated) composites. The example for composite reinforced by particles is concrete, for composite reinforced by fibers fiberglass, while the example for laminated composite is plywood. (Antić, 2016).

Composite materials are multicomponent multiphase materials whose phases have different physical and mechanical characteristics with clearly visible boundary area and surface between them. (Aleksić, Živković, Uskoković, 2015). The main phases of composites are reinforcement or active filler depending on the composite feature and matrix.

The discontinual phase in composites is the reinforcement and/or functional filler in the form of particles, tiles, whiskers, short and continuous fibers (Aleksić, Živković, Uskoković, 2015). Each type of reinforcement has its own characteristic application, and the most common are fibers due to many advantages, especially in mining according to the needs of modern support constructions.

The characteristics of composites depend on both phases (continual and discontinual), as well as the boundary area itself which has been formed within composites. Boundary area between two phases of composites plays a major role in defining the composite itself. It is used for transfer of mechanical stresses from matrix to reinforcement and vice versa, and also ensures the long-term stability of composites.

The main functions of fibers in composites are:

- to carry mechanical load;
- to provide stiffness, strength, thermal stability and other constructive features;
- to provide electrical conductivity or electrical insulation depending on the type of fiber etc.

The basic functions of the matrix material are:

- to connect fibers into a product of a certain shape and stiffness and to transfer external loads to fibers,
- to isolate the fibers in order to act individually in stopping the spreading of the cracks;
- to provide good surface quality and final or almost final shape of the product;
- to provide surface protection of the fiber from chemical and mechanical damage by wear and tear;
- to provide, depending on the nature of the chosen matrix material, ductility, toughness etc.

That the mechanism of damage and wear and tear primarily depends on the nature of the matrix material and its compatibility with the fibers.

The role of fibers as reinforcement in composites needs to be defined as this type of reinforcement is the most frequent during construction of construction materials. Other types of reinforcements should be mentioned, i.e. (Aleksić, Živković, Uskoković, 2015):

- particles for metal and ceramic matrices, particles for polymeric matrices;
- particles in nanocomposite materials, whiskers, sheets and tiles in composites.

Fibers as reinforcement in composites

Fibers are thin flexible materials whose length is many times greater than their cross cut. From the historical point of view composite fibers have been used since 1500 BC when Egyptians and the population of Mesopotamia were using the mixture of mud (matrix) and straw (fiber reinforcement) to build strong and functional houses. Today the main representative of fiber composite is fiberglass (polymer glass comprised of fiber and glass). This composite is characterized by superior mechanical properties with low density.

The most frequently used reinforcements in composites are fibers because materials reinforced with this type of reinforcement have outstanding physical and mechanical properties. Fibers can be classified according to type, length, diameter, orientation and hybridization. (Aleksić, Živković, Uskoković, 2015).

Fibers are used in composites in various forms, such as: individual fiber (monofilament) of diameter up to 10 μm, fiber bundles (multifilament, yarn, roving) which can contain up to several thousand fibers. Fiber bundles are sometimes combined with additives of resin or adhesive, or on the other hand, multifilament fibers can be translated into various textile forms by weaving, knitting or pressing. (Aleksić, Živković, Uskoković, 2015; S.V. Hoa, 2009).

The usage of fibers as reinforcements in composites is based on: small diameter of fibers, fixed ratio of length and diameter of fibers, if the fibers are discontinual (short) and highly flexible. Small diameter of fibers allows for greater proportion of material strength to be achieved than in larger diameter of the same material, since the defects in the structure of material contributing to the reduction of strength are less likely to occur. (Aleksić, Živković, Uskoković, 2015). According to researcher K.K. Chawla (2013) the tensile strength of carbon fibers with diameter of 8 μm is about 2,5 GP, while for fibers of the same material with diameter of 12 μm the tensile strength is about 1,5 GP.

The optimal ratio of length and fiber diameter allows for successful transfer of external load from matrix to fiber over boundary surface. The fibers must have high module of elasticity, high and uniform tensile strength as well as flexibility during processing into textile fabrics.

Composite materials based on glass and basal fibers

Composite materials have been used in various fields of building construction and mining facilities as a reinforcing agent for a number of years due to many advantages they have in relation to traditional iron and steel reinforcement processes. The widest application of composite reinforcement has been in the field of construction of infrastructural, underground and underwater constructions, facilities that come into contact with aggressive environments (such as underground atmosphere), constructions in which the functioning of magnetic and electromagnetic waves is undisturbed, reinforcement of new facilities and renovation and reconstruction of the existing ones.

Recently the traditional reinforcement mesh enclosed with steel during construction of roads, bridges, tunnels, subways, dams, reinforcement of rock mass, construction of abutment walls, foundations, piles, pools, geotechnical facilities, underground mining facilities has been replaced with modern composite reinforcement mesh.

The reasons for the above-mentioned facts are the superior characteristics of composite reinforcement and they are as follows:

High resistance to corrosion and aggressive environments, resistance to high and low Ph values and chloride ions, increased resistance of bearing walls to seismic activities.

Strength which is considerably higher than steel.

Consistency, estimated exploatation life of concrete is up to 100 years with the strength of reinforcement mesh of 1100MP.

Compared to steel the mass of composite reinforcing mesh is 4 to 5 times lighter, making it cheaper and easier to transport, store and handle the supplies, and it also makes physically demanding hard work during construction easier.

Non-conductivity of radio and magnetic waves, electricity and heat/coldness

The price, considerably cheaper than steel, galvanized and laminated reinforcement. On average, the price is 40% cheaper plus savings on transport and store costs.

Non-toxicity, composite reinforcement is completely non-toxic

Black steel is cheap and there is lots of it. Since it is highly susceptible to corrosion, in most cases, its surface is covered with corrosion even before installation. Corrosion degrades the concrete from the armature to the interior and exterior of the construction which can be noticed in many constructions in the form of stains on the concrete surface (see Picture 3). For that reason, renovation is needed on the surface parts of the facilities in the first 5-10 years, while capital renovation is necessary in the period of 20-40 years depending on the environment. An ideal solution in overcoming this problem lies in application of stainless anti-corrosion composite reinforcement in concrete.

Picture 3. Influence of corrosion of steel reinforcement on concrete

Since composite reinforcement is double the tensile strength of steel (1100 MP) its exploatation life in concrete is very long (up to 100 years), and it is possible to lower the diameter of built-in composite reinforcement in relation to steel. Table 1 shows the compared characteristics of steel and composite reinforcement.

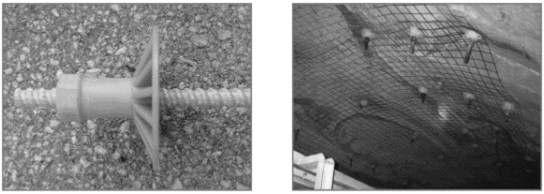
Table 1

Compared characteristics of steel and composite reinforcement

Characteristics of the material	Steel reinforcement	Composite reinforcement
Strength, MPa	500	1100
Module of elasticity, MPa	200000	50000
Deformation, %	25	2,34
Coefficient of linear expansion, $\alpha x \cdot 10^{-5}/°C$	13-15	9-12
Density, t/m^3	7	1,9

Characteristics of the material	Steel reinforcement	Composite reinforcement
Corrosion to aggressive media	corrodes	Stainless material in alkaline environment
Thermal conductivity (Vt/m^2)	56 (thermal conductivity)	0,48 (heat resistant)
Electrical conductivity	Electrical conductivity	Dielectric
Diameter, mm	6-40	4-32
Length, m	Rods 6-12 m long	Rods up to 100 m
Durability	According to rules and regulations	80 years and more

Technical characteristics of anchors made from glass fibers impregnated with polymer (Picture 4) are given in Table 2.



Picture 4. Fiberglass anchors

Table 2

Technical characteristics of fiberglass anchors

Diameter, mm	25	27	32
Cross cut, mm^2	350	400	580
Tensile strength, MPa	650	650	650
Weight, kg/m	0,9	1,04	1,50
Shear strength, MPa	317	317	317
Module of elasticity, MPa	50000	50000	50000
Deformation, %	2,5	2,5	2,5
Strength of steel matrix l=100 mm	180	200	200
Strength of fiber matrix, l=100 mm	124	124	124

Fiber reinforced shotcrete

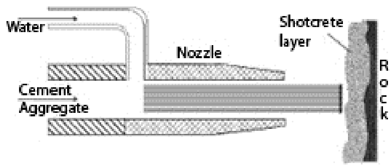
Micro reinforced shotcrete represents composites obtained by reinforcement of cement matrix with evenly dispersed fibers. The installation of micro reinforced shotcrete is performed by the procedure called shotcreting (placing concrete on formwork or a wall).

Micro reinforced shotcrete is mostly used for reinforcing tunnel vaults, rocks and slopes, construction of high quality linings (support constructions) as well as repairs and renovation of facilities.

There are two types of micro reinforced shotcrete - dry mixture and wet mixture i.e. dry and wet method of preparing and installation.

The wet method is a technique where cement, aggregate and water are dosed and mixed before they are poured into a pump and transported through the hose to the nozzle where they pneumatically hit the surface. Compressed air enters the flow of the material in the nozzle so the material can be led towards the substrate. Wet mixture usually contains additives and can also contain micro-fibers.

The dry method is a technique where cement and aggregates are dosed, mixed and delivered into a designated machine, whereby materials are pneumatically transported through hoses or pipes towards the nozzle where water is drawn in so the mixture can get wet and placed on the substrate. This type of shotcrete can also contain additives or fibers or a combination of both. Picture 5 shows the installation of micro reinforced shotcrete by applying the dry method.



Picture 5. Sketch of pouring micro reinforced shotcrete by applying the dry method (Trifunović, Tokalić and Đukanović, 2009)

Fibers used for reinforcement of micro reinforced shotcrete can be divided according to size, type of material and function.

According to size and function we can distinguish:

Micro-fibers (diameter below 0,3 mm) are used to reduce the formation of cracks related to concrete shrinkage: they improve durability, resistance to cold, impact and abrasion: typical dose is 1-3 kg/m^3.

Macro fibers (diameter above 0,3 mm), steel and macro synthetic fibers are considered to be structural fibers as they increase ductility and strength of micro reinforced shotcrete, help prevent cracks in built-in micro reinforced shotcrete; the usual dose of steel fibers is between 25 and 60 kg/m^3, while the usual dose for synthetic fibers is 4,5-9 kg/m^3.

According to the material from which they were made, fibers can be divided into:

metallic fibers (usually steel), the main disadvantage is corrosion, as well as high melting point compared to synthetic fibers, which can contribute to forming of the cracks in micro reinforced shotcrete; steel fibers can be straight, wavy with hook or flat endings;

synthetic fibers - the most frequently used synthetic material is polypropylene, but nylon, aramid, polyester and acrylic fibers can also be found; they are highly resistant to oxides and alkaline environment of concrete, they have less mass and are less resistant to erosion;

fiberglass, can be expensive, it is necessary to use alkaline resistant fibers;

hybrid fibers, recent studies show that steel and polypropylene hybrid fibers have been mechanically improved.

A comparative analysis of steel mesh, steel and polypropylene fibers as reinforcing elements in micro reinforced shotcrete in laboratory conditions has been performed by two authors Cengiz and Turanli (2004). They have carried out an assessment of the characteristics of the samples doing experimental research at panels with steel mesh, steel fibers, polypropylene fibers, as well as a combination of steel and polypropylene fibers. The results of their research was published in the conclusion of their research paper (Cengiz and Turaneli, 2004) and illustrate the positive sides of application of hybrid fibers. Physical and mechanical characteristics of steel fiber reinforced shotcrete are much better than applying only one type of reinforcing fiber.

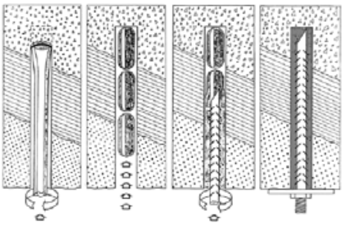
Application of composites in underground construction

Underground mining facilities in „Rudnik“ mine

Underground mining facilities in „Rudnik“ mine, especially those constructed through ore, have to be systematically supported so that they can subsequently collapse in order to form an excavation. For that reason it has been decided that in these underground mining facilities (excavated corridors), as well as in later formed excavations themselves, support construction using fiberglass anchors should be applied combined with two-componenet mixture which will, after blasting the roof in order to form an excavation, enable the uninterrupted operation of the loading machinery (which wasn't the case with application of steel anchors). For the sake of technical and economic justification the testing of the above-mentioned anchors has been performed at a specific location together with monitoring of the parameters of drilling, embedding and the results of the anchoring test. (Mitić et al., 2011).

This type of anchors belongs to a group of anchors that have a bearing capacity over the entire length of the drill hole and function on the principle of connecting the direct lower roof with the healthy upper roof together with a filling of the cracks with mixture. The built-in anchors form a solid beam structure.

The installation of fiberglass anchors has been performed in four phases (see Picture 6): drilling and cleaning of a 1,6 m long drill hole, setting up of cartridges of two-component mixture, installing anchors to the bottom of the drill hole (cutting of cartridges of two-component mixture) using a rotary drill and setting up of a fiberglass base plate at the mouth of a drill hole.



Picture 6. Phases of installation of fiberglass anchors

Anchors are built into the drill holes with a diameter of 34 mm (*d*=34 mm) made with drilling hammer AC BBD-90/91 („Panther") equipped with monobloc chisel l=1600 mm long. Installation into the side and roof of the facilty has been carried out. Two-component mixture used for connecting consisted of two cartridges, one quick setting (to the bottom of the drill hole Lokset resin 28×250 mm HS Fast) and one slow setting (Lokset resin 28×600 mm HS Slow). The total length of resin cartridges for connecting is 850 mm. The process of installing an anchor takes up to 30 seconds. (Mitić et al., 2011).

After the hardening of the resin occurres, a checkup of the tensile force of the nut and fiberglass anchors is performed. The nut breaks where the tensile force amounts to F=86 kN, while the anchor breaks where the tensile force amounts to F=116,5 kN. The anchor is interrupted at the end, outside of the drill hole. Picture 7 shows the interrupted anchor. (Mitić et al., 2011)

Picture 7. The place of anchor interruption (Mitić et al., 2011)

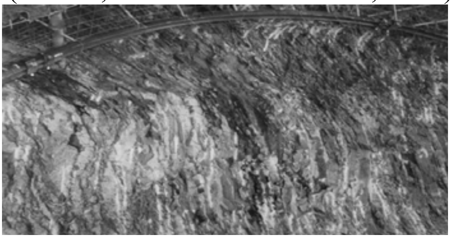
Furthermore, in the Rudnik mining facilities where there have been instances of smaller pieces of rock mass breaking off into the facility, a combination of fiberglass anchors and steel mesh has been used as support construction.

Mitić et al. (2011) have concluded that the application of fiberglass anchors with the security coefficient of 1,5 is applicable in the „Rudnik“ mining facilities where bearing capacity of minimally 77 kN has been calculated. It should be stated that the nominal capacity of anchors is higher, beacuse the anchor itself has not been cut out, but the interruption occurres outside of the drill hole.

Dekani tunnel in Slovenia

During construction of the Dekani tunnel a comparison of two types of support construction was made through various laboratory and field tests. On the one side there was traditional support construction consisting of steel frames, reinforcing mesh and ordinary micro reinforced shotcrete, and on the other side there was micro-reinforced shotcrete with carefully selected length, types of fibers and appropriate accelerators.

Geology in Dekani tunnel zone predominately consists of flysch which is characterized by alternating layers of marl and sand, and the layers themselves vary from couple of centimeters to two meters of thickness. Furthermore, tectonics is highly complex and there are many irregularities in the arrangement of layers and cracks as can be seen in Picture 8. (Jovičić, Šušteršić and Vukelić, 2009).



Picture 8. Flysch texture in the Dekani tunnel zone (the colour grey – marl, the colour brown – sand) (Jovičić, Šušteršić and Vukelić, 2009)

During construction of the tunnel measuring posts have been set for monitoring the vertical and horizontal movement of support elements at certain sections. The task was to check whether the application of micro-reinforced shotcrete of the concrete brand MB C20/25 with steel fibers16 mm long and diameter of 0,4 mm is as effective as traditional way of support construction made from steel arches TH21 (1,5 m distance), steel mesh (diameter of 6 mm with the density of 15 cm) and ordinary micro-reinforced shotcrete of the above mentioned brand. In both cases radially distributed anchors with diameter of 28 mm and 6 m length have been installed with mutual distance between anchors of about 2 m and mutual distance between rows of anchors of 1,5 m.

Jovičić, Šušteršić and Vukelić (2009) have proved that the micro-reinforced shotcrete support construction in the Dekani tunnel is the adequate choice. The reasons for choosing this type of support construction as opposed to a traditional one are easier and safer work operations, lower consuption of support materials and time saving during installation. By performing laboratory tests they have proved that the micro-reinforced shotcrete has shown high strength and ductility after only three days of production. By applying the modern type of support construction during construction of the Dekani tunnel great savings have been acomplished in work force and materials. Furthermore, the progress was 7 m per day compared to the original plan of 5 m per day (tunnel was finished 4 months earlier, due to this). By applying modern type of support construction all work operations were finished without any danger whatsoever to the work force. (Jovičić, Šušteršić and Vukelić, 2009).

Conclusion

On the basis of the previously stated it can be concluded that by using modern, composite materials in support construction positive results can be achieved with regard to shortening the duration of work operations during support constructions, and consequently the whole cycle of construction of facilities. The safety of hired work force is increased, stabilization of contours of underground facilities is faster, work process is less demanding and mechanized which results in reduced number of hired workers and leads to positive economic results. Modern materials are a good choice for support construction in mining conditions which can often be very demanding (aggressive atmosphere, inflow of toxic waters etc.).

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INTERCOMMUNICATION OF CONTENT OF QUALITATIVELY-TECHNOLOGICAL INDEXES OF MINERALS IS WITH DEVELOPMENT OF MOUNTAIN WORKS

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Summary
Object of research. The estimation of mean value of the naturally-spatial placing of changeability of content of quality indexes in the bowels of the earth is based on research of changeability of description of signs, analysis of ore-mining geometrical parameters of blocks, areas of ore body and bed of deposit of ferrous quartzites and network of reconnaissance mining holes. For the regular networks of assay of the naturally-spatial placing of changeability of content of quality indexes of minerals in the array of balance-industrial supplies the calculations of coefficients are conducted in good time for the most tyFigal configurations of mutual location of block and tests that participate in an estimation.

Methodology of researches. Taking into account the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass in un magnetic mineral forms, four technological groups are distinguished ferrous quartzites that differ on content of qualitatively-technological indexes of minerals.

Purpose and research problems. This estimation of annual and perspective plans of development of mountain works taking into account the dynamics of changeability of qualitatively-technological descriptions of signs of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass at moving of mountain works both on reaching and on the depth of quarry. The methods of the automated geometrizing of content of quality indexes of minerals of areas of ore body and bed of deposit of ferrous quartzites are considered.

Keywords: ferrous quartzites, content of qualitatively technological indexes, mountain works.

Introduction

A problem and her connection are with scientific and practical tasks. Within the framework of general conception of management of the prepared products qualitatively technological indexes from positions of approach of the systems construction of the system of surveyor-geological management naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass is based on principles [1]:

- to the management unbreak naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass and by volume of producing of the prepared products;
- to the management complexity on functions, tasks and informative providing;
- to unity of management process on all cycle of forming of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass from planning to the consumption;
- standardizations of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass as to the means of management.

At development of annual and perspective plans of development of mountain works the dynamics of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals is taken into account in an array and loosening iron-ore mass at moving of mountain works on reaching and on the depth of quarry.

Analysis of researches and publications. Establishment of conformities to law of changeability of content of qualitatively-technological indexes of minerals in iron-ore mass and determination of limits of oscillation on the different areas of ore body and bed of iron-ore deposit on a career will allow clearly and reasonably to plan direction of development of mountain works, stabilize content of qualitatively-technological indexes of minerals in iron-ore mass and improve the technical-economical indexes of work of ore mining and processing combine [2].

Rising of task. Coming from the requirements of mountain production and functions of surveyor-geological departments on the content management of qualitatively-technological indexes of minerals in iron-ore mass embrace practically all complex of tasks, have connection with planning and planning of mountain works, directly with the booty of balance-industrial supplies in the process of development of areas of ore body and bed of iron-ore deposit up to shipping of commodity iron-ore mass on an ore mining and processing factory [3].

To planning and calculation of the average systems of content of quality indexes of minerals in iron-ore mass on an ore mining and processing combine research of statistical descriptions of content of qualitatively-technological indexes of minerals in an array and iron-ore mass on the areas of ore body and bed of iron-ore deposit and transformation is preceded in the process of booty of balance-industrial supplies and exception of content of quality indexes useful to the component, related to magnetite [4].

Exposition of material and results. From data of operating assay of blast holes of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array on the career of ПрАТ «South ГЗК» the built graphic arts of oscillation of content of quality indexes of minerals in iron-ore mass on a general and magnetite useful component (Fig. 1) [5].

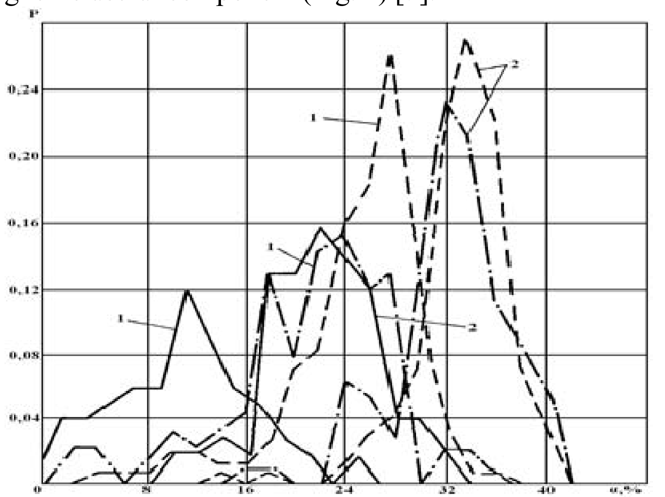

Fig. 1 - Naturally-spatial placing of changeability of content of quality indexes of minerals in the array of balance-industrial supplies on the career of ПрАТ «South ГЗК»: *1* - α_{mg}; 2 - $\alpha_{заг}$; (----) - horizon is minus a 15 m; (-- --) - it is horizon minus a 30 m; (-- · --) - horizon is minus 45 м

It is shown on graphic arts that for to all considered horizons distribution of content of quality indexes of minerals in iron-ore mass is near to the normal law [6].

Statistical data about distribution of content of qualitatively-technological indexes of minerals in iron-ore mass on extractive horizons of quarry testify to the considerable vibrations of content of qualitatively-technological indexes of minerals to iron-ore mass: $\alpha_{заг}$=3,8-4,6 %; σ_{mg}=5,4-6,9 % [6].

For research of changeability of content of qualitatively-technological indexes of minerals at the component quartzites of IV of ferrous horizon of quarry of ПрАТ «South ГЗК» on the chemical and phase analyses of mining holes of operating secret service distribution of different components curves are built.

Analyses are used here equipartition for areas deposits of ferrous quartzite on the quarry. On correlation of minerals, to the chemically-analytical and texture criteria on Skelewativske deposits distinguish three groups of basic varieties of ferrous quartzite: to magnetite, silicat-carbonat-magnetit and to gematit-magnetit. An additional group is presented by the poorly-oxidized and semi oxidized varieties [7].

They got histograms certify the wide range of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass on Skelewativske deposit of ferrous quartzite.

Distribution of content of qualitatively technological indexes useful to the component of general iron and related to magnetite near to the normal law. They change accordingly within the limits of 20-48 %; 10-42 %.

Interval of changeability of content of qualitatively-technological indexes useful to the component, related to magnetite in carbonate and silicate and gemmated far fewer and folds 1-17 %; 1-13 %.

The analysis of distribution of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass on a career shows that on ore mining and processing combines takes place considerable oscillation of content of qualitatively-technological indexes of minerals in iron-ore mass (σ=4,8-15,4) (Fig. 2.) [8].

The necessary condition of effective organization of mountain works in the mode of stabilizing of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass are an analysis and taking into account of changeability of content of the useful component related to magnetite crises-cross and on reaching of ore body.

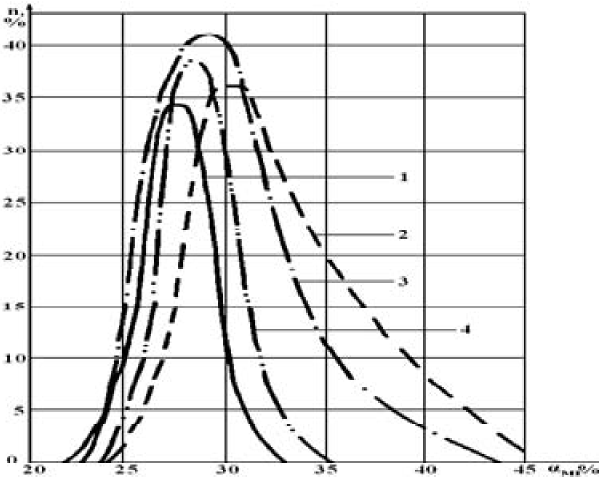

Fig. 2 - Naturally-spatial placing of changeability of content of quality indexes useful to the component, related to magnetite in the mineral varieties of Skelewativske of deposit on the career of ПрАТ «South ГЗК»: *1* is magnetite; *2* is an anhydroferrite; *3* - silicate; 4 - carbonates

Yes, changeability and coefficient of variation on content of qualitatively-technological indexes of the useful component, related to magnetite crises-cross reaching considerably higher than on reaching. For the ground of optimal places and volumes of average of content of qualitatively technological indexes of minerals in iron-ore mass on an ore mining and processing combine determined statistical descriptions and transformation of content of qualitatively technological indexes of minerals in iron-ore mass from one technological stage to other. Commodity iron-ore mass that is shipped from career is characterized [5]:

- by large heterogeneity of content of qualitatively-technological indexes of minerals in commodity iron-ore mass (coefficient of variation of V=5,0 %);
- on an exit from the factories of oscillation goes down approximately twice (V=2,2-4,3 %);
- in the process of growing and magnetic separation shallow there is further substantial reduction of oscillation to content of qualitatively-technological indexes of minerals in iron-ore mass (of V=0,94 %).

Oscillation of quality indexes of concentrate on compositions of ore mining and processing factories (σ=0,45) considerably exceeds norms (0,2 %) what recommended by normative documents [9].

Thus, oscillation of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass has considerable amplitudes that diminish in the process of booty of balance-industrial supplies exception of content of qualitatively-technological indexes of the useful component related to magnetite in 4-6 times.

The vibrations of qualitatively technological indexes of concentrate on compositions considerably exceed norms that need to the metallurgical enterprises that testify to the unsatisfactory process of average nation of content of qualitatively-technological indexes of minerals state in iron-ore mass on ore mining and processing combines. For the terms of working off the quarry of ПрАТ «South ГЗК», at development of annual and perspective plans of development of mountain works the dynamics of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals is taken into account in an array and loosening iron-ore mass at moving of mountain works on reaching and on and on the depth of quarry.

Methodology of establishing a connection is examined between deepening of mountain works and naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass. Establishment of conformities to law of changeability of content of qualitatively-technological indexes of minerals in iron-ore mass with deepening of works on a career, at taking into account of properties and useful to the component, related to magnetite, will allow to define the limits of oscillation of content of qualitatively-technological indexes of minerals on different horizons and in every obtaining coalface.

By a necessary condition for the decision of one of tasks of mountain production - planning and process control of booty of balance-industrial supplies. It allows clearly and reasonably to plan direction of development of mountain works, stabilizes content of qualitatively-technological indexes of minerals in iron-ore mass that is given on the entrance of ore mining and processing factory and to improve the technical-economical indexes of work of ore mining and processing combine. Content of qualitatively-technological indexes of the useful component related to magnetite the varieties of ferrous quartzites of deposit have much general. Magnetite, in the packs of these varieties of minerals, up-diffused evenly, means values hesitate in an interval 28-30 %.

On content of qualitatively-technological indexes of useful component of iron general, the basic varieties of ferrous quartzites differ through different content of unmagnetic mineral forms of bivalent ore triple valent iron (silicats, carbonates and hematite). Correlation of unmagnetic mineral forms testifies that they have characteristic for every variety mean values that interrelate with content of qualitatively-technological indexes of useful component, iron general in dump tails (Fig. 3.) [5].

Analyzing the methods of count [10], what are used in surveyor practice come to the conclusion, that majority from them is oriented to simplification of calculations. By the aim of the use of them at the hand method of calculations of volumes of mountain works and balance-industrial supplies. The question of creation or modernization of existent methods appears for the use in programmatic foods of the computer providing of the surveyor measuring and calculations.

Concentration of production and continuous increase of volumes of booty of balance-industrial supplies and exception of content of qualitatively-technological indexes useful to the component, related to magnetite in iron-ore mass on an ore mining and processing combine, stipulates, necessity of development of new and perfection of existent methods and technical equipments of assay and control of content of qualitatively-technological indexes of minerals for iron-ore mass.

If necessary, hand corrections are entered, or fully repeat a decision with the use of computer technologies. By such method, conduct planning of mountain works taking into accounts the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass.

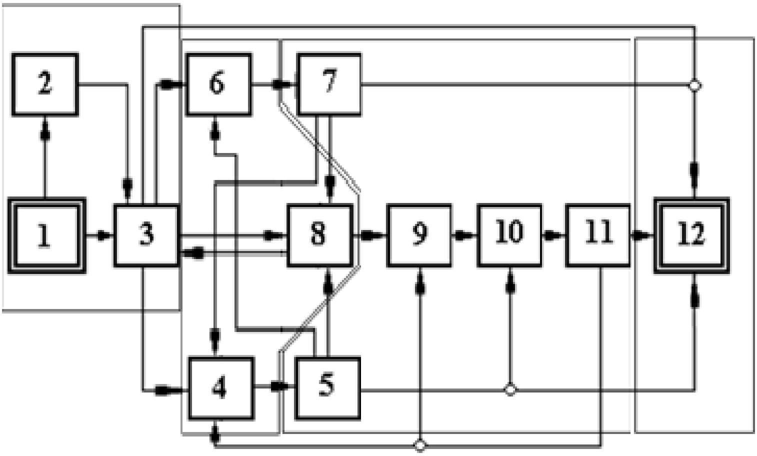

Fig. 3 - Chart of transformation of the naturally-spatial placing of changeability of content of quality indexes of minerals in the array of balance-industrial supplies and loosening iron-ore mass: *1*; *2*; *3* is the naturally-spatial placing of changeability of content of quality indexes of minerals in an array and loosening iron-ore mass, accordingly from the array (natural quality) removed (by loosening drilling and blasting works), in transport capacities; *4*; *6*; *8*; *10* is the naturally-spatial placing of changeability of content of quality indexes of minerals in an array and loosening iron-ore mass accordingly on an entrance at capacity of grading ore mining and processing and sintering factories; *5*; *7*; *9*; *11* is the naturally-spatial placing of changeability of content of quality indexes of minerals in an array and loosening iron-ore mass accordingly on an exit from capacities, grading and sintering factories; *12* - content of quality indexes useful to the component in the prepared products (concentrate, agglomerate, pellets)

The perspective and operative planning assists the rational booty of high-quality and balanced on maintenance qualitatively-technological indexes minerals of supplies in a career and averegenation of content of qualitatively-technological indexes of minerals in loosening iron-ore mass for the separate intervals of time.

Thus, coming from content of qualitatively-technological indexes of useful component, iron general in unmagnetic mineral forms; four technological groups of ferrous quartzites are distinguished. They differ on naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass. An estimation of varieties of minerals is in iron-ore mass on content of qualitatively-technological indexes of the useful component related to magnetite and is acquired by a value at the annual and perspective planning of correlation of varieties of minerals in iron-ore mass. From commodity iron-ore mass content of qualitatively-technological indexes of the useful component related to magnetite and power of ore mining and processing factory is determined is withdrawn.

This estimation is executed at Sunday-day's planning of average nation of content of qualitatively-technological indexes of minerals in iron-ore mass. Aim of estimation of receipt at withdrawn on the ore mining and processing factory of the content of qualitatively-technological indexes of the useful component related to magnetite in dump tails with the possible interval of vibrations set by a plan.

In this connection, there is a necessity, research of intercommunication of geological properties of useful minerals in iron-ore mass with direction of development and deepening of mountain works on careers.

2. Methods of the automated geometrizing of the naturally-spatial placing of changeability of content of qualitatively-technological minerals of areas of ore body and bed of deposit of ferrous quartzites

Without regard to intensive development of mathematical methods of analytical and digital design of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in the array of areas of ore body and bed of iron-ore deposit, wide use of computer technologies, presence serially published of graphic recording devices, the problem of automation of construction of ore-mining geometrical graphic arts is yet distant to the successful decision.

It is conditioned by a large volume and variety of graphic documents that is used for the decision of surveyor-geological tasks on the different stages of mastering of areas of ore body and bed of iron-ore deposit. Also complete developments of the mathematical providing of modern devices are absent, that is oriented to the use in the conditions of ore-mining enterprises.

Experience of the executed developments shows that the most successful application of facilities of computer technique and technology is for the decision of tasks ore-mining geometrical maybe at the use of ideas and methods of the classic geometrizing, formalization of basic heuristic procedures and creation of the standard programmatic modules. Exactly these principles are fixed in basis of methods of the automated geometrizing that is worked out in the Kryvyi Rih national university [11].

The structure of the mathematical providing is determined by the analysis of logical, calculable and graphic procedures that is used for geometrizing of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in the array of areas of ore body and bed of iron-ore deposit by traditional methods. Economic feasibilities of modern graphic -builders are taken into account here.

The existent methods of geometrizing are based on an idea about placing of indexes in the balance-industrial supplies of areas of ore body and bed of iron-ore deposit in the bowels of the earth as geochemical field. The geochemical field is described by the function of coordinates of point of space of $P=f(X, Y, Z)$, or $P=f(X, Y, Z, t)$ depending on that it is an index - structural or quality. Coming from supposition, that the function of P satisfies to the terms of completeness, unambiguity, continuity and smoothness, in geometry of bowels of the earth the method of isoclines' as one is worked out of basic methods of image of surfaces of topographical order naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals is in the array of areas of ore body.

Depending on the degree of studied of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals the method of isoclines' is used in the array of areas of ore body and bed of iron-ore deposit, form of presentation of weekend of data (regular, irregular, continuous assay), quantitative descriptions of changeability of index.

The method of isoclines' will be realized to one of five methods: invariant lines and devil-fishes, polyhedron, profiles, statistical and indirect. A logic analysis with the aim of formalization allowed to distinguish the calculable, graphic and heuristic following of the automated geometrizing [12]:

- a ore-mining geometrical analysis of weekend of data and choice of method of graphic design or combination of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals are in the array of areas of ore body;

- transformation of weekend of data is on the induced network;
- construction of isoclines' of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array at the linear approaching with the next smoothing;
- registration graphic of the document.

Because of an ore-mining geometrical analysis of geological data general conformities to law of placing of index are set in the balance-industrial supplies of areas of ore body and bed of iron-ore deposit in the bowels (Fig. 4.) of the earth [5].

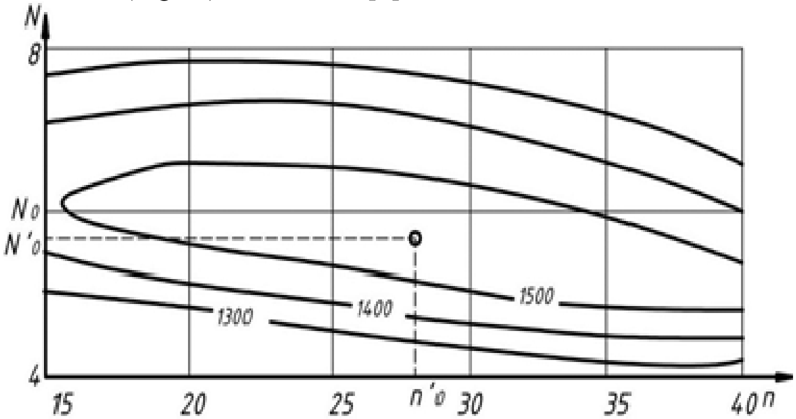

Fig. 4 - A surface $\Delta T=F$ (N, n, m=15) is in the isoclines' of values of commodity products in a money equivalent. Ferrous quartzites are on the career of ПрАТ «Inhuletskyi ГЗКа»

On this stage invariant lines and devil-fishes of the represented surface are set, the borders of areas of geometrizing are determined. Exactly this stage is difficult formalized for the use of computer technologies.

The results of study of changeability of index give additional data for a ore-mining geometrical analysis. If invariant lines are near to the lines, then at research of changeability main directions of anisotropy approximately gather or perpendicular to the invariant lines.

Quantitative estimations of parameters of changeability are the heights of cut of isoclines' used for determination. The choice of method of construction of isoclines' is determined by the results of ore-mining geometrical analysis and researches changeabilities.

The methods of polygons, invariant lines and devil-fishes are used for the construction of structural plans at an irregular reconnaissance network. So as qualitatively-technological indexes contain considerable part of casual constituent, then for the reflection of properties the method of statistical window is used. Window that smoothes out extent, certainly coming from the height of cut of isoclines'. Transformation of weekend of data on a regular rectangular or square network comes true by means of analytical models of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in the array of areas of ore body and bed of iron-ore deposit by smoothing methods, including the methods of optimal statistical interpolations.

Regardless of that is used the network of data (three-cornered or rectangular), determination of coordinates of isoclines on the first stage comes true by means of linear interpolation on the «ribs» of network on two adherent knots. After it the broken isoclines' are smoothed out. Nonlinear approximation of the represented surface of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals comes true in an array.

The enumerated operations at the graphic design of areas of ore body and bed of iron-ore deposit are formalized, except for the row of heuristic procedures. For the selection of invariant lines and devil-fishes and for triangulation of reconnaissance networks the additional surveyor-geological is used information.

The analysis of logical connections between the graphic and heuristic procedures calculated, determines principle of construction of the mathematical providing of tasks of graphic design of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in the array of areas of ore body and bed of iron-ore deposit.

The mathematical providing of the automated geometrizing contains the programs of three levels: base, functional and applied. Of the mathematical providing at level answers the degree of working out in detail of the shown out graphic information [13].

The base programs worked out by the plant-manufacturer of graphic buildings allow carrying out the construction of the simplest geometrical elements (alphabetical and digital symbols, segments of lines, arc of circumference, polylines and so on.).

The functional programs are developed taking into account content of the applied tasks. In the complement of the functional mathematical providing of tasks of graphic design of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in the array of areas of ore body and bed of iron-ore deposit enter [14]:

- a construction of coordinate scales (continuous, dotted, stroke of dotted) is with coordinates on the perimeter of net;
- a construction of plans of assay of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals is in the array of areas of ore body (with causing of mining holes, names, content of qualitatively-technological indexes of minerals in an array and so on.);
- construction of basic elements of plans of mountain works; twisting of coordinates (affine, functional and other);
- processing of graphic documents (causing of scopes, accompanying text and so on);
- draft of charts of functions, that is set analytically in an obvious kind and self-reactance;
- draft of charts of functions, that is set tabular, with the use of different methods of interpolation.

Application programs of graphic design of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array, areas of ore body and bed of iron-ore deposit provide plotting of isoclines' different methods [15]:

- geological cuts with the isoclines' of quality indexes;
- plans of mountain works;
- block of diagrams;
- models of mining-and-geological objects in axonometric projections and so on;
- by volume images of surfaces of topographical order.

The methods of graphic design of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in the array of areas of ore body and bed of iron-ore deposit contain identical procedures of interpolation (twisting of coordinates, smoothing of isoclines', subscription of isoclines' and so on.). The most rational principle of packet of the application programs assembly is module.

The algorithm of construction of mathematical model of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in the array of areas of ore body and bed of iron-ore deposit consists of chain of procedures, each of that appears an independent task. The decision of separate from them differs in a substantial mathematical novelty.

To basic procedures belong:

- construction of border of zone of mineralization; choice of model parameters;
- smoothing of results of primary assay of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals is in the array of areas of ore body;
- Kriging histograms;
- realization of switch control statement is to the new system of assay of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array.

The linear sizes of blocks excel AV distance between tests in a few times, and the volumes of blocks are far less to the volume of areas of ore body and bed of iron-ore deposit, in that they are, then an estimation can be simplified [16].

Number of tests that get n equals in the estimated block of volume υ, and the number of all tests within the limits of the field of V equals N. Then as an estimation of integral expression is used

$$\tilde{C} = \lambda C_n + (1-\lambda)\overline{C}_N \tag{1}$$

where $\overline{C}_n = \frac{1}{n}\sum_{k=1}^{n} C_k$ is a middle arithmetic estimation of index on tests in the middle of block υ; $\overline{C}_N = \frac{1}{N} = \sum_{m=1}^{N} C_m$ it is a middle arithmetic estimation of index on the tests of all field of V.

Assuming that tests within the limits of block υ are located by chance and a block is located by chance in the field of V (hypothesis of casual Kriging), then expressions for λ and dispersions will have a comfortable for calculations view [17]

$$\lambda = \frac{\sigma_\nu^2}{\sigma_n^2 + \frac{n}{N-n}\sigma_\nu^2};$$

$$\sigma_k^2 = (\sigma_n^2 - \sigma_v^2)\left[1 - \frac{\sigma_n^2 - \sigma_v^2}{\frac{N}{N-n}\left(\sigma_n^2 + \frac{n}{N-n}\right)\sigma_v^2}\right], \tag{2}$$

where $\sigma_n^2 = \frac{1}{n^2}\sum_{k=1}^{n}\sum_{m=1}^{n} K\left(\left|\bar{r}_k - \bar{r}_m\right|\right)$, if the number of tests of N on all field of V considerably excels the number of tests of n in a middle a block υ, that is estimated, then close formulas are used

$$\lambda \approx \frac{\sigma_\upsilon^2}{\sigma_n^2}; \qquad \sigma_k^2 = \left(1 - \frac{\sigma_\upsilon^2}{\sigma_n^2}\right)\sigma_\upsilon^2 \tag{3}$$

The parameters of estimation and dispersion of Kriging block υ depend on the number of tests, and also from descriptions of autocorrelation or structural naturally-spatial hashing of changeability of content of qualitatively-technological indexes of minerals functions in the array of areas of ore body.

Thus, at the estimation of large and small blocks bulky equalizations of Kriging are considerably simplified. Calculations are conducted quickly in default of appropriate constituent in placing of sign of *C* (*X*, *Y*, *Z*, *t*). At presence of trends', all considered approaching become incompetent. In this case it is necessary to distinguish a trend, using a least-squares method [18].

Coefficients of equalization of trends', conditioned on it method not displaced, however they minimize dispersion, if deviations of values of sign from trends' are auto correlated. At determination of trends' a rejection is investigated on autocorrelation.

At presence of estimation of coefficients of autocorrelation that is calculated on differences will be displaced. Geostatistical calculations are for deviations from trends' that is conditioned by a least-squares method, appear ineffective. The autocorrelation of rejections is taken into account at the selection of equalizations of trends. There is not a necessity in such account, if number of points of assay more than 100. In opposite case it costs to modify approach to the problem of estimation that results in equalizations of craining [19].

At the construction of mathematical model of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in the array of areas of ore body and bed of iron-ore deposit there is the field of concentrations of the useful component, related to magnetite occupies the local volume of bowels of the earth, limited to the that area of space, the process of mineralogenesis took place in that. Objectives are the limits of the mineralized zone, built from data of geological supervisions and by the results of assay of geological survey mining holes.

Smoothing of results of primary assay comes true by middle weight of averagination of data of assay of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array on the segments of mining holes between the surfaces of separate ledges [20].

Successful increase of rates of booty of balance-industrial supplies from the areas of ore body and bed of iron-ore deposit by open, underground or combined methods, the use of the newest measuring devices, high-performance equipment and computer providing on the ore-mining enterprises of Ukraine begins to dictate certain requirements and to change out-of-date methodologies. Availability and development to the proper level of the computer providing it is allowed to use them in a sufficient amount by every worker in a surveyor department. It gave possibility to automatize workaday tasks that is executed by the workers of surveyor department at cameral treatment of the field measuring. Among main problems that must be decided there is a question of count and account of volumes of mountain works balance-industrial supplies subject to condition the use of the geographic information systems, namely, what methods of count to use for providing of necessary exactness, comfort of the use and reduction to time on implementation of works.

Analyzing the methods of count [21], what are used in surveyor practice come to the conclusion, that majority from them is oriented to simplification of calculations. By the aim of the use of them at the hand method of calculations of volumes of mountain works and balance-industrial supplies. The question of creation or modernization of existent methods appears for the use in programmatic foods of the computer providing of the surveyor measuring and calculations.

Thus, it is possible to assert on the basis of the stated, that at application of computer technique and technology for the decision of mining-geometrical tasks, high results it maybe to get using ideas and methods of the classic geometrizing, formalization of basic heuristic procedures and creation of the standard programmatic modules.

At the construction of mathematical model of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in the array of areas of ore body and bed of iron-ore deposit there is the field of concentrations of the useful component, related to magnetite occupies the local volume of bowels of the earth, limited to the that area of space, the process of mineralogenesis took place in that [22].

The model of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals built after this algorithm in the array of areas of ore body and bed of iron-ore deposit is used for the count of balance-industrial supplies, optimal planning, and corporate strategic and medium-term planning of mountain works on operating enterprise.

Conclusion

1. Oscillation of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass has considerable amplitudes that diminish in the process of booty of balance-industrial supplies exception of content of qualitatively-technological indexes of the useful component related to magnetite in 4-6 times.

2. The model of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals built after this algorithm in the array of areas of ore body and bed of iron-ore deposit is used for the count of balance and industrial supplies, optimal planning of mining enterprise, and also corporate strategic and medium-term planning of mountain works on an operating enterprise.

3. This estimation of annual and perspective plans of development of mountain works taking into account the dynamics of changeability of qualitatively-technological descriptions of signs of the naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass at moving of mountain works both on reaching and on the depth of quarry. The methods of the automated geometrizing of content of quality indexes of minerals of areas of ore body and bed of deposit of ferrous quartzites are considered.

4. Automated control system of management is on a career, allows to do the optimal forming and management naturally-spatial placing of changeability of content of qualitatively-technological indexes of minerals in an array and loosening iron-ore mass on all stages of planning of mountain production and his management in a career.

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MODELING OF GEOMECHANICAL PROCESSES IN THE CONDITIONS OF UNDERGROUND MINING OF ORE DEPOSITS

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I. Simulation of tectonic stress fields of gold deposits

Article is devoted to questions of forecasting of stress of rocks massif by underground mining of Zarmitan gold deposit zones. It is proved that as a result of maintaining and deepening of mining operations probabilities of emergence of dynamic phenomena grow in rock massif. By mathematical modeling and a mine experiment sizes of horizontal squeezing tectonic tension, the exceeding vertical tension more than 1,4 times are determined that predetermines formation of stress of rock massif under influence of influence of mining developments. The offered analytical method of definition of main vector of tectonic tension and final and element model of the intense deformed state can be used at further to form a basis of forecasting of probability of emergence of mine and tectonic bump and design planning and conducting mining operations in modern conditions of development the difficult structurally of Zarmitan gold deposit zones at big depths.

The mining of gold deposits in the Republic of Uzbekistan have 500 m. There is a tendency to further increasing of development depth in this field. The rock pressure in a dynamic form is at great depths mining operations. There is the probability of rock bursts.

Currently, operating mines, mining operations conducted by all the negative consequences and some data of geodynamic and geomechanical state of the rock mass. In these circumstances, forecasting the stress-strain state of the rock massif in the seismotectonic active zones taking into account the structural and geomechanical conditions are actual for theory and practice.

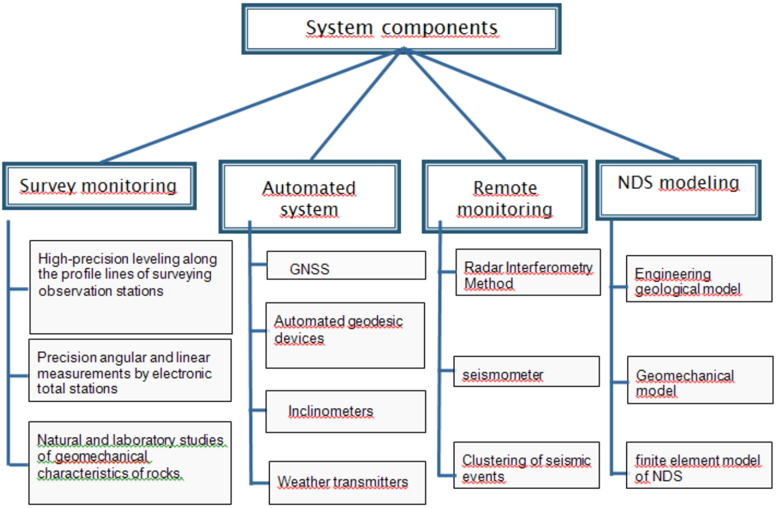

Figure 1. The concept of state surveying monitoring of rock mass a based on modeling of geomechanical processes and high-precision surveying instrumental observations with use of modern electronic equipment, laser and satellite technology position

In order to solve these problems there was produced simulated stress- strain state of rocks in different ways.

The adequacy of the developed model have to be the level of methods development of solving the problems. It must comply with certain requirements and accuracy.In the first stage, it was create engineering-geological model on the basic of initial information.

Engineering - geological model created on the basic of the geological analysis, physical, mechanical data and schematization and investigated rock mass of region.

Schematization is a mere aggregation of rocks based on the physical and mechanical properties and characteristics taking into account externalities. It is mean that it is necessary development and subsequent analysis of partial models, which there are some lithology, structural features, the initial stress state, exogenous changes. In many cases, this model creates conditions for forecasting in mountain ranges - avalanches, landslides, unloading, and other weathering. Close connection between models will allow qualitative assessment the phenomena and processes occurring in the investigated rock mass under the influence of mining.

A priori, if they based on the relevant engineering-geological classification of rocks, that it can geometric basis for zoning of rock mass with the release of the individual blocks. State of stress and rock properties may be as homogeneous.

For example, Zarmitan gold ore zone is composed of igneous rocks caused by lateral bending of the Paleozoic strata of the South. Result is formed Charmitan field.

This region is a distinct block structure of the rocks occupying the entire central part of Koshrabod. It has formation of Proterozoic and Paleozoic.

The ore bodies are mainly represented by thin quartz veins and linear zones of mineralized gold in mineralized albitized and chloritizedgranosyenites on the deposit area.

It is localized bending of the bulk of the ore, forming a tubular sub-vertical ore deposit with multiple apophyses [1].

An array of rocks Zarmitan gold ore zone has layered with a predominantly longitudinal violations northeast strike, which inherits the orientation of the open position of deep faults. Direction is the north-west of Koshrabod intrusion. Widely developed a break with the steep angles of incidence that share deposits in the area of the complex structure of separate blocks with a fracture.

There are three types of areas of tectonic origin in the degree of fracture in the field: weak, middleand strong.

Moreover, strong fracture rocks are in the zone of tectonic faults.

The cracks in the rocks have cleavage, because of tectonic compression forces, which is 45° to the axis of the fold structures.

Petrographic variety of rocks and ore deposits grouped into five geotechnical types, which there are differ from one anotherin: strength properties, the size of the elementary building block of the lattice geometry and characteristics of fractures.

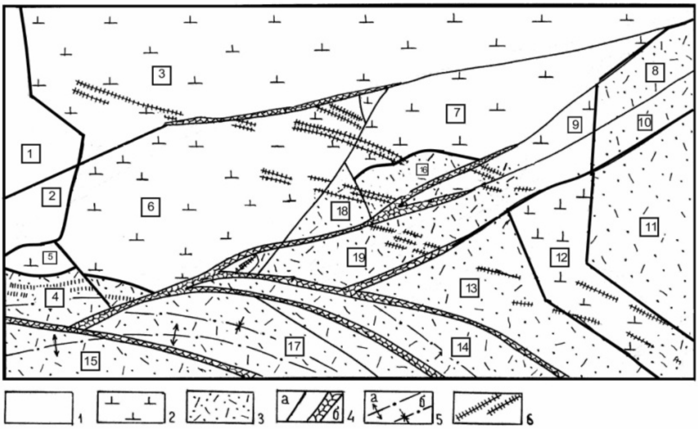

Figure 2. Engineering and geological model of the rock mass field Charmitan:
1 - Quaternary deposits; *2* - Koshrabad complex (C_2): granosyenites;
3 - dzhazbulakskaya suite (S1): sandstone, siltstone, shale, tuff; *4* - faults (*a*), crushing (*b*); *5* - fold axes: *a*- anticlinal, synclinal - *b*; *6* - gold-bearing quartz veins

As a result, engineering and geological modeling allocated 19 geotechnical units, each of which quantified the parameters of orientation of cracks and blocking of various orders, the depth of the zone of disintegration of rocks and groundwater level (Figure 2).

There was developed geomechanical model of a particular mining and technological situation, underground construction or mining, taking into account the need to assess and solved problems of geomechanics based on the engineering- geological model.Constructing such a model, preceded by selection of types and kinds of different structure, taking into account the geomechanical state of these objects [2].

The transition from engineering-geological geomechanical model to include a compilation of raw materials reflecting the particular stress-strain state and behavior under load of individual sections of the array within the specific elements of the systems development - excavation, pillars, etc.

The whole complex of baseline data on the state of stress, the properties and structural features of the rock mass, supplemented by data on the parameters of mine workings and other elements of development systems needed to achieve the objectives in relation to the object of the main content of the geomechanical model.

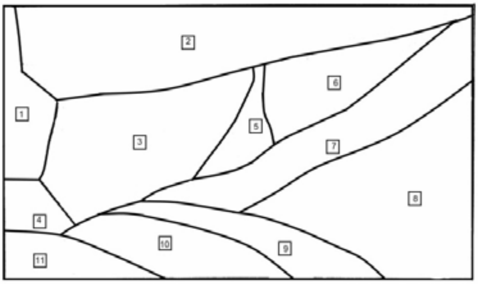

Figure 3. geomechanical model Charmitan gold deposit: - structural elements of the model and their numbers

The model was created by the generalization of engineering- geological model and the geometry of the rock mass.

The next step, examine the conditions likely to bump hazard Zarmitan gold ore zone are full-scale experimental work by a slit discharge.

The essence of the method of discharge slit is in the formation of cracks (Fig.3) radius of 0,3 meters and measuring deformation of its walls to crack. In this case, a zone of unloading rock mass is three dimensions of the gap, which it is significantly reduced the impact of different mountain breeds in this zone [3,4].

The magnitude of the stress (MPa) acting perpendicular to the plane of the gap determined by the formula.

$$\sigma_{\perp} = U_{AB} E\pi / \left[8R - \pi l \left(1 - K_{II(\perp)} + \mu K_{II(\perp)} \right) \right], \qquad (1)$$

where U_{AB} - deformation of the array between points A and B after education gap, cm; E - modulus of elasticity of rock, MPa; R - the radius of the gap, cm; l - distance between points A and B, cm; $K_{II(\perp)}, K_{II(\perp)}$ - stress concentration $\sigma_{\perp}$respectively in the directions perpendicular and parallel slits which are determined graphically (Fig. 3,4).

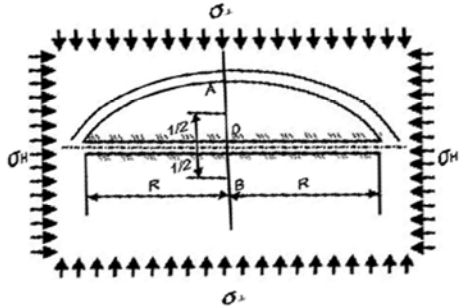

Figure 4. The scheme of determining stress by unloading slit

Application of the method of discharge slit provides information about the voltage in the area of about 1 m. To obtain this information on a large plot, it is necessary to increase the number of measurements by the slit unloading. Especially when the rock massif is intense fracturing and heterogeneity.

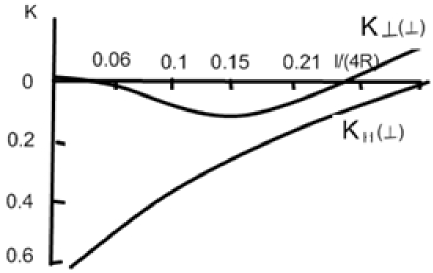

Figure 5. Dependence of stress concentration factors of marginal parts of the discharge gap (K) on the ratio of the distance between the reference points to the radius of the gap e/(4R) under the action of the load, perpendicular to the plane

According to the above procedures have been carried out field measurements at the mine in the next field; Zarmitan drift (720 m), eastern mountains(780), at an angle and vehicle tilt(660 m).

The total amount of discharge slots is 26 (Table 1). According to the results of measurements were determined the stresses in the rock mass, acting vertically σ_v along the strike of the ore bodies σ_{pr} and across the strike of deposits σ_p.

Table 1

Place of measurement stresses at the mine Zarmitan	Kind of stress	theoretical Voltage by A.Geymu $\sigma_v=\sigma_{pr}$ $\sigma_p=\gamma H$, MPa	Tectonic stress according tositu measurements		
			middle Values of voltage σ_{Mid}, MPa	the results of determination stress method slotted discharge σi MPa	relationship stress $\sigma_г/\sigma_в$
660 m	Vertical	6,9	7,2	5,1, 6,1, 6,8, 7,3, 8,5, 8,3, 5,9, 7,5, 8.5	1,4
	Horizontal		9,7	6,7, 8,7, 9,1, 10,5, 10,3, 9,5, 9,6, 10,1, 13,1	
720 m	Vertical	6,2	6,1	4,2, 5,6, 5,6, 5.7, 6,6, 6,9, 7,0, 7,1, 8,0	1,29
	Horizontal		7,9	5.7, 6.9, 6.8, 7.7, 8.3, 7.9, 8.5, 9.1, 9,9	
780 m	Vertical	5,9	5,6	4,1, 4,8, 5,1, 6,5, 5,6, 6,3, 5,6, 6,7	1,27
	Horizontal		7,1	5,0, 6,3, 6,7, 6,9, 7,9, 8,5, 6,2, 9,2	

The confidence interval determining average values of stresses of 12,7-40.3% of their absolute values, witch corresponds to the to the limiting accuracy of measurements of rock stress modern methods. Therefore, the measured values of the initial stress can be used to assess the conditions of bump hazard fields. Also, it can be determined the parameters of the structural elements of underground structures and then clarify their values by field measurements.

Analysis of the determination of the initial stress of rock mass allows us to state that measurement stations are uneven distribution of stresses in the array, which is caused by a complex tectonic structure and modularity rocks. It is limited joint systems and disorders, as well as the mountainous terrain. The actual vertical stress of rock mass is almost equal to the gravitational stress on the weight of overlying rocks equal γH on the game, where γ - volume weight of rocks, MH/ m^3, H - depth development and maximum values are the stresses acting across the strike of the ore deposits, σ_{pr}, which 1.4 times higher than the vertical stress.

Exceeding the horizontal stress σ_p of vertical σ_v explain the presence significant stresses of tectonic origin, which is supported by modern neotectonic crustal movements [5].

There were made stress-strain state (SSS) circuit workings intact array and wall development, which were based on the data mining experiments and finite element modeling (FEM) calculations.

Estimation of host rocks is considered for example of №19:the capacity of which is 1.67meters, angle of incidence is 80°, the maximum depth is 450 m, development system with ore, floor height is 60 m, length of the block along the strike is of 60 m, the thickness of the ceiling pillar is 5 m, and the inter block- pillars is 6 meters.

Table 2 shows the measurement results of measurements and shaft, where were used maximum values of the measured voltages for calculation. Mechanical properties of the surrounding rocks is : P=2520 kg / m^3, E=$1,888\cdot10^4$ MPa, ν=0.23 , σ_s=34,78 MPa, σ_r=3.25 MPa. Parameters section of the workings, which were measured: the height of the wall is 2.62 m, height of the arch - 2,18 m, width - 4.6 meters, the height of -1.31 m.

Table 2

Measurement parameters

Object	Horizon, m	Depth, m	Axis direction of development, degree	Measuring point in the plane (x,z)		σ_v, MPa	σ_h,MPa
				X, м	*Z*, м		
East drift	780	140	160	1347	-666	5,6	7,1
Field drift	720	200	163,5	-15	555	6,1	7,9
Inclined transport Congress	660	260	75	-47,5	387	7,2	9,7

To investigate Stress- strain stateit is necessary preliminary study VAT array. The relationship between the voltage at the point of elastic half-space under the simultaneous action of tectonic forces and gravity derived from the following formula. Consider an arbitrary point in the horizontal plane (*x* and *z*) of the array at a depth of H, where the vertical axis «y» acts voltage is

$$\sigma_y = Pgy. \tag{2}$$

In the plane (x,z) relative deformations determined by Hooke's law [5]

In the case of combined action intact array gravity and tectonic forces directed along the axis X and does not change with depth, the principal horizontal stresses may be represented [5]

$$\begin{aligned} \sigma_x &= \sigma_T + \chi \cdot \sigma_y, \\ \sigma_z &= \nu \cdot \sigma_T + \chi \cdot \sigma_y. \end{aligned} \tag{3}$$

On the wall of conventional generation , free from the normal (in the direction of the axis Z) and shear stresses, the deformation conditions for horizontal section have ratio: $\varepsilon_x = \lambda\varepsilon_t$, $\sigma_z = 0$, $\sigma_t = E\varepsilon_t$.

If $y=0$: $\sigma_y=0$, $\sigma_x=\sigma_t$, then $\lambda=1$ in the general case of variation

$$\sigma_{xv} = \sigma_{tv} + \nu\sigma_y \tag{4}$$

Where «v» is the component stresses in the direction of the axis of development.

Thus, the formula (4) allows us to identify the tectonic stress components on the axle generation (σ_{tv}) the results of measurements (σ_{xv}) in the development.

In the Table 3 shows components of tectonic stress acting in the direction of axes workings.

Table 3

The components of the stress axis workings MPa

Object	Depth, m	Axis direction of development, degree	σ_{xv}, MPa	Calculated values		
				σ_y, MPa	$\nu\sigma_y$, MPa	σ_{tv}, MPa
East drift	140	160	7,1	3,46	0,8	6,3
Field drift	200	163,5	7,9	4,94	1,14	6,8
Inclined transport Congress	260	75	9,7	6,43	1,48	8,2

Results table 3. Shows that tectonic stresses have stability.

To determine the components of tectonic stress on arbitrary axis will considered the plane of the array untouched taking into account the tectonic forces, Figure 6.

The main stress of such a model is defined for σ_y=0: σ_1=σ_t, σ_2=$v\sigma_t$. According to this, for any direction n, we have [5]

$$\sigma_n = \frac{1}{2}(\sigma_1 + \sigma_2) + \frac{1}{2}(\sigma_1 - \sigma_2)\cos 2\theta, \tag{5}$$

θ - where the angle between the directions n and σ_1.

For m,n and β, we obtain a voltage axes

$$\begin{aligned} \sigma_n &= \left(\frac{1+\nu}{2} + \frac{1-\nu}{2}\cos 2\theta\right)\sigma_t, \\ \sigma_m &= \left(\frac{1+\nu}{2} + \frac{1-\nu}{2}\cos 2(\beta+\theta)\right)\sigma_t. \end{aligned} \tag{6}$$

Designating

$$\delta=\sigma_m/\sigma_n, \tag{7}$$

and dividing the right side of the equation (6) we will have

$$\delta = \frac{1+\nu+(1-\nu)\cos 2(\beta+\theta)}{1+\nu+(1-\nu)\cos 2\theta}.$$

Further, this expression is consistently transformed and it allows us to obtain a formula for the determination of angle θ

$$\begin{aligned} A &= \frac{1-\nu}{1+\nu}\sqrt{1-2\delta\cdot\cos 2\beta+\delta^2}, \\ \varphi &= \arccos\frac{\sin 2\beta}{\sqrt{1-2\delta\cdot\cos 2\beta+\delta^2}}, \\ \theta &= \frac{1}{2}\left(\arcsin\frac{1-\delta}{A} - \varphi\right). \end{aligned} \tag{8}$$

A,φ are conversion factors.

From (7) we obtain the formula for determining the value of the main tectonic stress σ_t(compression)

$$\sigma_t = \frac{2\sigma_n}{1+\nu+(1-\nu)\cos 2\theta}. \tag{9}$$

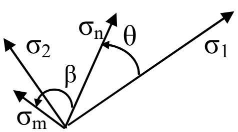

Figure 6. The direction of the vector of tectonic stress fields

Thus, the formulas (7-9) shows the relationship between the main vector tectonic stress (σ_t) and its components along two arbitrary directions (n, m).

The calculation results σ_t formulas (7-9) are shown in Table 4, and the m-axis is directed to the field on the roadway, and the axis of in Inclined transport Congress.

Table 4

σ_n, MPa	σ_m, MPa	δ	β°	N	A	φ°	θ°	σ_1, MPa	σ_2, MPa
8,2	6,8	0,829	88,5	0,23	1,145	88,36	-39,89	12	2,76

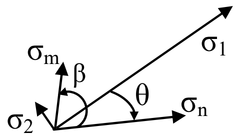

Figure 7. Directions of vectors of tectonic stress fields

Tectonic stress field is determined by the three right-hand column of Table. 4 and Fig. 7, a characteristic that the main vector of tectonic stress crosses towards the production of the Inclined transport Congress 320°.

Construction of models and numerical analysis of the VAT in an array of finite element method (FEM) was performed using the software package Solid works- Cosmos [6]. The solution found by using the finite element method must satisfy the boundary conditions for acceptance to study the array. As a rule, the boundary conditions near the studied object in the array (for example, excavation) are complex and unknown ones. The most simple expression of these conditions become true when the block model is chosen outside the area of influence of the recess. Driving directions of the axes of developments in the plan allows to simplify the task of boundary conditions of the block shown in Fig. 8.

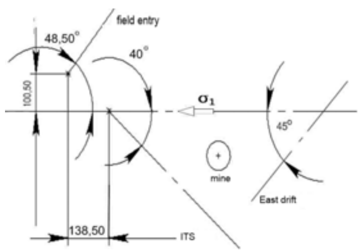

Figure 8. Scheme of stress axes in terms of the workings and the main vector of tectonic directions σ_1

Distance from the axis of the shaft to the axles workings; East - sht. 165.1 m Field sht. - 527, 9 - m and 146.0 m VTC.

The joint action of untouched array of gravity and tectonic stress fields, the main horizontal stresses can be represented as formula (4), which gives grounds to assert the values of variability stresses (σ_1,σ_2), without changing their direction (Figure 8). Table 5 shows the values of the principal stresses under the action of gravity and tectonic stress fields which will then be used as boundary conditions for the simulation models.

Table 5

The main stress of the array untouched by the action of the forces of tectonic stress

σ_{t1}, MPa	σ_{t2}, MPa	σ_y, Pa	χ	σ_1, Pa	σ_2, Pa
12	2.76	25200Y	0.2987	12000000+7527. 24 Y	2760000+7527. 24 Y

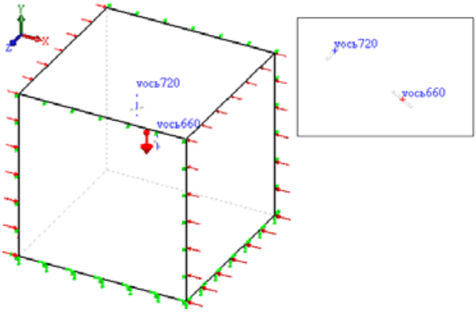

Figure 9. Finite element model for stress analysis

Dimensions of the finite element model adopted in accordance with the above confirmed conditions, with the height of the model was assumed large enough to reduce the effect of fixing it in the underlying layer of tension in the zone of excavation . The geometric parameters of the model: height - 700 meters, the size of the *X*-axis unit - 700 m, on the *Y* axis - 700 m; finite element mesh parameters : the element size - 40 meters , the size of the condensation - 1 m, *a*, *b* - parameters of thickening grid software and computer *a*/*b* =3, the number of layers - 5.

Since the circuit is free from the generation load, the axial stress at the wall is mainly determined by the maximum value and the formula

$$\sigma_{1,2} = \frac{\sigma_x + \sigma_y}{2} \pm \sqrt{\left(\frac{\sigma_x - \sigma_y}{2}\right)^2 + \tau_{xy}^{\;2}}, \qquad (10)$$

where α - the angle between the axis σ_x component output.

Below are the values and graphics stress components recessed into the wall (see Table. 6, Figure 9).

Table 6

Object	α, degree	σ_y, MPa	σ_x, MPa	σ_z, MPa	τz_x, MPa	σ_1, MPa	Δ, MPa	σ_2, MPa
On the contour generation								
h720	48,5	-0,78	-1,82	-2,67	2,93	-5,3	2,6	0,7
h 660	-40,0	-4,23	-4,39	-2,98	-3,94	-7,7	2,0	0,3
30 cm from the generating circuit								
h720	48,5	-3,0	-3,8	-2,84	2,85	-6,2	1,7	-0,4
h 660	-40,0	-7,4	-6,12	-2,84	-3,93	-8,7	1,0	-0,2

70 cm from the generating circuit								
h720	48,5	-6,0	-5,97	-2,77	2,73	-7,5	0,4	-1,2
h 660	-40,0	-8,55	-9,0	-3,23	-2,97	-10,3	-0,6	-2,0

Best Match axial stress σ_1 measurements are achieved on the burial of the wall - 70 cm. This is due to two reasons:

used in processing the results of measurements of the elastic constants, do~~es~~ not match the surrounding rock excavation;

0.7 m penetration is substantially less than the minimum mesh size thickening (parameter finite element mesh). In other words, based on FEM minimum wall element has a size of 1 m, and the measurement results are confirmed within this element.

As seen from the above mentioned results of the study, stresses the value of the circuit is complex workings. Compressive stresses exceed the corresponding calculated values by 2-4 times. The vertical design stresses in the area of the measurement is in the range of 2-4,0 MPa, when only gravity values of the voltages are within 6,3-8,2 MPa, which is 35-45% more than estimated. The joint action of gravity and measured tectonic stress on contour mines, composed of 12 MPa compressive stress in the horizontal array increases to 30-40 MPa.

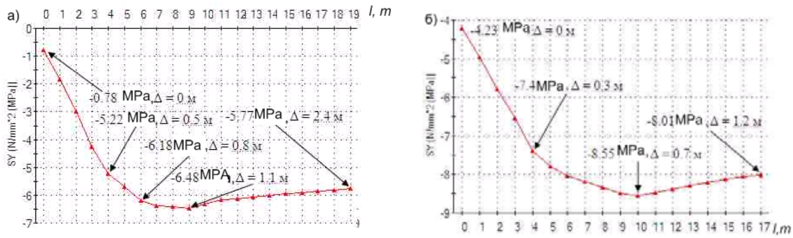

Figure 10. Graph of stress components in the production of burial in the loop: *a* - horizon. 720, σ_y, deep into walls on 2,4 m; *b*- horizon. 660, σ_y, deep into walls on 1,2 m

Thus, the results suggest the following conclusions:

The rock mass fields are uneven component of stress field, which is dominated by horizontal compressive stresses oriented in sub latitudinal direction. They are 1.1-1.4 times the vertical stress, suggesting a determining influence of tectonic forces in the formation of the natural state of stress.

The heterogeneity of natural stress fields due to the complexity and characteristics of the tectonic structure Zarmitan gold ore zone is further enhanced with the anthropogenic impact on the rock mass as a result of mining.

It was found that the maximum voltage at the field is horizontal, directed across the strike of the ore bodies, thus horizontal preparatory mine workings should be carried out as possible taking into account these trends.

Manifestation of rock pressure in a dynamic form depends largely on the level of the existing array of natural tectonic stresses and their directions of action regarding the substantive preparations for mining.

An analytical technique to determine the principal vector of tectonic stresses in the array by measuring axial stresses in three mines is developed and investigated for finite element model of the VAT in the area of two skew openings located at different depths. It is shown that the prerequisites are confirmed by the analytical procedure up to the minimum size of the finite element mesh thickening.

Installed in this paper the tectonic stress field can be used in future studies of VAT array of disturbed mining operations in the development of deposits of Zarmitan gold ore zones and evaluation of the stability of structural parameters of the system development at deep levels of mining.

II. Geomechanical processes when recognition of purposes from cleaner chambers for repeated development of scarn-sheelite deposits

Skarn-scheelite deposits are characterized by a variety of properties, complexity of occurrence of deposits and enclosing rocks. The development areas of these deposits are located in zones where intensive tectonic movement and modern seismodynamics processes occur.

To ensure reliable design, construction and operation of mines, an array of rocks must be carefully studied, as a complex system, where main geomechanical processes take place in extraction of minerals.

A more complete account of geological structure of mountain massif in assessing stress state and stability of mining workings is an important condition for improving safety of mining operations.

It is necessary to carefully approach primary geological information about mountain massif, especially its tectonic structure, fracturing of rocks and mineralogical composition.

The core of exploratory wells should be used as the primary source of geological information in such a way that the obtained data at the stage of detailed exploration allowed to form an idea of the geometry of the spatial location of the strength and elastic characteristics of rocks.

This will allow to carry out purposeful work on engineering-geological study of the mountain massif and prepare the information necessary for the design of mining developments.

The practice of studying composition of structure and state of rock mass allows you to choose your own algorithm for substantiating geomechanical conditions for extraction of pillars during re-engineering.

In presence of a systematic approach to isolation and modeling of structure, in-depth ideas about mechanism of development of deformation and destruction of rock massif and classification of rock collapse in mine workings are possible.

The deposits rocks of hydrothermal genesis of contact structure are characterized by two pairs of chipped fracture systems and a system of cracks in the bedding. Orientation of fracture coincides with orientation of discontinuous disturbances and agrees with zones of tectonic structures. Taking into account genetic and geometric relationships of parameters of cracks and microstructure of rocks with tectonics of deposit determines structural model of rocks of the skarn zone.

The spatial distribution of coefficient of structural attenuation is closely related to geometry of location of fracturing and stress-strain state of mountain massif. In this case, allocated elementary structural block of rock massif must be taken into account when assessing stability of mine workings.

The intensity of formation in mine workings is described by probabilistic-static models on basis of a quantitative evaluation of inhomogeneity of rock mass properties from data of fracture geometrization, coefficient of structural weakening of rock strength parameters.

The implementation of proposed algorithm allows us to determine scope of chamber-pillar development system, to clarify conditions for safe conduct of mining operations and to justify necessity of removing ends from cleaning chambers when skarn-scheelite deposits are being re-developed.

In this case, study of stressed-deformed state of mountain massif will be very effective on basis of numerical methods of continuum mechanics, in particular, differential sweep method.

The application of Runge-Kutta method based on principles of beam theory of elasticity is proposed. Comparison of solution of problems showed high efficiency of proposed method.

In long term, solution of problems to assess volumetric stress-strain state of complex mining facilities will become real in complex dynamic geometrization of forms, occurrence conditions and geomechanical parameters of rock massif on basis of system analysis.

For example, if we consider underground structures based on beam theory, we can model problem as follows (Fig. 1). The zone (*C*) is represented in form of a beam of variable cross-section rigidly clamped at ends with elastic supports (*A*). The upper boundary of the zone (*C*) is determined by line of stratification of soil. The influence of upper (B) on zone (*C*) is determined on basis of Winkler hypothesis under assumption of existence of a one-way connection, i.e. The reaction of upper zone $P(x)$ on beam is proportional to deflection of neutral axis (x) at each point of contact, and if deflection at point is negative, then reaction is zero.

1. Program definition

Earth surface

Assuming that proportionality factor K_1 depends on reaction coordinate x $F(x)$ can be written as

$$P_1(x) = \frac{1}{2}(1 + sign\omega ign\omega))\omega \tag{11}$$

where - $E_1(x)$ modulus of elasticity of material of which zone (E) in compression.

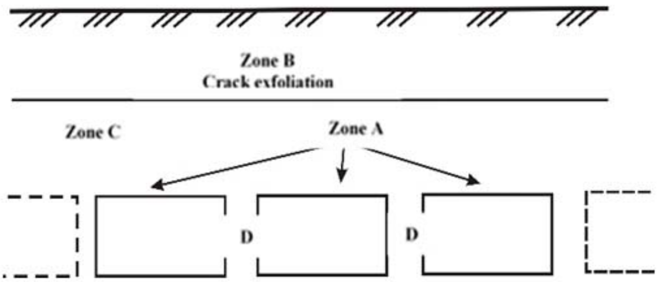

Figure 11. Scheme of an underground structure in a section

$$sign\omega = \begin{cases} 1, if \omega > 0 \\ 0, if \omega = 0 \\ -1, if \omega < 0 \end{cases}$$

The reaction elastic supports $P_2(x)$ is also determined based on Winkler hypothesis (Figure 11).

$$P_2(x) = K_2(x)\omega \tag{12}$$

where, $K_2(x) = \sum_{i=1}^{n} \Gamma_{L_i}^{L_{i-1}} E_1(x)$

$E_1(x)$ - modulus of elasticity of material from which i-th bearing.

$$\Gamma_{L_i}^{L_{i=1}} = \begin{cases} 1, if L_{i=1} \le x \le L_i \\ 0, if (x < L_{i-1}) \prod x > L_i \end{cases}$$

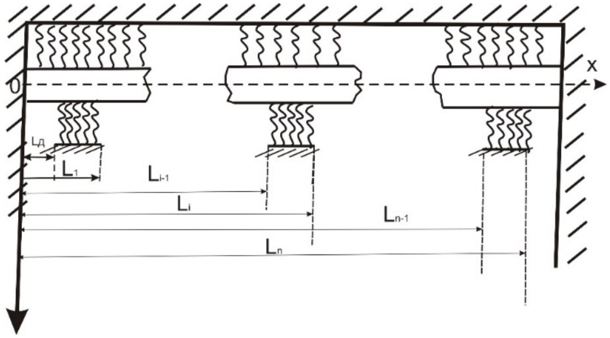

Fig.12. Scheme for determining response of elastic (pillar) supports

When calculating beams of an alternating cross-section without taking into account deformation along height of section and friction between beam and base, following differential equation for bending of elastic line of beam is usually used

$$(S(x)\omega'')'' + P(x) = f(x) \qquad (13)$$

Where $S(x)=I(x)\,E(\mathrm{x})$ is rigidity of beam,

$P(x)$ is reaction of contacting surface,

$F(x)$ is active load.

The active load will be given in form of distributed forces by intensity

$$f(x)=H(x)q(x), \qquad (14)$$

Where $H(x)$ is depth of zone (C), $q(x)$ is linear weight of material from which zone (B) consists.

Since reactions of upper contacting layer of surface and lower supports constitute a general reaction,

$$P(x) = K_2(x) = \tfrac{1}{2}(1 + sign\omega)E_1(x) = f(x) \qquad (15)$$

Substituting (5) into (3), we obtain

$$(S(x)'')'' + K_2(x) - \tfrac{1}{2}(1 + sign\omega)E_1(x) = f(x) \qquad (16)$$

For rigidly constrained edges of beam, boundary conditions of problem will have form

$$\omega(0) = \omega(L) = 0; \omega'(0) = \omega'(L) = 0 \qquad (17)$$

Thus, search for deflection of midline of roof of underground structures under action of mass forces on basis of beam theory reduces to solving ordinary nonlinear equation (6) with boundary conditions (2) and (7).

To solve problem (6), (7), we apply differential sweep method [Mukhitdinov, 1975].

The solution of problem (6), (7) will be sought in form

$$\omega = \alpha(x)[S(x)\omega''] + \beta(x)[S(x)\omega'']' + \gamma(x) \tag{18}$$

where - $\alpha(x),\ \beta(x),\ \gamma(x)$ unknown adjusting factors .

As a result of differentiating (8) and transforming results obtained, we obtain relations forming Cauchy problem for determining running coefficients. Having solved this problem, we define running coefficients along entire length of beam and values of running coefficients.

Having values of functions $M(x)$, Q_x and running coefficients, we can determine deflections ω and angular displacements ω', which in our case will have form

$$\begin{aligned} \omega &= \alpha M + \beta Q + \gamma \\ \omega' &= PM + qM + r \end{aligned} \tag{19}$$

The program is tested on test tasks. In one of them equation

$$((1+x)\omega'' + x\omega = 2(1 = x) + x^2(1-x)^2) \tag{20}$$

With boundary conditions

$$\omega(0) = \omega''(1) = 0,\ \omega'(0) = \omega'(1) = 0 \tag{21}$$

The exact solution of problem

$$\omega = x^2(1-x)^2 \tag{22}$$

in which it is easy to give expression to rotation angle, moment M and shear force Q

$$W' = 2x(1-2x)^2$$

$$M = (1+x)\omega^2 = 2(1+x)(1-6x^2)$$

$$Q = ((1+x))' = 2 - 24x - 36x^2$$

The algorithm of Runge-Kutta method applied to problem (20) consists of following main steps.

1. Broken interval [0.2] for N equal segments points x_k with a constant pitch $\Delta x = L/N$.

2 determines initial values of vector function $z(x)$ for point $x_к=K\Delta x$. Moreover, when K=0, it is determined from initial condition.

3. At point $x_к=\Delta xK$= calculated to vectors

$$\overrightarrow{R_n} = [R_{n1}, R_{n2}, ..., R_{n6}](n = \overrightarrow{1,4})$$

By formulas

$$\overrightarrow{R_n} = \Delta x\vec{F}x_k, Z_1..., Z_6$$

$$\overrightarrow{R_1} = \Delta x\vec{F}\left(x_k + \frac{\Delta x}{2}\right), \quad Z_1 + \frac{1}{2}R_{11}..., Z_6 + \frac{1}{2}R_{16}$$

$$\overrightarrow{R_2} = \Delta x\vec{F}\left(x_k + \frac{\Delta x}{2}\right), \quad Z_1 + \frac{1}{2}R_{11}..., Z_6 + \frac{1}{2}R_{26} \quad (24)$$

$$\overrightarrow{R_3} = \Delta x\vec{F}\left(x_k + \frac{\Delta x}{2}\right), \quad Z_1 + \frac{1}{2}R_{31}..., Z_6 + \frac{1}{2}R_{36}$$

$$\overrightarrow{R_3} = \Delta x\vec{F}\left(x_k + \frac{\Delta x}{2}\right), \quad Z_1 + \frac{1}{2}R_{21}..., Z_6 + \frac{1}{2}R_{16}$$

4. Use of vector *is* vector $\vec{Z}(x)$ of unknown value for a point $x_{k+1} = \Delta x(K+1)$.

Here it should be noted that equation (6) is nonlinear, since coefficient $K(x)$ depends on sign of unknown function $\omega(x)$. Therefore, in general case problem is solved by an iterative method. In first approximation, problems are solved in assumption that deflection is positive along entire parapet of beam. If, in doing so, found points of deflection value at a certain point change sign, then these points are fixed and, taking this into account, problem is solved again. As is well known, coefficient $K(x)$ does not depend on quantitative value of deflection $\omega(x)$, and therefore, with first iteration after taking into account sign of $\omega(x)$, gives a solution with sufficient accuracy.

As an example, let us consider problem of numerical calculation of stability of a roof with a length of 12 m with one clutch support. The support is 2 m wide and is located in center of roof.
The problem is modeled as a rigidly constrained beam by elastic bonds and supported by a support placed centrally in beam.

Material beams and supports same view, which is equal to modulus of elasticity E=9.5×10^4 MPa. The volume weight of rocks above roof is determined by active load acting on beam, intensity of which is given by relation, MPa

$$f(x)=Hq(x)=2{,}62\cdot10^4 . \qquad (25)$$

Elasticity coefficient determining response upper contacting communication is E=10^{-2} E.

Table 7

Results of calculation of roof

X[M]	ω[M]	ω'	M[kg·m]	Q[kg]	σ[kg/M^2]
0,0	0,0	0,0	$2{,}5219\cdot10^4$	$-3{,}6748\cdot10^4$	$2{,}3048\cdot10^3$
1,0	$2{,}51\cdot10^{-5}$	$3{,}66\cdot10^{-5}$	$5{,}6263\cdot10^2$	$-1{,}4109\cdot10^4$	$8{,}7861\cdot10^3$
2,0	$5{,}68\cdot10^{-5}$	$2{,}28\cdot10^{-5}$	$-6{,}9908\cdot10^3$	$-2{,}8737\cdot10^3$	$1{,}9902\cdot10^4$
3,0	$6{,}71\cdot10^{-5}$	$-2{,}77\cdot10^{-6}$	$-7{,}7859\cdot10^3$	$6{,}8467\cdot10^2$	$2{,}3447\cdot10^4$
4,0	$5{,}21\cdot10^{-5}$	$-2{,}58\cdot10^{-5}$	$-5{,}3122\cdot10^3$	$5{,}1422\cdot10^3$	$1{,}8248\cdot10^4$
5,0	$2{,}16\cdot10^{-6}$	$-2{,}92\cdot10^{-5}$	$5{,}4043\cdot10^3$	$1{,}8101\cdot10^4$	$7{,}5926\cdot10^4$
6,0	$6{,}23\cdot10^{-5}$	0,0	$9{,}2271\cdot10^3$	$-1{,}8187\cdot10^2$	$2{,}1807\cdot10^4$
7,0	$2{,}16\cdot10^{-5}$	$2{,}92\cdot10^{-5}$	$5{,}4043\cdot10^3$	$-1{,}8100\cdot10^4$	$7{,}5926\cdot10^4$
8,0	$5{,}21\cdot10^{-5}$	$2{,}58\cdot10^{-5}$	$-5{,}3119\cdot10^3$	$-5{,}1419\cdot10^3$	$1{,}8248\cdot10^4$
9,0	$6{,}71\cdot10^{-5}$	$2{,}77\cdot10^{-6}$	$-7{,}7853\cdot10^3$	$-6{,}8453\cdot10^2$	$2{,}3479\cdot10^4$
10,0	$5{,}68\cdot10^{-5}$	$-2{,}28\cdot10^{-5}$	$-6{,}9904\cdot10^3$	$2{,}8729\cdot10^3$	$1{,}9902\cdot10$
11,0	$2{,}51\cdot10^{-5}$	$-3{,}66\cdot10^{-5}$	$5{,}6129\cdot10^2$	$1{,}4106\cdot10^4$	$8{,}7889\cdot10^3$
12,0	0,0	0,0	$2{,}5213\cdot10^4$	$3{,}6743\cdot10^4$	0,0

Table 1 presents results of numerical calculations, i.e. Numerical values of investigated values: ω-deflection, ω'-angle of rotation, M-bending moment, Q-shear force and σ-stresses, which appear at contact of upper layer at different points of span of beam. These quantities have a symmetrical character with respect to center of span, with bending moment and shear force taking largest values at extreme points. This is because effective load, evenly distributed over span of beam, is rigidly clamped.

In area of support of beam, bending moment has a local maximum, which is achieved at center of diagram, and extreme values of shearing force reach at edges of diagram and are equal to zero at its center.

As a second example, we consider a problem similar to the first, differing from the first by presence of two supports. In area of support, qualitative picture of distribution of investigated quantities is repeated. Hence it can be concluded that in calculation of symmetrically loaded structures it is sufficient to perform studies with one support.

Below is an example of use of this technique for re-excavation of pillars in Koitash deposit. Investigation of stress-strain state of rock massif was carried out with reference to following conditions. Specific gravity of roof rock γ=2,7 kg/m^3 and developing depth H=100 m. Block length 38 m, width 2 m pillar width passage chambers 6 meters. The immediate simulate roof like elastic beam height $h_{b,}$ which lies above main roof. Modulus immediate roof and pillar accept equal $E=4\times10^4$ Pa. The rigidity of ends is assumed to be same and is determined by formula

$$C=\alpha^{T}E \tag{26}$$

where - ratio of specific fracture pillars.

Load from rocks weight of main roof per running meter of its length is, MPa

$$P=\gamma N=2{,}7\times10^4 \tag{27}$$

The elasticity coefficient determining reaction of upper contacting bond is equal to

$$E=\beta E=10^{-2}\,E, \tag{28}$$

Where β is a coefficient that depends on degree of opening of fissure cracks throughout beam.

Note that support stiffness is inversely proportional to specific fracture (α'), therefore stress-strain state of both beam and stresses arising in area of contact areas depends on this factor.

The areas of compressive appearance and tensile stresses depend on rigidity of ends. With increasing stiffness of pillars, compressive stresses appear not only at ends of span, but also in region where beam contacts ends.

As a rule, tensile stresses are more dangerous for brittle ore deposits, so knowledge of areas of occurrence of tensile stresses and their extreme values plays an important role in assessing stability of structures.

Maximum tensile stress appears at low severity pillars in center span σ_p =11 MPa, and under more stringent pillars before last span σ_p=560 MPa.

The compressive stress at contact with supports is calculated by formula

$$\sigma_0=\alpha^T E\omega \tag{29}$$

So here Single measurement H_0(m) and deflection in cm, so, MPa

$$\sigma_0=\frac{4\cdot 10^5\omega}{100H_0}=4\cdot 10^2\ \omega/H_0 \tag{30}$$

With increasing stiffness of pillars, stress in them increases, and tensile stress arising in upper layer decreases.

At same time, relative rate of increase in voltage at extreme ends is much greater than in central ones.

As a result, greater rigidity of ends, better average stress on supports is equalized, i. E. These voltages will differ little on supports.

With absolutely rigid supports, these stresses will be same in internal supports, and deviations on outriggers will be insignificant [Rakhimov V.R., 1998; Sayyidkosimov S.S., 2016].

Now we examine some parameters of this problem with a certain removal of ends from right to left, i.e. We perform several successive calculations, deleting each time one row of ends. In this case, length of the first span will increase due to neighboring span.

The maximum values of deflections of roof increase and shift to the right, and their relative growth depends not only on length of adjacent span, but also on order of removal.

For this task, after the first removal, maximum deflection value increased by 7%. After removing second row, it increased by 82%, with maximum span length becoming 22 m. This is 366 times greater than length of remaining spans.

After first removing tensile stress values increase slightly max/$\sigma_x(x)$/=5,1 MPa, but maximum stress position is shifted to right.

After removal of second row of pillars maximum stresses are increased by 11% and become max/$\sigma_x(x)$/=11,2 MPa, elongation relative span is 32% (L_1=22 m).

Here it should be noted that increasing length of a given span leads to an increase in stresses on same span, on adjacent spans voltage decreases.

When removing first row of springs, a redistribution of stresses occurs. As a result, in the second row, voltage will increase by 20%, in third by 5-8, next row of noticeable increase in voltage does not occur.

With successive excavation of second row on third, voltage increases by 80%, and pillars will increase by almost 2,5 times and their safety factor will decrease from 2,5 to 1,3.

Thus, permissible size of outcrops will reach 18-20 m, which corresponds to removal of two rows of pillars without preliminary collapse of roof in rework chamber.

When Koitash deposit was re-developed in process of mining, constant redistribution of stresses in worked blocks and surface displacement was carried out.

From analysis of results of observations it is clear that stress distribution in pillars is cyclical [Sayyidkosimov, 2016].

First of all, concentration of stresses in edge part of casing increases, then central part is loaded, over time (1-2 months), stresses are distributed quite evenly over entire section of cage.

With repayment of a group of islands, distribution of loads along system of remaining pillars in block occurs unevenly. The most part (70-80%) is taken by lobes located at a distance of 1-15 m from boundary of exposed roof. The smaller part (30-20%) falls on pillars, located 20-50 m from boundary of outcrop of roof.

When blocks are reprocessed, supporting ore pillars remain stable at all stages of development, safety factor is R=1,3-4,0.

According to calculation, maximum deformations of surface are no more than 9.4 cm, which is permissible and excludes possibility of collapse, dips, shifts and formation of funnels on earth's surface.

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SUBSTANTIATION AND DEVELOPMENT OF PREDICTING METHODS FOR THE DYNAMICS OF GAS EMISSION FROM THE COAL-BEARING ROCK STRATUM BEING UNDERMINED

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Abstract

Based on mine experiments, the maxima of methane release and corresponding to them removals of stope faces from face entries have been set. The stope face sizes and empirical dependences for calculating the angular and linear parameters of the crustal movement and rocks displacement, corresponding to the maxima of gas emission, have been determined.

The theory of the joint influence has been considered of coal production and the stope works development on the gas balance formation in the extraction site. The compliance of the adopted scientific provisions with the practice of conducting the stope works has been examined on the basis of experimental data, obtained under conditions of 'A.F. Zasyadko' Mine, 'Suhodolskaya - Vostochnaya' Mine PJSC 'Krasnodonugol' and 'Gazeta Izvestiya' Mine Donbasantratsit, DP.

The performed theoretical and experimental studies made it possible, for a first approximation, to propose a general procedure for predicting the basic parameters of gas emission dynamics at all stages of the extraction sites operation.

Introduction

The relation between the level of gas emission from coal of the contiguous seams and its residual gas content with the intensity of the undermined rocks displacement is qualitatively confirmed by the results of mine experiments [1]. The change in gas pressure and absolute methane release from the undermined seams under the influence of the working stope face has been established by the studies [2]. Up to date, there are no works in which the parameters of rocks displacement and gas emission dynamics would have been considered together, when removing the stope faces from the first cut workings, as well as development of mining operations within the extraction sites and mine field as a whole.

Consideration of this question is relevant to the creation of safe conditions during the stope works conducting, as in this period the primary and subsequent main roof subsidence occurs and the gas emission maxima from the undermined coal-bearing rock stratum are achieved. It has been found [3], that when removing a stope face from the face entry after the main roof subsidence, a periodic increase in methane release into the gas drainage boreholes and into mine workings, is observed. The alternation of gas emission maxima has been set through the entire time of mining the extraction sites [4]. The purpose of the work is to determine the parameters of the stope mine workings and the rock displacements corresponding to the maxima of gas emission. To do this, on the basis of mine experiments, the maxima of methane release and the corresponding to them removals of stope faces from the face entries have been set. Having determined the dimensions of the stope mine workings and using empirical dependences, it is possible to calculate the angular and linear parameters of the crustal movement and rocks displacements [5-8], corresponding to the maxima of gas emission.

Up to now, the joint influence of the coal production level and the degree of stope works development on gas emission from the undermined sources has not yet been studied.

The disclosure of the mechanism for the gas balance formation in the extraction sites under the influence of specified factors is necessary for the development of effective scientifically-based measures for the safe mining of gas-bearing coal seams.

The method of work provided for a theoretical consideration of the joint influence of coal production and the stope works development on the gas balance formation in the extraction site, conditioned by the methane release from the undermined sources. The compliance of the adopted scientific provisions with the practice of conducting the stope works was examined on the basis of experimental data, obtained under conditions of 'A.F. Zasyadko' Mine [9], 'Suhodolskaya - Vostochnaya' Mine PJSC 'Krasnodonugol' and 'Gazeta Izvestiya' Mine Donbasantratsit, DP.

Factors determining the level of gas emission

In almost all cases, a change in a wide range of coal production and parameters of the displacement process of the rocks being undermined occurs in the initial period of the extraction site operation when the stope face is removed from the face entry. At this stage, the coal production increases from zero to planned indicators. In parallel with the coal production growth, the displacements processes of undermined rocks are developing, under the influence of which the main roof subsidence occurs during this period. The achievement of the planned indicators of coal production and the main roof subsidence leads to an increase in gas emission to maximum values. In practice, providing for the planned coal production and the main roof subsidence may not coincide in time and space. For this reason, in addition to the considered variant, the other ones are possible. For example, if the planned load is provided almost from the first days of the longwall face operation, then an increase in gas emission is observed in a greater degree under the influence of the stope works development and the displacement processes of the rocks being undermined. It should be noted that in all cases of the initial stage of the extraction site mining, the most integral parameter of the gas emission characteristic is the distance between the stope face and the face entry. It is connected both with the growth in coal production (the rate of the stope face advance) and the development of the displacement processes of rocks being undermined. Given the above mentioned, the first stage of the extraction site operation should be considered with account of two criteria – the achievement of planned coal production and the maximum level of gas emission as the stope face is removed from the face entry.

The stable (planned) coal production corresponds, as a rule, to the next stage of the extraction site operation. During this period, there is almost no correlation dependence of gas emission from an insignificant change in the load on the stope face, and the non-uniformity and change in the level of methane release from the undermined sources are determined by the displacement processes of undermined rocks or by other factors [10].

The final stage of the extraction site operation, with account of the specifics of mining the coal seams with long longwalls, is connected with a gradual reduction in coal production until its complete shutdown. During this period, there is a strong correlation dependence of gas emission on the level of coal production [10]. This is obviously caused by a decrease in the intensity of the undermined rocks displacement with an insignificant rate of the stope face advance.

The most important period for ensuring the safe conditions for coal seam mining is the growth of gas emission when removing the stope faces from face entries. The increase in methane release in this case is related both to the development of displacement processes of coal seams and host rocks that are being undermined, and with the growth of coal production when the planned indicators are reached. It is believed that the first absolute maximum of gas emission is determined by the main roof subsidence. According to the calculation schemes, with all other conditions being equal, the level of subsequent gas emission maxima should not exceed its value at the initial main roof subsidence. The main roof subsidence can be considered as an established fact, if the methane release into the gas drainage boreholes, drilled over the face entry (end), reached a maximum, and subsequently decreases to insignificant values [11].

The consideration of the total gas emission into mine workings and gas drainage boreholes allows to determine the methane release maxima, regardless of the change in the efficiency of the sources degassing by the drainage boreholes during the entire period of mining the extraction sites. When analysing the dynamics of methane release, it is necessary to use the results of long-term mine observations related to mining both separate longwall faces and the entire mine field.

An increase in the level of gas emission from the host rocks and coal seams undermined by stope mine workings was established in the last century.

In the following years, a number of works of domestic and foreign researchers was devoted to study of the relation of gas emission with the stress state of rocks, as well as the parameters of their displacement. These works are aimed at solving the problems of degassing of contiguous seams and host rocks, preventing the sudden coal and gas emissions, increasing the protective effect of degassing on the emission risk of contiguous seams, predicting the level of gas emission into mine workings and many other problems of mining science.

The most significant factors determining the level of gas emission from the undermined coal-bearing rock strata are absolute coal production (the rate of the longwall face advance) and the degree of stope works development [12].

The only one factor is considered by a normative document [13], when predicting the gas emission with other conditions being equal - the level of coal production. Such an approach does not take into account the degree of stope works development and does not allow to predict the dynamics of gas emission within the extraction site or the entire mine field. Up to the present, the quantitative relations of the parameters of the stope mine workings and the rocks displacement with the processes of gas emission have not been studied.

The characteristic parameter of the degree of stope works development at the extraction site is the removal of the stope face from the face entry, at which, respectively, the gas emission growth from the undermined sources begins and its maximum value is reached. Based on the theoretical prerequisites, the local gas emission maxima should be achieved after the main roof subsidence, and the absolute ones - after the processes of displacement have reached the rocks of the earth's surface. In the first case, only the sources of gas emission, caused by the primary main roof subsidence, fall into the zone of stope works influence. In the second case, there is a disturbance of the initial natural state of the entire coal-bearing rock stratum from the mined seam to the earth's surface.

Analysis of the experimental and calculated parameters

The coal seams mining may be accompanied by different ratios between the length of the operating longwall face $L_л$ and removal of the stope face from the face entry, respectively, with the main roof subsidence L_0 and the achievement by the displacement processes of the undermined rocks of the earth's surface L_H. The different ratios between the parameters $L_л$,L_0 and L_H determine the upper boundary of extension of the complete displacement zone of undermined rocks H_p with their discontinuity [14].

The parameter H_p characterizes the height of the undermined rocks zone, from which the gas may flow into mine workings. In some cases, the value H_p after the main roof subsidence remains constant, in other cases it increases. This, ultimately, determines the conditions for achieving the maximum level of gas emission from the undermined sources (table. 1). The gas emission (I_{m}^{3}) will almost be equal to zero in compliance with the ratio $L_{\text{л}}<L_0<L_H$. The absolute maximum value of methane release (I_{m}^{6}) is achieved under the condition $L_{\text{л}}<L_0<L_H$. In other variants of ratios $L_{\text{л}}$, L_0 and L_H, the possible gas emission maxima will be in the range of values between I_{m}^{3} and I_{m}^{6} .

In the conditions of mines 'Gazeta Izvestiya' and 'Suhodolskaya - Vostochnaya' the observations have been conducted of the dynamics in gas emission during the mining of sixteen extraction sites in total. By an experiment, the distances between the stope faces and the face entries have been determined, at which the growth of gas emission from the undermined sources began l_H^1 and gas emission maxima were achieved, respectively, into the gas drainage boreholes $l_m^{скв}$ as well as its total emission into mine workings and into the gas drainage boreholes l_m^c. Additionally, the data obtained in the conditions of 'A.F. Zasyadko' Mine were used [9].

Table 1

The ratio between the parameters $L_{\text{л}}$, L_0 and L_H and corresponding to them possible levels of gas emission with the displacement processes of undermined rocks and earth's surface

Ratio between $L_{\text{л}}$ and L_H	Possible variants of $L_{\text{л}}$ and L_0 ratio	Distinctive features of the processes that determine the upper boundary of extension of the complete displacements zone of the undermined rocks and the degree of undermining the earth's surface.	The height of the upper boundary of the complete displacement zone with discontinuity	Possible features of the gas emission processes	Conditions for achieving the maximum gas emission I_m
$L_{\text{л}} < L_H$	$L_{\text{л}} > L_0$	The primary main roof subsidence occurs before the formation of the square of mined-out space with side $L_{\text{л}}$. The subsequent subsidence is possible, but the processes of rock displacement do not reach the earth's surface.	$H_{\text{р}}^1 - var$	Gas emission occurs only from a certain part of the undermined sources, since during the formation of the square of mined-out space with side $L_{\text{л}}$, the displacement processes do not reach the earth's surface	I_m^1 is achieved gradually with the removal of the face from the face entry at a distance equal to $L_{\text{л}}$.
	$L_{\text{л}} = L_0$	The primary main roof subsidence occurs before the formation of the square of mined-out space with side $L_{\text{л}}$. The position of the upper boundary of the complete displacement zone with respect to the undermined seam remains constant. The processes of rock displacement do not reach the earth's surface and trough is not formed.	$H_{\text{р}}^2 - const$		Gas emission occurs intensely at $L_{\text{л}} = L_0$, $I_m^2 > I_m^1$.
	$L_{\text{л}} < L_0$	The primary main roof subsidence is absent. The crustal movement does not occur. The parameters ratio is typical for extensive mine workings or pillar mining of seams.	$H_{\text{р}}^3 = 0$		$I_m^3 \approx 0$.
$L_{\text{л}} = L_H$	$L_{\text{л}} > L_0$	After the primary main roof subsidence, the consequent displacements occur till the formation of the square of mined-out space with side $L_{\text{л}} = L_H$. The processes of rock displacement reach the earth's surface; the trough is not formed.	$H_{\text{р}}^4 - var$	Gas emission is possible from all the sources, since the displacement processes reach the earth's surface during the formation of the square of mined-out space.	Maximum value of I_m^4 is achieved gradually with the removal of the face at a distance equal to $L_{\text{л}}$.
	$L_{\text{л}} = L_0$	The primary main roof subsidence occurs during the formation of the square of mined-out space with side $L_{\text{л}} = L_H$. The processes of rock displacement reach the earth's surface; the trough is not formed.	$H_{\text{р}}^5 - const$		Intensive gas emission occurs when the face is removed at a distance more than $L_{\text{л}}$. $I_m^5 > I_m^4$.
$L_{\text{л}} > L_H$	$L_{\text{л}} > L_0$	The primary main roof subsidence occurs before the formation of the square of mined-out space with side $L_{\text{л}}$. The subsequent subsidence is possible, as a result of which the processes of rock displacement reach the earth's surface with the formation of a trough.	$H_{\text{р}}^6 - var$	The displacement processes of undermined rocks and the earth's surface receive their maximum development, which determines the maximum level of gas emission.	Value I_m^6 is achieved gradually. The following ratio between the maximum values of gas emissions is provided: $I_m^6 > I_m^5 > I_m^4 > I_m^2 > I_m^1$.

Based on the empirical equation [15] for mining-geological and mining-engineering conditions under study, the calculated values of the distances between the face entries and stope faces have been set L_H, at which the displacement processes of undermined rocks reach the earth's surface.

Similarly, according to [7], the calculated height of the rock displacement zone with their discontinuity for each object have been determined H_p^1 and H_p, respectively, when removing the stope face at a distance $L=L_H$ and $L=L_л$. When removing the stope face at a distance L, which is equal to the length of the longwall face L, the square of mined-out space is formed, determining the zone of stope works influence.

With further advance of the longwall face, the zone of mined-out space influence is formed, in which the parameters of the new displacements area are not changed until the completion of mining the entire extraction site [16].

Reasoning from this scientific position, the values H_p during operation of each extraction site, will remain approximately constant, and this affects the character of the gas emission dynamics.

A joint analysis of the experimental and calculated parameters has shown that the maxima of the total gas emission into mine workings and gas drainage boreholes depend on the degree of stope works development (*B/H*) in the mine field (fig. 1).

The maxima of total methane release into mine workings and gas drainage boreholes I_m^c are determined by the linear dimension of mined-out space *B* and the depth of stope works conducting *H*.

The achievement of the value I_m^c occurs when the stope face is removed from the face entry at a distance l_m^c, the absolute value of which is described by the exponential equation depending on *B/H.*

The conditions of incomplete earth's surface undermining $B/H \leq 1{,}4$ correspond to higher values of l_m^c. When $B/H > 1{,}4$, the distance l_m^c remains approximately constant and is in the range of 80-120 м.

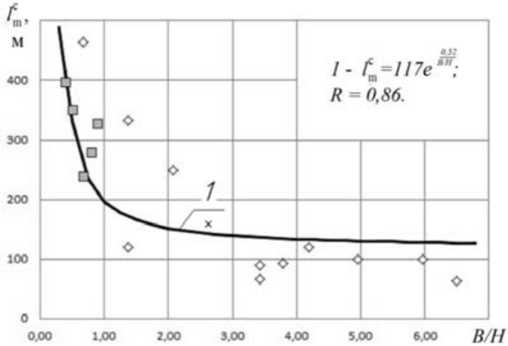

Fig. 1 - The dependence of the distance between the stope faces and face entries l^c_m on the degree of stope works development in the mine field *B/H*, at which the maximum of total gas emission is achieved: *1* - smoothing curve; *R* - correlation ratio; ◇, ■, × - experimental data obtained respectively in conditions of mines 'Gazeta Izvestiya', 'Suhodolskaya - Vostochnaya' and 'A.F. Zasyadko' Mine [9]

The parameter L_H depends on the depth of stope works conducting *H*, thickness of mine seam (*m*) and rate of the stope face advance $\upsilon_{оч}$ [15]. The main of these factors is the depth, it determines the value of the parameter L_H to 80% [17].

The research method provided for the analysis of possible factors determining the gas emission from the sources being undermined at different stages of the extraction sites mining with the subsequent examination of compliance of the adopted scientific provisions with the results obtained in mine conditions. These factors at the first stage of the extraction site operation included the growth of coal production, the stope works development (removal of the face from the face entry) and the related processes of the undermined rocks displacement.

At the stage of the stable stope face work, the main factor determining the level of gas emission is the planned coal production. With an approximately constant load on the stope face, the fluctuations in gas emission are caused by the displacement processes of undermined rocks.

At the final stage of the longwall face work, the coal production decreases until the complete shutdown of stope works, which causes a reduction in gas emission to almost zero.

Based on logical reasoning, it was assumed that after the beginning of the extraction site operation and the removal of the stope face from the face entry L, the coal production growth A occurs according the exponential dependence

$$A = A_m(1 - e^{-k_1 \cdot L}), \qquad (1)$$

where A_m - planned (maximum) level of coal production, tons/day;

k_1 - empirical coefficient characterizing the change in coal production, after the beginning of the extraction site operation.

When choosing the dependence of gas emission on the distance between the stope face and the face entry L, the presumption was that the methane release into this mine working is insignificant before the beginning of the stope works operation, and in rare cases exceeds 1 m^3/min. This is explained by the absence of coal-getting during assembly operations. Their duration can range from several weeks to several months. During this time, gas emission from the exposed surfaces of a face entry decreases to minimum values. For this reason, the background value of gas emission before the beginning of stope works is close to zero, and the current change in methane release I_c as the distance L increases, may also be described by an exponential curve coming out of the beginning of coordinate grid

$$I_c = I_m(1 - e^{-k_2 \cdot L}), \qquad (2)$$

where I_m - gas emission level which corresponds to the planned coal production A_m, m^3/min;

k_2 - empirical coefficient characterizing the mining-geological and mining-engineering conditions of a particular longwall face operation.

With the development of stope works, the methane release occurs both into mine workings and into the gas drainage boreholes. In equation (2), it was considered their total value I_c.

The results of experimental data processing, according to the equations (1 and 2), are presented in the Table 2.

Table 2

Information on empirical coefficients of exponential equations (1, 2) and parameters characterizing the dependence of coal production (A) and gas emission (I_c) when removing the stope faces from face entries

Longwall face	Equation 1				Equation 2			
	production A_m, tons/day	empirical coefficient k_1	correlation ratio R	mean-square deviation σ_A, tons/day	methane release I_m, m^3/min	empirical coefficient k_2	correlation ratio R	mean-square deviation σ_I, m^3/min
'Suhodolskaya - Vostochnaya' Mine								
12th bis Eastern	1039	0,12	0,9	77	21,9	0,0	0,8	3,2
24th Eastern	2034	0,04	1,0	28	53,3	0,0	0,9	7,8
25th Western	1486	0,04	0,9	102	40,6	0,0	0,8	2,5
34th Eastern	687	0,09	0,9	65	12,8	0,0	0,9	1,0
37th Western	954	0,01	0,9	113	16,4	0,0	0,9	0,9
'Gazeta Izvestiya' Mine								
1st bis Western	280	0,01	0,8	62	-	-	-	-
2nd Western	1095	0,02	0,9	250	37,3	0,0	0,9	4,4
3rd Western	1313	0,01	0,9	201	37,7	0,0	0,9	4,4
4th Western	1722	0,00	0,6	397	-	-	-	-
5th western	1222	0,01	0,8	261	42,4	0,0	0,9	7,4
6th Western	775	0,42	0,3	280	22,7	0,0	0,8	4,8
7th Western	1195	0,02	0,9	145	-	-	-	-
8th Western	1342	0,01	0,9	71	65,1	0,0	0,9	9,2
9th Western	872	0,07	0,9	74	13,9	0,0	0,9	1,6
'A.F. Zasyadko' Mine [9]								
16th Eastern	2878	0,01	0,9	66	80,9	0,0	0,8	14,1

As an example (Fig. 2) the diagrams are given of a change in the level of coal production and total gas emission into mine workings and boreholes when mining the 25 th Western longwall face of the 'Suhodolskaya - Vostochnaya' Mine. It can be seen from the diagrams, that the coal production indicators have been achieved with removal of the stope face from the face entry at 64 m. The total gas emission, which corresponded to this level of coal production, occurred at a distance of about 200 m between the stope face and the face entry. This ratio indicates that when the distance is less than 200 m, the level of gas emission was influenced by two factors - coal production and the degree of stope works development. After removal at a distance of more than 200 m, the level of gas emission (40,6 m^3/min) was determined by a planned load (1486 tons/day) on the stope face of the 25th Western longwall face (Table 2), and the stope works development (removal of face from the face entry) no longer influenced the process of methane release (Fig. 2).

Such a change in coal production and gas emission was typical for extraction sites of mines ‘Suhodolskaya - Vostochnaya’ and A.F. Zasyadko. This is confirmed (Table 2) by high values of the correlation ratios (*R*) for both coal production (0,95÷1,00), and methane release (0,80-0,99).

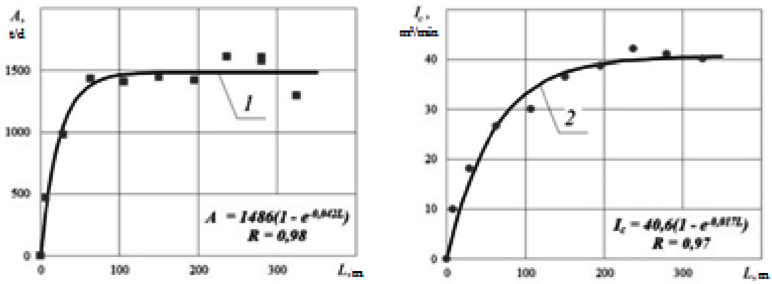

Fig. 2 – An example of the change in coal production (*A*) (a) and total gas emission (I_c) into mine workings and boreholes (б) from the distance (*L*) between the stope face and the face entry when mining the 25th Western longwall face of the i_3' seam in the ‘Suhodolskaya - Vostochnaya’ Mine: ■, ● - experimental data; *1*,*2* - curves of change in coal production and gas emission, respectively; *R* - correlation ratio

In the conditions of ‘Gazeta Izvestiya’ Mine, the growth of coal production occurred in a similar way (Table 2) at seven extraction sites out of nine (*R*=0,85-0,99). At the two sites, the correlation ratios were significantly weaker. When operating the 6th Western longwall face, such a dependence was practically absent (*R*=0,36). Not a typical change in coal production caused by the worsening of mining and geological conditions had practically no influence on the character of the change in total gas emission (*R*=0,84). This evidences a significant influence on the level of gas emission in the initial period of the extraction sites operation of the displacement processes of the undermined rocks when removing the stope faces from face entries.

The linearly proportional dependences of L_m on A_m have been set. They are highly individual for specific mining and geological conditions (Fig. 3).

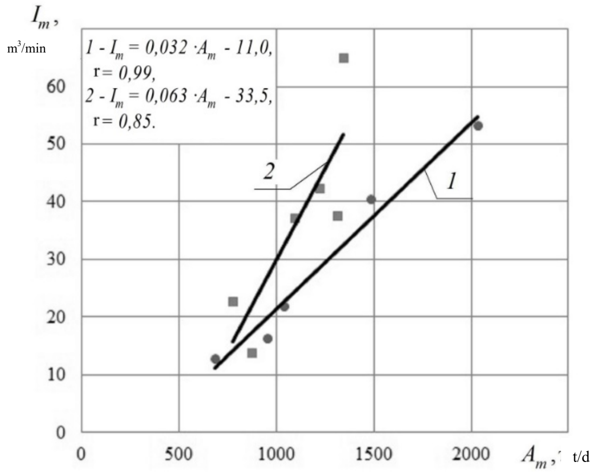

Fig. 3 - The dependence of maximum gas emission (I_m) on the planned load (A_m) on the stope faces: *1*,*2* - smoothing lines for extraction sites of 'Suhodolskaya - Vostochnaya' Mine and 'Gazeta Izvestiya' Mine, respectively; ●, ■ – experimental data; r – correlation coefficient

The empirical coefficient k_1 and k_2 did not practically depend on A_m. The correlation coefficients r are 0,30 and 0,14, respectively. In turn, the values of k_1 and k_2 correlate with each other (Fig. 4), which is obviously conditioned by the similar processes of the undermined rocks displacement in different mining and geological conditions.

In most cases (more than 90%), the growth of coal production A until reaching the planned loads A_m in the initial period of the extraction sites operation is described by an exponential dependence when removing the stope face L from the face entry. According to a similar dependence, a change in the total gas emission occurs $I_c = I_m(1 - \exp k_2 \cdot L)$. A linearly proportional dependence between the empirical coefficients has been set ($k_2 = 0.225 \cdot k_1 + 1.31 \cdot 10^{-2}$), which makes possible to plan the rate of achieving the planned loads on the stope face A_m and predict the dynamics corresponding to them of gas emission from the coal-bearing rock strata being undermined.

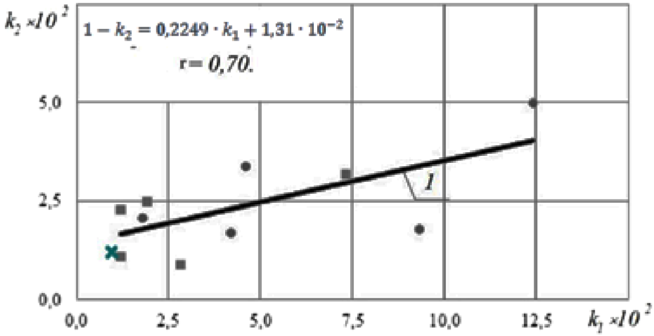

Fig. 4 - The dependence of the empirical coefficient k_2 on k_1:●, ■, × - experimental data for extraction sites of mines 'Suhodolskaya - Vostochnaya', 'Gazeta Izvestiya' and 'A.F. Zasyadko' Mine, respectively; *1* - smoothing line; *r* - correlation coefficient

The higher k_1 by the absolute value, the steeper the curve of coal production growth corresponding to the equation 1. Based on the ratio between $k_1 > k_2$, the dependence of gas emission according to equation (2) will be flatter if compared to the curve of coal production growth (1) when removing the stope face from the face entry. This indicates that the increase in gas emission falls behind by time and space from the growth of the load on the stope face.

For the extraction sites under consideration (Table 2), the correlation dependence has not been set between the mean-square deviation σ_A and the value of A_m (r=0,20), which is obviously related to the different technology and organization of production cycles at the extraction sites under consideration.

Between the mean-square deviation of gas emission σ_I and the value I_m, a linearly proportional dependence was observed (Fig. 5). The ratio between σ_I and I_m. indicates that the change σ_I occurs in proportion to I_m. This indicates that with an increase in I_m , the non-uniformity coefficient of gas emission remains approximately constant, rather than decreases, as it is accepted according to [13].

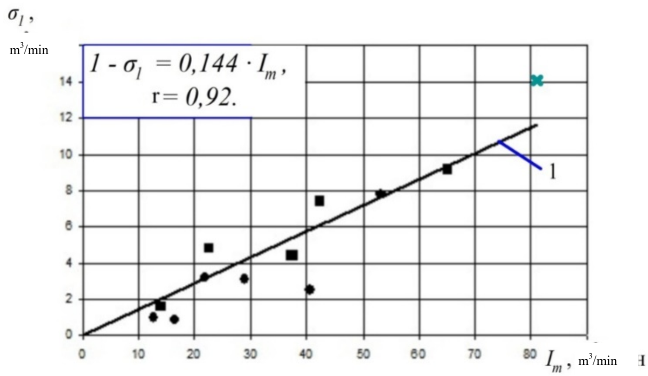

Fig. 5 - The dependence of mean-square deviation of methane release (σ_I) on its maximum value (I_m): ●, ■, × - experimental data for extraction sites of mines 'Suhodolskaya - Vostochnaya', 'Gazeta Izvestiya' and 'A.F. Zasyadko' Mine, respectively; *1* - smoothing line; r - correlation coefficient

Based on the performed research, a linearly proportional dependence has been set between the mean-square deviation of gas emission σ and the values of maximum gas emission $I_m \sigma$=0,144I_m. This evidences the constancy of the non-uniformity coefficient of gas emission from the coal-bearing rock stratum being mined ($K_H \approx 1.43$) regardless of the level of gas emission.

The second stage of the extraction sites operation was characterized by stable coal production and the gas emission level corresponding to it. The gas emission fluctuations at this stage occurred in a greater degree as compared with a change in coal production. For example, in the conditions of 'A.F. Zasyadko' Mine, according to data [9], the average monthly production was in the range of 2308-2716 tons/day, and gas emission within 57,7-108,5 m^3/min. In this case, with a change in coal production by less than 20%, the gas emission fluctuations were about 90%. This indicates that with a stable work of the extraction site, the non-uniformity of gas emission was largely determined by the displacement processes of undermined rocks or by other factors.

The performed theoretical and experimental studies made it possible, for a first approximation, to propose a general procedure for predicting the basic parameters of gas emission dynamics at all stages of the extraction sites operation. This procedure involves the following calculations:

having set the degree of closeness of the current values of coal production A to the planned indicator A_m and removal of the stope face from the face entry L, on which $A \approx A_m$ from the equation (1), the coefficient k_1 is determined;

according to the equation (Fig. 4) by the value of k_1, the coefficient k_2 of the equation 2 is calculated;

having accepted the current value of $I_c \approx I_m$ from the equation 2, the distance L is determined, at which the gas emission I_m will be achieved;

using one of the known methods for predicting the planned coal production A_m, the level of gas emission I_m is determined. In addition to predicting the gas emission according to the normative document [13], the empirical dependences similar to that shown in the Fig.3 can be used;

for the stage of stable extraction site work, the possible fluctuations of I_m are determined with account of the mean-square deviation (σ_I), being determined according to dependence (Fig. 5);

at the final stage of cleaning-up the longwall face, the gas emission will decrease in proportion to the reduction of coal production. The rate of its decrease can be predicted on the basis of the conditions for cleaning-up the extraction site.

Conclusions

1. The maxima of methane release into the boreholes drilled over the first cut workings are observed in the interval after the primary main roof subsidence before the removal of the stope faces from the installation chambers at a distance not less than the length of the longwall face.

2. In the initial period of mining the extraction site, the most integral parameter of the gas emission characteristic is the distance between the stope face and the face entry. This is conditioned by the coal production growth (the rate of the stope face advance) and the simultaneous development of the displacement processes of rocks being undermined. In this case, the operation conditions of the extraction site should be considered with account of two criteria - the achievement of planned coal production and the maximum level of gas emission as the stope face is removed from the face entry.

3. The maxima of total methane release into mine workings and gas drainage boreholes I_m^c are determined by the linear dimension of mined-out space *B* and the depth of stope works conducting *H*. The achievement of the value I_m^c occurs when the stope face is removed from the face entry at a distance l_m^c, the absolute value of which is described by the exponential equation depending on *B/H*. The conditions of incomplete earth's surface undermining (*B/H*≤1,4) correspond to higher values of l_m^c. When *B/H*>1,4, the distance l_m^c remains approximately constant and is in the range of 80-120м.

4. In most cases (more than 90%), the growth of coal production *A* until reaching the planned loads A_m in the initial period of the extraction sites operation is described by an exponential dependence when removing the stope face *L* from the face entry $A = A_m(1 - \exp k_1 \cdot L)$. According to a similar dependence, a change in the total gas emission occurs $I_c = I_m(1 - \exp k_2 \cdot L)$. A linearly proportional dependence between the empirical coefficients has been set ($k_2 = 0.225 \cdot k_1 + 1.31 \cdot 10^{-2}$), which makes possible to plan the rate of achieving the planned loads on the stope face A_m and predict the dynamics corresponding to it of gas emission from the coal-bearing rock strata being mined.

5. A linearly proportional dependence has been set between the mean-square deviation of gas emission (σ) and the values of maximum gas emission $I_m \sigma$=0,144I_m. This evidences the constancy of the non-uniformity coefficient of gas emission from the coal-bearing rock stratum being mined ($K_H \approx 1.43$) regardless of the level of gas emission.

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TOPICAL PROBLEMS OF RECLAMATION OF TECHNOGENICALLY DISTURBED LANDS OF KRYVBAS

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Abstract. Restoration of the landscape and land resources disturbed by mineral mining of is one of the major tasks of environmental protection in Kryvyi Rih iron ore basin. The article deals with the topical problem of reclamation arrangement and technology on disturbed areas. Ukraine's legislative provisions adapted to the European and world standards make the methodological background to the work which is aimed at considering the most efficient methods of reclamation of individual technogenic landscape formations. It is determined that restoration types, trends and methods should be chosen considering the technical state and length of functioning of a technogenic object, composition of rocks and the shape of the surface under reclamation.

Introduction.

Long-term activities of Kryvyi Rih ore mining enterprises, especially open pits of ore mining and concentrating combines (GOKs) have resulted in disturbance of large areas of urban and agricultural lands, landscape and hydrogeological conditions of the environment. At present, external waste dumps and tailing ponds of five Kryvyi Rih GOKs contain over 4 B m^3 of iron ore mining and concentration wastes; their total area exceeds 12 thousand ha (dumps - 5 thousand ha, tailing ponds - over 7 thousand ha). Within Kryvyi Rih iron ore basin, there are over 34 thousand ha of urban and suburban lands are technogenically disturbed and require reclamation, their restoration rates remain very low - 0.2-1,7% of the areas annually. Displacement and caving (crater formation) zones caused by underground mining make over 5 thousand ha of urban lands and require reclamation as well. Now, in the city's central part alone there are more than 600 ha of mined lands requiring immediate reclamation.

It is common practice in Ukraine that enterprises mine minerals under the authority of special licenses granting the right to temporary long-term (20-50 years) use of a subsurface site limited by a corresponding *mining lease*. Typically, under the authority of the obtained special permit for mining an economic entity leases land from local communities (*land allotment*) and develop a land management plan in which the layout of all facilities for economic activities on the allotted site is substantiated. Under Paragraph 4, Article 66 of the current Land Code of Ukraine, restoration (reclamation) of the disturbed land is obligatory on all post mining areas [1]. Under Paragraph 1a), Article 96 and the addendum to the Land Code provided for by the Law of Ukraine No. 1708-VI of November 5, 2009, a tenant is to "provide use of land according to the designated purpose and *bring it to the previous condition at their own expense* in case of illegal change of its relief…" (hereinafter the translation is ours). Paragraph 2, Article 166 of the Land Code states that "lands that have undergone changes in the structure of their relief, ecological state of soils and country rocks are subject to reclamation".

In view of the above, the problem of restoration of lands disturbed by mining is topical and indispensable for every mining enterprise of the city. As reclamation is an expensive process, the enterprises tend to apply the most rational and cheap technologies. The present article considers the most efficient methods of reclamation of individual technogenic landscape formations.

The current state of reclamation issues. Typically, reclamation of technogenically disturbed lands comprises two stages: mine-engineering and biological.

The main purpose of the mine-engineering (preparatory) reclamation (MER) stage is fulfilment of works on relief optimization and creation of surface areas suitable in their form for further economic use.

The character, volume and content of operations on the MER stage may vary greatly for each particular site of the area under reclamation due to the kind and degree of technogenic disturbances of the land surface. Within land allotment boundaries at iron ore underground mines, appearance of displacement zones and sinkholes (craters) at places of voids resulted from underground mining is the most common change of the landscape. In some cases, waste rocks are stockpiled on the surface forming heights – dumps. After geological processes of the surface caving stabilize, MER on the territory of underground mines' allotments provides for, first of all, filling of created sinkholes with waste rocks.

Surface mining operations result in creating open pits and external dumps of waste overburden or off-grade ores (in Kryvbas they are mostly dumps-stockpiles of so called oxidized ores). Besides, if surface operations are applied to mining lean ores with iron content of 17 to 45 %, they require concentration to reach marketable standards (60% and more) at specialized concentrating complexes which results in not only iron ore concentrate but also relatively barren concentration tailings. These wastes are typically placed on the surface as individual technogenic facilities – tailing ponds of an areal or a multilevel type. Areal tailing ponds are created in locations with large natural ravines or gullies and are relatively low hydraulic structures (under current concentration technologies tailings are transported and stockpiled in the pond in the form of slurry). Multilevel tailing ponds are high hydraulic structures resembling dumps. The above mentioned technogenic facilities (open pits and places of mining and concentration wastes accumulation) create huge areas of disturbed lands of hundreds of hectares. The existing open pits may reach the depth of 500 m and more, dumps and multilevel tailing ponds may rise 150-500 m above the surface. It should be noted that return of the GOKs' post-mining areas into the previous state is practically unachievable. That is why, reclamation of the mentioned territories can just be cosmetic and aimed at solving sanitary and hygienic tasks on reduction of contaminating impacts of the disturbed areas and technogenic facilities on the environment. At that, vertical walls of the open pits and steep slopes (over 18°) of "young" dumps and tailing ponds are practically inaccessible for any reclamation operations. Mine-engineering and followed-up biological reclamation can only be implemented on horizontal and slightly inclined areas.

Analysis of the available data shows that rocks of dumps, tailing ponds and backfill areas of displacement zones are conditionally suitable for biological reclamation and on the whole have a favorable influence on growth of trees and shrubs [6;7;8].

With the lapse of time, piled rocks of dumps and tailing ponds may become overgrown with vegetation. As in terms of latitudinal-zonal position, the area of Kryvyi Rih iron ore deposits is located in the south steppe (fescue-feather grass) sub-zone of the steppe zone, and in terms of geobotanical subdivision it belongs to Cis-Azov-Black Sea sub-province of the Euro-Asian steppe region, spontaneous formation processes possess traits of this sub-province.

The floristic investigation of slopes of open pits *Yuzhnyi* of the underground mine administration of the PJSC "ArcelorMittal Krivyi Rih" (PJSC AMKR) and *Severnyi* of GOK "Ukrmekhanobr" enable stating that ruderal vegetation (*mock cypress, bladder campion, horseweed*) is first to appear. With the lapse of time, the species composition gradually expands. Re-establishment of vegetational cover on the areas undergoes its first stage (weeds invasion) during the first two years.

Appearance of plants of other coenomorphs (e.g. *narrow-leaved bluegrass, coronilla*) is the first indication of the second stage of re-establishment - invasion of long-rooted plants. This stage results in not only increased phytodiversity but also enhanced soil-protective properties of vegetation. Totally, there are 33 overgrowing species of which 54% belongs to weed coenomorphs, 12% - to steppe weeds and 12% - to meadow weeds [9].

Observations show that areas filled with waste rocks 15 and more years ago are now intensively overgrowing. On this surface a multiple-aged uneven stand of trees has emerged. In some places the stand has closed with vegetation planted during previous steps of greening post mining areas.

Field inspections of iron ore dumps of Kryvbas show that the dumps without intentional planting are overgrown with such arboreal species as the apricot tree, the European white birch, the thorn, the Nanking cherry, the sour cherry, the Siberian elm, the oleaster pear, the common pear, the English oak, the Norway maple, the ash-leaved maple, the large-leaved lime, the black locust, the alycha, the Crimean pine, the Scotch pine, the white poplar, the Canadian poplar, the Lombardy poplar, the domesticated apple, the wild apple, the Fraxinus excelsior. Also there can be found bushes and lianas – the creeper, the Russian olive, the red raspberry, the common sea buckthorn, the dogberry, the black cherry, the bird cherry shrubs [10;11]. Seeds of these species have been brought to the dumps either by wind (anemochorously) or by animals (zoochorously).

The represented data demonstrate that the problem of reclamation of disturbed lands at Kryvyi Rih iron ore deposit can and should be solved through both intentional reclamation works and facilitation (stimulation) of self-revegetation processes in hard-to-reach areas of technogenic landscape formations.

One of the main conditions of successful reclamation of disturbed lands is sufficient financing as well as determining legal and economic issues of arranging and regulating reclamation works considering interests of the state and local authorities, a subsurface resources user and the population of the territories nearby developed deposits.

For Kryvyi Rih with its 8 of 11 Ukraine's largest iron ore mining and processing enterprises the problem of reclamation is one of the priorities in terms of environmental activities. The regional Program on solving Kryvbas environmental problems for 2011-2022 provides allocation of 24,9 bn UAH, including 3,96 bn UAH - for rational use of lands and reclamation of disturbed lands. At that, 324 mln UAH are only allocated direct for restoration of 116 ha of disturbed lands [12]. At the same time, roughly calculated area of lands disturbed by all mining enterprises of the city made 7301 ha (as of 2017). Expenses on reclamation of territories of these economic entities total 52,5 bn UAH for complete restoration of the topsoil (at 2018 values) [13]. It is quite obvious that centralized financing from the environment fund is not sufficient. According to the current legislation, reclamation of lands after mining is duty of the subsurface resources user [14]. But it is at this stage of the exploitation of the deposit that the problem of financing reclamation works appears as completion of mining and sales of products do not allow mining enterprises to use current cash flows. Due to this, to compensate for reclamation of disturbed lands, a subsurface resources user should have a special reserve fund. However, practically no user has a fund for this purpose at present. Nor are there completely developed and institutionalized norms of economic assessment of disturbed lands, normative and methodological framework for creating liquidation funds to finance reclamation [13].

Implementation of mine-engineering reclamation of disturbed lands of Kryvyi Rih region

The scope of these works depends on the actual character and the disturbance rate determined, in turn, by the technology previously applied. As mentioned above, disturbed lands within underground mines' allotments can include caved areas, craters, cave-ins, waste dumps of sinking operations, etc. Surface mining areas subject to reclamation include dead open pits, waste dumps, tailing ponds, settling ponds, etc.

Cavities of open pits and voids of the land surface caused by mine workings are the most difficult to reclaim. The amount and rates of mine-engineering reclamation of the mentioned areas are limited and will mostly depend on available land resources applicable to backfilling.

Waste materials resulted from sinking are predominantly used to backfill craters and voids. As volumes of cavities of these formations do not exceed 1-1.5 mln m^3, their backfilling is quite applicable. Yet, reclamation periods of these areas are limited by yearly output of waste materials produced, averaging 20-30 thousand m^3/year, and in case of considerable voids (of hundreds of thousands m^3), their reclamation may take decades. This process can be accelerated at the expense of auxiliary resources like overburden, yet it requires extra unproductive expenditures.

Reclamation of dead open pits is even more complicated. After de-activation of these open pits, no operations are undertaken including those of water pumping resulting in their gradual inundation and turning into water reservoirs - lagoons with granite banks. In Kryvyi Rih, there are some former granite open pits of this type on the territory of Zarechnyi micro-region (*Oktyabrskyi* open pit) and Karachuny (*KDZ* open pit). Open pit № 1 of Novokryvorizskyi GOK (up to 200 mln m3) is half filled with water and located within the PJSC AMKR allotment.

At present, this open pit is backfilled with waste materials from adjacent open pits № 2-bis and № 3 of the mentioned enterprise. The area is reclaimed by applying internal dumping using dump trucks and extraction-and-loading excavators of a dragline type (ЭШ-6/45) with a flexible bucket hanger. In Kryvyi Rih iron ore basin, stockpiling of waste rocks into dead open pits for their backfilling should be considered *one of the best* dumping methods requiring no extra areas for mining waste disposal.

The same reclamation technology is applied to a 3,89 ha site of the dead open pit of the mine named after Valyavko-Severnaya at the PJSC AMKR. This dead open pit of 2,4 mln m^3 was backfilled with overburden in 2016-2017. Hard rocks of the backfill were covered with a 1,5-2 m layer of conditionally fertile soil (loam). Currently, here biological reclamation (planting of trees and shrubs) is undertaken to create a green area of general use. The mentioned reclamation targets exclusion of the reclaimed area from the GOK's allotment list and its transition to the city land fund.

Open pits of the existing Kryvyi Rih GOKs have excavations ranging from 300 mln m^3 (Petrovo open pit of CGOK) to 4,2 bln m^3 (InGOK open pit) that cover the surface area from 270 to 650 ha. Backfilling of such huge open pits is possible only at the expense of overburden or off-grade ores from external waste dumps. However, the return of millions of cubic meters of rocks requires almost the same expenditures as their removal from the open pit, i.e. billions of hryvnas. For this reason, only *a special fund* can become the source of reclamation expenses for this type of operations at open pits. The fund should be financed by means of depreciation payments during the whole operational cycle of an open pit, these payments being included into the cost of end products.

The mentioned approach can be implemented for newly created open pits of the same economic entity as existing Kryvyi Rih open pits created in the 1950s-1960s and privatized by current owners have no reclamation funds at all. For this reason, there is no possibility of backfilling currently mined open pits of the region because of considerable expected costs of such operations which are unproductive. Most likely, huge craters from dead open pits of Kryvyi Rih GOKs will turn into silent ugly reminders of the frontal economy.

Preliminary calculations indicate efficiency of mine-engineering reclamation of dead open pits by backfilling in case of their relative low volume of up to 25-30 mln m^3 and availability of corresponding amounts of waste rocks. For larger open pits, it would be more efficient to form initial *internal waste dumps* not only for overburden, but for off-grade (oxidized) rocks as well. This very approach will provide an opportunity to save hundreds of hectares from mining waste disposal (waste dumps) and reduce expenditure for reclamation of dead open pits. It would be quite expedient *to enact into law and order* all "old" and operating Kryvyi Rih GOKs to start developing relevant technologies and designs of stockpiling re-processed waste rocks and oxidized ores in particular into internal waste dumps only, while further development of external waste dumps both in height and width should be banned. Development of technologies of stockpiling ore concentration tailings inside the mined-out part of open pits is also quite promising.

The technology of internal dumping should predominantly depend on the technology and advance of open pit mining operations. Advancing - longitudinal mining creates the most favorable conditions for internal dumping as there is a mined-out space that can be reclaimed (backfilled) next to the pit walls in the opposite to mining direction. At deeper open pits, there can be circular haul roads along the perimeter of an open pit. That is why, backfilling should be started below the lowest road (usually by the overpass method), backfilled sites being used for constructing new roads.

For mining deep open pits, internal dumping parameters are specified by individual mining designs and should envisage step-by-step backfilling of mined-out benches. If waste materials of ore mining include not only overburden rocks, but also off-grade ores (for example, oxidized ores which are potential iron ore materials), an internal dump should be formed by differentiating a "cultural" zone (a storage for potential raw materials to be re-processed) and a waste disposal zone.

In mine-engineering reclamation of the open pits backfilled to the height of their final contours and multilevel tailing ponds that are not going to be used as raw material sources for backfilling and are suspended as artificial hills and mountains, a technogenic relief as close to the natural one as possible is formed, it being safe and suitable as to its geometrical parameters and shape quality for people and animals. This stage should include the following operations: sanitation of disturbed lands to remove industrial waste, temporary facilities and rubbish; development of the vertical and finishing land leveling of the disturbed surface to meet safety standards for environmental objects and protect them from linear and plane weathering; fixation of unstable surfaces of weatherable techno genic formations by engineering and chemical means.

Vertical land leveling should be applied to dump slopes of overburden rocks (flattened to meet angles of natural slopes of 30-37°) and excavation terraces at open pits. Berms of overburden dumps are to be subject to fragmental cleaning and caving; slope edges are to be subject to wall control followed by downward caving along safety banks; horizontal parts of the berms are to be subject tofinishing land leveling.

After leveling the areas to be reclaimed, there are conducted final operations of mine-engineering reclamation, namely, earthing of all horizontal and slightly inclined (up to 18°) sites with 1.5-2 m layers of conditionally fertile soils (loams) and 0.3 m layers of fertile soils (chernozem). It should be noted that the fertile layer should be applied to the leveled surface after its natural shrinkage only under the action of precipitations (in 1-2 years at the earliest). The material for earthing is stockpiled (according to the standards of GOST 17.5.3.06.85) soft overburden rocks.

In engineering reclamation of the 'beach' surface of tailing ponds containing fine fraction slurry, no earthing is envisaged as only banks are cut and leveled because the surface itself is potentially suitable for biological reclamation.

Implementation of biological reclamation of Kryvyi Rih disturbed lands

According to 17.5.3.04-835302-85 GOST (with N1 changed and approved in September, 1986 (ИУС 11-86)) "Nature protection. Lands. General requirements to land reclamation", technogenically disturbed lands should be predominantly turned into arable fields and other farm lands. Yet, reclamation of lands disturbed by the activity of underground mines and concentrating plants of Kryvyi Rih iron ore basin located mostly within the city area is not reasonable from the agricultural point of view and does not comply with the General development plan of the city. For this reason, the sanitary-hygienic trend of biological reclamation is implemented which envisages fixing surface (dusting) layers of soil by planting trees, shrubs and perennial herbs.

Interconnection of soil and plants is one of the key problems to be solved by biological reclamation. Mineral mining involves displacement of geological strata when underlying rocks different in their granulometric and chemical composition from zone surface soils come up to the surface.

Investigations indicate that *waste rocks* resulted from sinking operations and *stripping rocks* of open pit mining are noted for almost the same characteristics. As for its mineralogy, this substrate is presented by granite (up to 2%), jaspilite martite (up to 21%), hematite-martite hornstone (up to 42%), quarts-sericite-chlorite shales, hematite talcose shale (up to 33,5%) and loams (up to 1,5%). In granulometric composition, coarse fractions of lumps, gravel and crushed rocks (21-30%) prevail, while sand and dust make 12% and 8% respectively.

As for their chemical composition, rocks are characterized by a great amount of silica, soluble iron and metal oxides. Concentration of ions of heavy and toxic metals and parameters of their natural radioactivity indicate low toxicity of single samples of these rocks related to the fourth class of hazard and toxicity. In the present form, the given elements cannot be absorbed by plants and have phytotoxic properties. Moreover, silicon, aluminium, calcium, magnesium, iron and other elements are microelements that are vital for plants. Reaction of the aqueous extract of rocks is close to neutral. The rocks are of medium and high absorption rate.

It is worth noting that at dumps and multilevel tailing ponds subject to biological reclamation and surrounding landscape formations above the daylight surface, an artificial hydrological mode characterized by absence of underground aquifers is formed. Water content of the surface layer is supplied only by precipitations and intensity of their evaporation. Considering Kryvyi Rih region's climate with its scarce rains and high air temperature in summertime, it can be stated that there are created quite dry weather conditions for plants, this fact being taken account of while choosing plant species.

Thus, plants occurring on dump rocks find themselves in changed edaphic conditions relative to the earth's surface ones. That is why, successful artificial planting for biological reclamation requires choice of species capable of surviving under certain specific conditions. To achieve this, the following steps are recommended:

- privileging the plant species naturally growing in natural biotopes adjacent to reclamation zones;
- using plants with evident adaptive properties in “poor” soils, i.e. those growing and fruiting intensively;
- using oligonitrophilic and drought-resistant plants.

While choosing plant species, it is necessary to consider biochemical and specific compatibility of arboreal and dumetosous species suitable for planting in the same area.

The following trees are recommended to plant in hard rocks: the Norway marple (*Acer platanoides*), the apricot tree (*Prúnus armeniáca*), the Tatarian maple (*Acer_tataricum L.*), the Chinese elm or an English elm (*Ulmus parvifolia*), the Lombardy poplar (*Populus nigra pyramidális*), the Scotch pine (*Pínus sylvéstris*), the Crimean pine (*Pinus nigra subsp. pallasiana*), the Fraxinus excelsior (*Fraxinus excelsior*), the angustifoliate olive (*Elaeagnus angustifolia*), the alycha (*Prunus divaricata)*. The following species of bushes are recommended for these conditions: the Russian olive **(***Elaeágnus angustifólia*)*,* the common privet (*Ligustrum vulgare*)*,* the common sea buckhorn (*Hippophaë rhamnoidesa*)*,* the tamarix (*Tamarix tetranda*), the cinnamon rose (*Rōsa majalis*)*,* the hawthorn deceptive (*Crataegus fallacina*), the dogberry or the bloodtwig dogwood (*Cornus sanquinea L.*), bird cherry shrubs or the Mahaleb cherry (*Padellus magaleb*), the indigo bush (*Amorpha fruticosa*).

Biological reclamation of extremely steep slopes at dumps and multilevel tailing ponds of mining and concentrating combines is one of urgent problems of the landscape environment of Kryvyi Rih region. Vast areas of bare slopes of these formations are powerful dusting sources under the action of wind that causes intensive pollution of the atmospheric air in surrounding areas. This issue is especially evident at dumps and tailing ponds of the southern group of mining and concentrating combines (GOKs) of Kryvyi Rih where these technogenic formations take hundreds of hectors and are up to 150-200 m high.

The resulted blowing erosion dust from wind plumes at dump slopes spread over 7-8 km covering dwellings and agricultural lands with a dust mantle. The dust from dumps and tailing ponds in its chemical and mineralogical composition is known to contains not only quarts-containing compounds, but also elements of toxic substances of the 1st and 2nd hazard classes (compounds of lead, zinc, cadmium, manganese, iron, etc.). There are evident hazards for people's health and the whole environment resulted from dust of the mentioned composition in the surface zone. Thus, solution of the problem of reducing dust content on the slopes of technogenic objects of Kryvyi Rih GOKs is essential for ore mining greening.

Biological reclamation operations conducted by iron ore mining enterprises involve application of fertile substrates (poor chernozem, loams and their mixtures) of 40-50 cm thick to cover dump slopes and crater-formation zones followed by planting. This method should consider slope angles. If they exceed 18^o, chernozem or loam earthing becomes inefficient as potentially fertile substrates are unable to be fixed on these slopes and are washed off with precipitations.

Considering the fact that all slopes of operating GOKs' dumps have natural dumping angles (35^0 and over), application of this greening method is almost impossible. The method is also cost-ineffective - 1 ha of the slope surface of 0.4 m thick requires 4-4.5 thousand m^3 of soil to be delivered and bulldozed. Uniform soil covering of slopes can be reached only with the slope height of under 20-25 m. Earthing of slopes also requires dozens of hundreds of tons of scarce conditionally fertile soils.

Application of manual hole digging technologies to planting the slopes of fine fraction using ropes is also of little efficiency and expensive for 20-35° slopes. Subsequent planting and watering becomes very complicated and sometimes even unfeasible. The mentioned operations at coarse-grained hard rocks conducted at "young" slopes (of 20-25 years old) of operating GOKs and underground mines are not permissible as it becomes hazardous for employees' health with safety standards violated.

There is a patent offering a hydraulic fill of seeds together with the soil substrate mixture onto the slopes of 35-40°. Yet, this method cannot be widely applied because of absence of required standard equipment suggested by its authors.

In 1989, the employees of our Institute tested the technology of slope reclamation by means of an updated hydromonitor at InGOK dumps [16]. Herb seeds were put into the water container of the hydromonitor and applied onto slopes with a hydro-jet. However, the method did not gain broader acceptance because of absence of the required equipment, the limited range of action of the hydromonitor (up to 20 m) and very low establishment of plant seeds at rocky slopes (5-7%) as well as massive mortality of vegetative plants without watering during the first year.

To increase plant establishment, there were made efforts to encapsulate seeds with life-giving substrate before their applying onto dump slopes by the hydro-method. Yet, this method failed to be developed because there is no standard equipment to perform such operations which are also not cost-effective.

In compliance with the above-mentioned, we consider that natural plant expansion to remain the most effective method of dump greening. Many researchers including those of Kryvyi Rih Botany Garden of the Ukraine's National Academy of Sciences confirm the advantage of this approach to dump reclamation at ore mining enterprises [8]. Researches indicate that dumps are capable of being overgrown with trees, shrubs and herbs within 10-15 years. There are 32 species of self-planted trees and shrubs at Kryvyi Rih open pit dumps reclaimed intentionally, *the Lombardy poplar, the white poplar* and *the European white birch* being the most widely spread species.

Thus, natural plant expansion is the most efficient method among five known greening methods for GOK dumps and coal mine wastes from both biological and economic points of view. Some methods of encouraging and accelerating plant expansion can be considered useful and recommended like planting 2-3 rows of high-yielding trees or shrubs at the distance of up to 1-1.5 m from the upper edge of the slope.

After coming into fruiting, plants start active colonization of dump slopes under the action of wind, precipitations and birds. Plant expansion can be enhanced by manual dissemination of seeds from slope edges (1kg of seeds consumed per 10 lin.m of a site). Yet, this method is hazardous when employees come too close to the slope edge (up to 1 m), so the dissemination zone usually makes a few meters from the upper part of the earthfill. Application of flying apparatuses (for example, drones) can make seed dissemination along dump slopes entirely safe. The flight courses of drones along objects and dissemination intensity can be easily programmed.

The biological stage of reclamation of plant expansion sites can be considered finished when vegetation density at slopes makes not less than 65% of the surface and this process can be continued.

Horizontal sites of dumps, filled open pits and tailing ponds at the city outskirts can also be subject to forestry-engineering and agricultural reclamation.

In forestry-engineering reclamation of dumped hard rocks, it is important to create a potentially fertile soil layer which is sufficient for developing a tree root system. Ideally, this layer comprises two components - the continuous covering layer of quaternary loams or loess soils of 1-1.5 m thick and a 0.2-0.4 m covering layer of humus-containing substrate (chernozem) in the amount of up to 3000 m^3 per 1 ha. If chernozem is not available, *a local hole-digging method* of planting can be applied which enables saving both fertile substrates and costs required for reclamation.

In this case, a hole of up to 0.3×0.3×0.35 m is dug directly in the available substrate including weathered hard rocks of the dump, while the fertile soil (chernozem) of 10-15 dm^3 together with a combined nitrogen-phosphoric fertilizer (up to 100 g per root) is put directly into the hole while planting. The plants are watered (10-15 l). The rest of the hole is filled with fine fractions (up to 3 cm in diameter) of rocks or loam.

Agricultural reclamation envisages turning disturbed lands into farm fields, so the brought surface fertile layer of soil should be not less than 0.5 m thick. Initial agrochemical conditions of the layer are essential including the content of humus, nitrogen-phosphor-potassium compound and microelements. If the soil is lean, some *land improvement* steps are taken including improvement of the lean soil structure by its mechanical cultivation, improvement of chemical conditions to make them suitable for plants by using fertilizers and chemical ameliorators and biological activation by introducing humus and soil bacteria, mulching by organic and non-organic materials.

Conclusions

The presented material enables the following conclusions:

1. The regional degradation of lithosphere during the 140-year

operation of Kryvyi Rih iron ore basin resulted from application of frontal economy principles has long reached **the point of no return**, this process being escalated.

2. Restoration of disturbed lands and landscape structures of Kryvyi Rih region is one of the most urgent problems of creating a healthier environment.

3. Neglected issues of reclamation of technogenic disturbed lands of dead and operating iron ore mines and objects of mining and concentrating combines require huge investments which are not currently available.

4. Urgent implementation of Regulations on obligatory accumulation of special funds (payments included into mineral mining costs) intended for reclamation of disturbed lands and landscape structures by all mining enterprises exploiting mineral deposits is required.

5. Costs necessary to form a reclamation fund should be determined as obligatory for all mining designs, this being reflected in regulatory design documents (DBN A.2.2-3:2014) and controlled by examining design plans and specifications.

6. Open pits, dumps and tailing ponds of mining and concentrating combines are the most urgent problem within the structure of technogenic disturbed lands and landscapes. Application of internal dumping can be considered as a promising method of reducing negative effects of mining on the earth's surface. The local hole-digging and advanced methods of activated plant expansion at dump slopes are the most cost-effective for greening external dumps and multilevel tailing ponds.

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MODELLING OF SOLID LUMPY MATERIALS DESTRUCTION IN DISINTEGRATORS BASED ON THE LOGARITHMIC GAMMA DISTRIBUTION

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Abstract

The subject of research is the distribution of disintegrator product on size of particles. The goal is to show, that the distribution on logarithmic size of product while crushing a narrow size class corresponds to gamma distribution**.** The main assumption has been took, that the probability of a separate piece destruction is equal for all the pieces and doesn't depend on their size. Experimental research on the granite crushing by press plates and also on the limestone impact destruction have been proved the accuracy of developed theoretical formula. But the tests of silicon carbide grinding in a vibrational mill have shown high divergence between experimental and analytical data. Obtained results can be used for modelling of destruction process for disintegrators subjecting rocks to from one to several crushing acts, both for static and dynamic loadings. This will allow to improve the techniques for calculation of disintegrators operating parts parameters and to decrease the consumption of energy.

Introduction

Processes of mining rocks disintegration are of high importance in mineral dressing. Huge amount of energy is spent for these purposes. It is up to 20% of processing industry power consumption, according to various estimations [1]. Many scientific works are devoted to enhancing of the mining equipment energy efficiency [2, 3].

Nevertheless, the development of energy efficient mechanical equipment and technologies for crushing and grinding is one of the most important questions of today.

One of the ways to reduce the energy consumption is absence of overgrinding of lumpy material during technological operation, that means not to exceed the reduction ratio over the sufficient value. In order to setup the equipment in a right way, it is necessary to fulfill modelling of the particle size characteristic changing dynamics for the disintegrator product during its passing through the machine's operation area. Firstly, it eliminates the need to consider all the process as "black box" and to choose the operation mode for every change in working conditions by expensive empirical way. Secondly, in some cases, there are restrictions on usable sieve range for the particle size distribution quick analysis. Also, it is hard to determine the finest size content while having high reduction ratio. The finest particles goes under the smallest sieve, and their formation requires the greatest part of energy.

So, it is expediently to develop the analytical dependences, that will allow to predict the result of mining rocks disintegration with high reduction ratio, based the disintegration product analysis data obtained for the case of low reduction ratio.

State of Question and Research Problem

The necessity of particle size distribution for destruction product prognostication requires its description by so called the size characteristics equations. Today many types of equations describing the particle size distribution curves are known, such as Kolmogorov, Goden-Andreev, Rozin-Rammler, polynomial equation and some others [4, 5]. At our opinion, the common feature of these equations is "ascertaining of the fact" of availability of certain particle size distribution at definite moment and at definite spot of the disintegrator operation area. They are usable just for convenient description of the disintegration splinters size distribution and, frequently, don't have any physical sense, for example, Veinig polynomial equation includes only coefficients from known mathematical formula. Also, the usage of particle size logarithmic scale should be noted, that is mainly caused by the geometrical similarity of lumps destruction into smaller and smaller splinters in consecutive stages. It is shown, for example, in the work by A.N. Kolmogorov [6], that the distribution of particles on the logarithm of their characteristic linear size approaches to the normal distribution dependence at unlimited time of processing. This equation is obtained for the common case of the initial material particle size distribution and, thus, it cannot define the intermediate size distributions.

More complex approach is applied in the work [7], where concepts of the destruction function and the selection function are introduced for fuller description of the disintegration process. The destruction function is responsible for splinters size distribution in the elementary act of material lump destruction. The selection function relates the lump destruction probability with its size and the size of neighboring lumps. Two mentioned functions require also the empirical determination for each process stage, that doesn't simplify the problem of the product particle size distribution prognostication.

It makes sense to use the analogies of probabilistic approach for description of the mining rocks destruction, mentioned in the work [8], in order to calculate the particle size description. Available distribution of stresses and deformations depends strongly on composition of the deformed material characteristics and the loading zone configuration [9-12]. Also, the nature of destruction is sufficiently influenced by the speed of loading application [13].

The friction forces between the material lumps, depending sufficiently on the liquid phase availability, make an impact on the technological process while processing of great set of lumps [14, 15].

So, it is possible to formulate the main problem having to be solved in this article: it is required to substantiate such a size distribution of disintegration splinters, whose parameters will directly reflect the physical peculiarities of disintegration process and depend on the characteristic factors, such as the reduction ratio, the specific surface gain, the energy consumption etc.

Analytical substantiation of splinters size distribution

As mentioned above, the lumps disintegration process consists of multiplied single disintegration acts, accompanied by formation of many geometrically similar splinters, also featured by insignificant variability of the destruction function in coordinates of the ratio of initial lump size and its splinters size values. Deviations from the mentioned algorithm may be explained by the peculiarities of some mining rocks formation and structure, for example, as it was made in the work [16]. These deviations make an influence only in the case of high total reduction ratio of a crushed material.

Thus, one of the properties of size distribution functions must be their infinite divisibility. It means, that the function is presented as a sum of any number of independent composed functions, having the similar distribution type. It corresponds to the physical essence of disintegration process.

Let suppose, that it is required to fulfill the conditional "elementary" crushing act of the solid lumps having very narrow size dispersion and the initial average linear characteristic size of d_0. Here, we assume, that its splinters characteristic sizes are distributed on narrow size ranges

$$d_1,\ d_2 \ldots d_n \ldots d_\infty \to 0,\ d_0 > d_1 > d_2 > \ldots > d_n, \tag{1}$$

and, for the calculation convenience, having the value of minimal module of neighboring sieves for the splinters screening equal to

$$m_0 = \frac{d_0}{d_1} = \frac{d_1}{d_2} = \ldots \frac{d_{n-1}}{d_n}. \tag{2}$$

Let's change the splinters size scale to the logarithmic one, using expression for the reduction ratio logarithm of n-size range ("logarithmic size" for short)

$$\delta_n = \lg \frac{d_0}{d_n}. \tag{3}$$

Here, the format of δ is chosen in such a way, that it is rising and having positive value during grinding process, that is $\delta \in [0;\infty)$.

Further, we will calculate the value of logarithmic size minimal step, taking into consideration Eq. (2)

$$\Delta = \lg m_0 = \lg \frac{d_{n-1}}{d_n}. \tag{4}$$

We suppose, that being based on the infinite divisibility principle, mentioned above, the splinters "plus-yield" (part of material passing not through the sieve of certain mesh size) on the sieves of mesh size ranges of d_n and d_{n-1} will be correspond to each other at elementary disintegration act according to the formula

$$1-\beta(d_n)=\lambda\cdot[1-\beta(d_{n-1})], \tag{5}$$

or

$$\frac{d\beta}{d\delta}=\lambda\cdot(1-\beta), \tag{6}$$

where λ is a proportionality coefficient.

The solution of the last equation having the zero initial condition is the expression

$$\beta=1-\exp(-\lambda\cdot\delta), \tag{7}$$

corresponding to the exponential distribution law.

Its density function is expressed by the formula

$$\frac{d\beta}{d\delta}=\lambda\cdot\exp(-\lambda\cdot\delta). \tag{8}$$

We assume, that the size distribution density function for each d_m narrow size range of splinters, being available at the previous act, will be described by Eq. (8), but it should be noticed, that the value of δ must be counted from the current size

$$\delta_{n,m}=\lg\frac{d_m}{d_n},\ n\geq m. \tag{9}$$

Thus, any amount of material of d_m size range after any destruction act will have such size distribution density function

$$\frac{d\beta}{d\delta}=\lambda\cdot\exp\left[-\lambda\left(\delta-\delta_{n,m}\right)\right],\ \delta\geq\delta_{n,m}. \tag{10}$$

Here, the value of β shows the "plus-yield" of destruction splinters just within such amount of material, which corresponds to the size range d_m in the given destruction act.

It is necessary to summarize a-times the destruction results of every material portion for every certain size range according to Eq. (10), in order to determine the total size distribution function after destruction act with number a.

Let's note, without providing here the formula conclusion, that the sum of functions having exponential distribution will have gamma distribution, as it is known from probability theory and mathematical statistics [17]. The distribution density function, according to gamma distribution, is described by expression

$$\frac{d\beta}{d\delta} = \frac{\lambda^a}{\Gamma(a)} \delta^{a-1} \cdot \exp(-\lambda \cdot \delta), \tag{11}$$

and the integrated distribution function is

$$\beta = \lambda^a \cdot \Gamma(a)^{-1} \cdot \int_0^\delta \delta^{a-1} \cdot \exp(-\lambda \cdot \delta) d\delta, \tag{12}$$

where, in this case, λ is the scale coefficient, depending on the material structure peculiarities and the way of loading application; a is the shape coefficient being equal to the number of conditional disintegration acts.

Here, we will have the exponential distribution function if a=1, or the normal distribution function if $a\rightarrow\infty$, that corresponds to the conclusions of A.N. Kolmogorov [6].

Experimental research by press unit

Granite of size 6-12 mm has been disintegrated by approaching to each other of two parallel steel plates, without side restrictions for the material. The total “plus-yield” curves of crushing products are shown in Fig. 1.

The curve 1 of size distribution function has been taken as the characteristic of basic destruction act. It has been approximated by the integral function of logarithmic gamma distribution (12). Then, the parameters λ and $a=a_1$ has been determined by the least squares method.

The size distribution characteristics 2,3 and 4 have been also approximated by the equation (12), but it has been fixed λ=const and only corresponding empirical coefficients a_2, a_3 and a_4 have been determined.

The given data prove good accuracy of experimental results approximation by the curves of logarithmic gamma distribution, but the approximation error rises together with the reduction ratio increasing.

It may be explained by deviations from the accepted assumption, that the splinters destruction probabilities are independent from their sizes and the condition of other surrounding splinters.

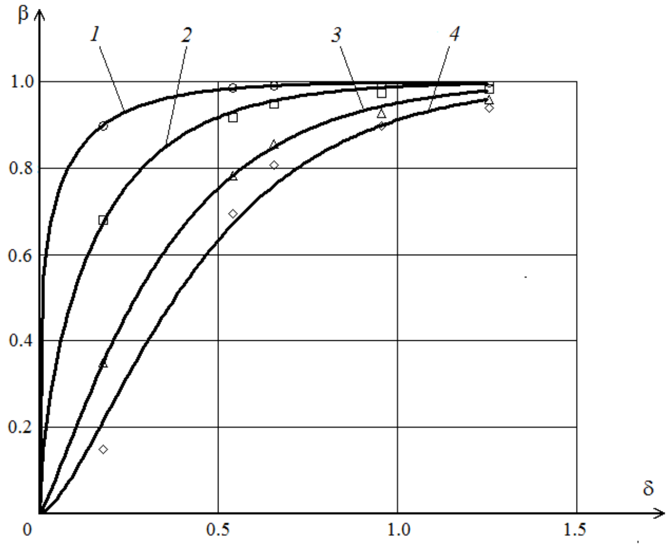


Fig. 1. Total "plus-yield" values for the crushed granite products of range 6-12 mm and the press force value (analytical/experimental values): 1/circles are for 4,4 kN; 2/squares are for 9,8 kN; 3/triangles are for 19 kN; 4/rhombus are for 28,9 kN

Also, the distribution density curves as functions of the material logarithmic size are given in Fig. 2.1 and 2.2. The plots show, that the coincidence of analytical curves with experimental data is satisfactory.

One of the gamma distribution properties is direct ratio of the dispersion and the mathematical expectation

$$M_\delta = \lambda \cdot D_\delta , \tag{13}$$

where M_δ - mathematical expectation; D_δ - dispersion.

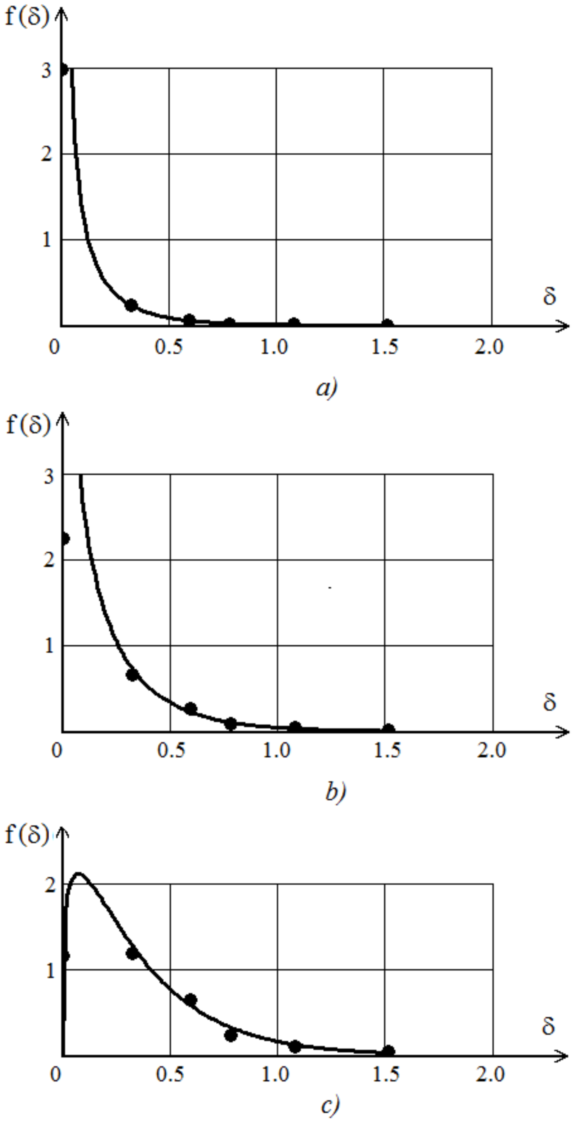


Fig. 2.1. Distribution density for the crushed granite products of range 6-12 mm and the press force value: *a* - 4,4 kN; *b* - 9,8 kN; *c* - 19 kN (lines are analytical curves, points are experimental data)

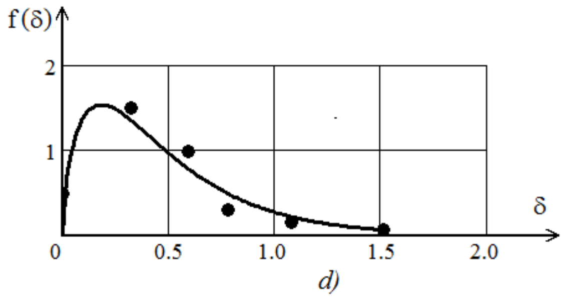


Fig. 2.2. Distribution density for the crushed granite products of range 6-12 mm and the press force value: *d* - 28,9 kN (lines are analytical curves, points are experimental data)

The data processing results for the granite disintegration by press unit, having various material mass between crushing plates and, correspondingly, various intensity of loading influence on the material, are provided in Fig. 3, in order to illustrate the mentioned gamma distribution property.

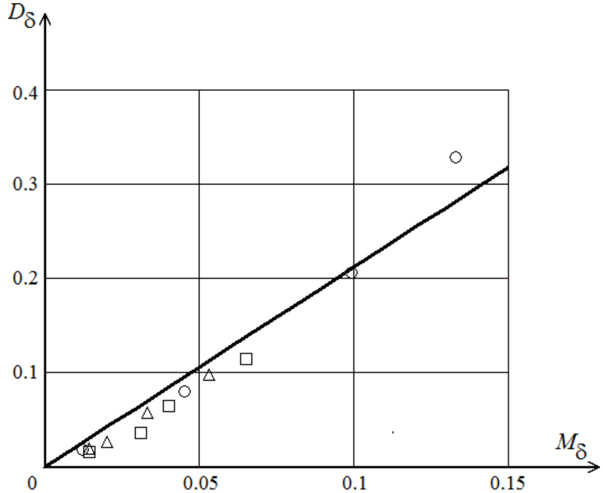


Fig. 3. Dependence of the distribution dispersion from its mathematical expectation

Here, circles are for granite portion mass 32 g, squares are for mass 64 g, triangles are for mass 128 g, solid line is the linear approximation

The required dependence in close to the linear one, as the last figure shows. Bigger error for higher reduction ratio can be explained by the limitation of sieves number for size distribution analysis of the finest material size range, having low partial yield but, on the contrary, the biggest logarithmic size.

Experimental research by vibrational mill

Initial size range has been 0,4-0,63 mm. The grinding of material portions of the same mass and various processing time values has been carried out. The total “plus-yield” curves for the grinding products are shown in Fig. 4.

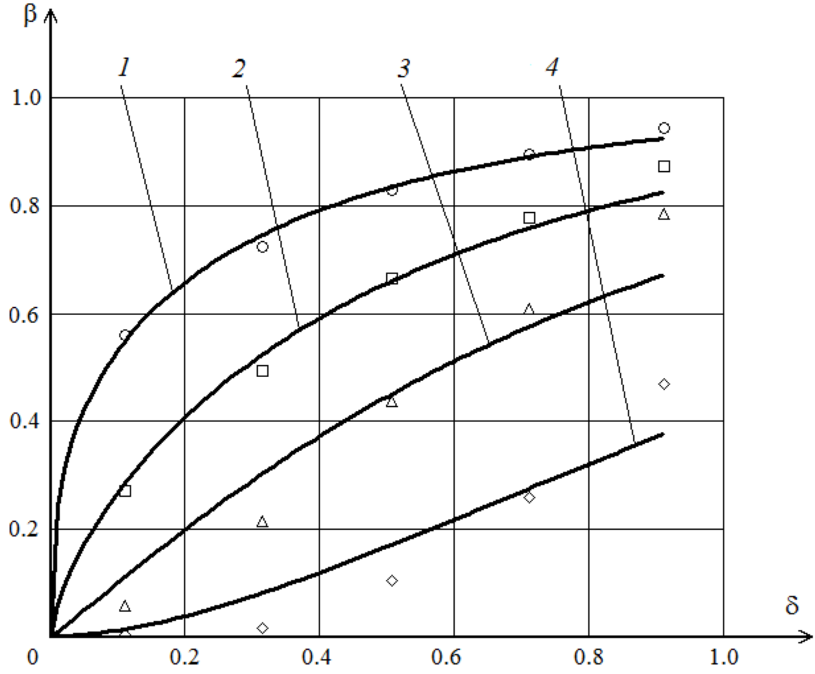


Fig. 4. Total “plus-yield” values for the ground silicon carbide products of range 0,4-0,63 mm and the processing time value

Here (analytical/experimental values): 1/circles are for 80 s; 2/squares are for 165 s; 3/triangles are for 250 s; 4/rhombus are for 400 s.

Also, the approximation lines corresponding to Eq. (11) are given. In general, the procedure has been identical to the case of granite crushing tests.

The density functions for the silicon carbide distribution on logarithmic size are given in Fig. 5.1 and 5.2.

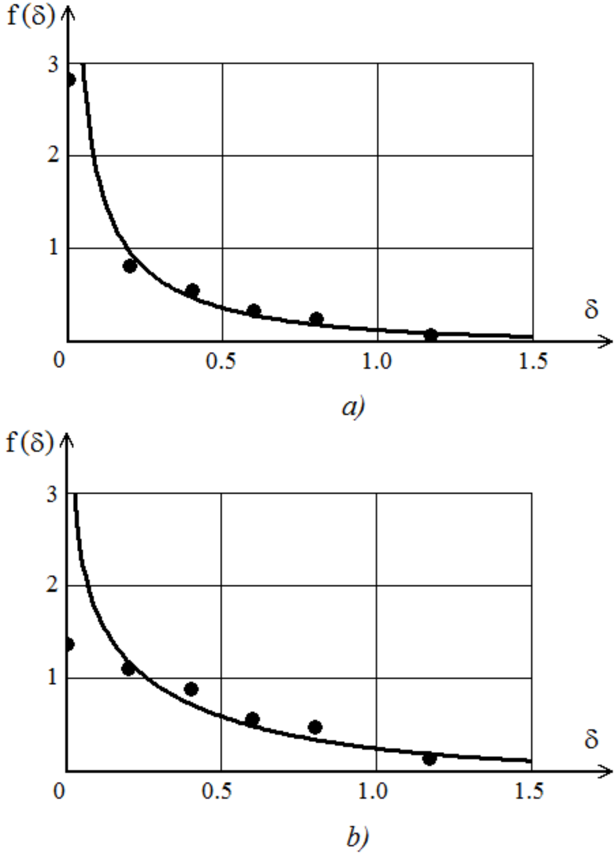


Fig. 5.1. Distribution density for the ground silicon carbide products of range 0.4-0.63 mm and the processing time value: *a* - 80 s; *b* - 165 s. (lines are analytical curves, points are experimental data)

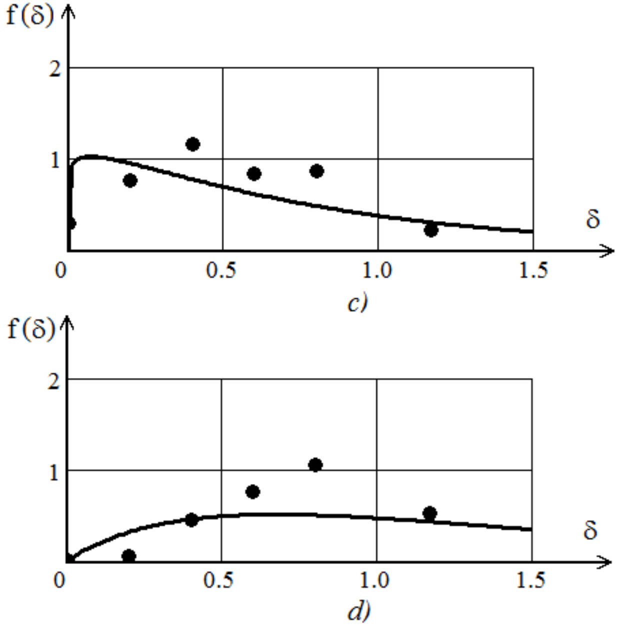


Fig. 5.2. Distribution density for the ground silicon carbide products of range 0.4-0.63 mm and the processing time value: *c* - 250 s; d - 400 s. (lines are analytical curves, points are experimental data)

The analysis of "*c*" and "*d*" plots shows, that the material size distribution is still adequately described by Eq. (11) with insignificant deviations for the biggest size splinters, if the reduction ratio is low. But in the case of higher reduction ratio values, the character of analytical and experimental curves differs strongly. This fact confirms the selection function action signs, where the principle of independence of the lumps destruction results and probability from their size is violated.

Research of crushing by free impact

This experiment was made earlier and described in work [18]. The laboratory unit was able to accelerate the limestone balls of diameter 30-35 mm to the speed of 20-60 m/s, that corresponds to the operation conditions of impact crushers, and to crush them by interaction with solid motionless barrier. The total "plus-yield" dependences for the limestone impact crushing products, approximated according to Eq. (11) and similar to mentioned above tests, are given in Fig. 6.

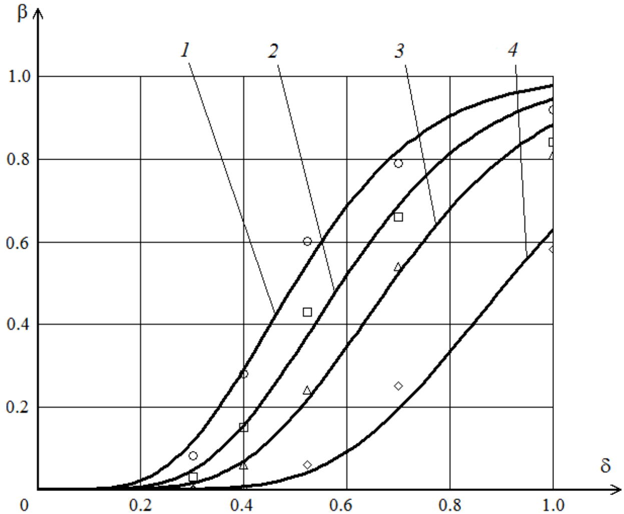
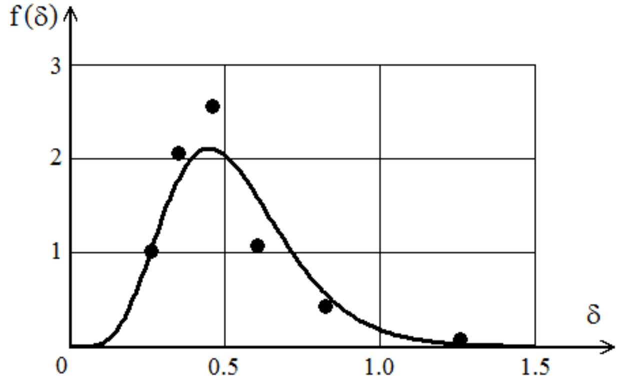


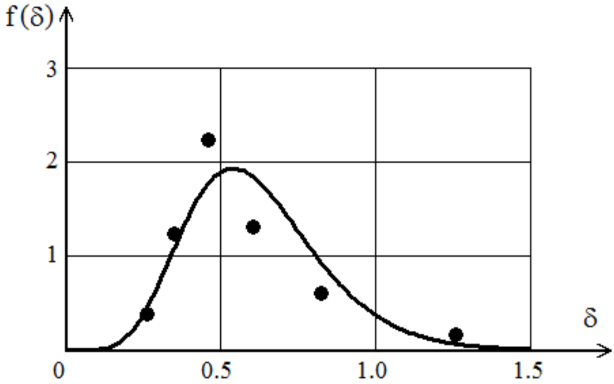
Fig. 6. Total “plus-yield” values for the limestone balls of diameter 30-35 mm crushed product and the impact speed value (analytical/experimental values): 1/circles are for 20 m/s; 2/squares are for 30 m/s; 3/triangles are for 40 m/s; 4/rhombus are for 60 m/s

The density functions for the limestone ball splinters distribution on logarithmic size are shown in Fig. 7.1 and 7.2.

One should note satisfactory coincidence of the analytical and the experimental values for the case of free impact crushing. There is also some approximation error, and it doesn’t rise together with the reduction ratio increasing, but it even decreases, in contrast to two previous tests. On the one hand, the enhanced error for lower reduction ratio values can be explained by failure to follow the condition of geometrical similarity of the splinters shape to the initial ball shape. On the other hand, the model developed in the article describes not bad the size distribution of impact crusher product. In this case, the splinters are formed by multiplied geometrically similar cracks branching, and the splinters shape is stabilized with process development.



a)



b)

Fig. 7.1. Distribution density for the limestone balls of diameter 30-35 mm crushed product and the impact speed value: *a* - 20 m/s; *b* - 30 m/s; (lines are analytical curves, points are experimental data)

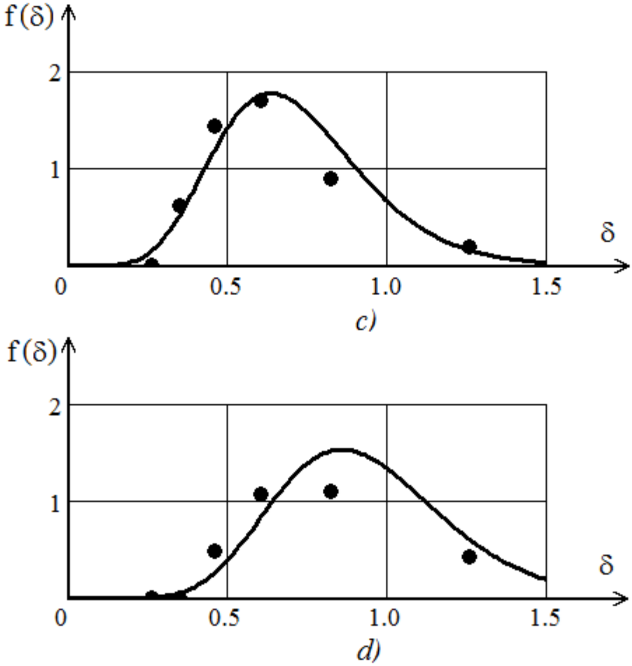


Fig. 7.2. Distribution density for the limestone balls of diameter 30-35 mm crushed product and the impact speed value: *c* - 40 m/s; *d* - 60 m/s (lines are analytical curves, points are experimental data)

In general, the logarithmic gamma distribution approaches well for the description of single influence of disintegrator on the ground material, irrespective of that, whether it is quasistatic compression of material between two plates or impact destruction by contact interaction with one surface. But, in the case of multiplied influences, as it has been in the vertical vibrational mill, the model deviation are sufficient because of the lumps selection function impact on the product composition.

Identification of logarithmic gamma distribution coefficients

The value of scale coefficient λ of logarithmic gamma distribution has been taken as constant during approximation of experimental data for each set of tests. It is equivalently to the assumption, that this coefficient depends just on the way of loading application and the material structure. You can see, that the most part of experimental data is approximated satisfactory by the curves corresponding to the logarithmic gamma distribution. It proves the validity of given assumption.

The experimental dependences of the shape coefficient value of logarithmic gamma distribution curves for the destruction splinters, calculated by the least squares method, on the achieved reduction ratio of material portion are presented in Fig. 8 in semi-logarithmic coordinates. The analysis of plots testifies, that the set of curves can be described with adequate accuracy by the following equation:

$$a = b \cdot \lg(i-1), \qquad (14)$$

where b is the coefficient depending on the destructed material properties and the way of loading application to it.

So, characteristics of the logarithmic gamma distribution are directly connected with the current reduction ratio of material during its disintegration and demand experimental determination of two coefficients λ and b being constant for the given material and way of its destruction.

Conclusions

The curves of the disintegrators product size distribution are described by gamma distribution on the destruction splinters logarithmic size. Here, the value of distribution scale coefficient is constant during disintegration process, and the distribution shape coefficient if directly proportional to the logarithm of material reduction ratio increment.

Fulfilled tests on the granite of size 6-12 mm crushing by a press unit between two parallel plates, together with tests on the free impact destruction of limestone balls of diameter 30-35 mm have showed good coincidence with proposed analytical model.

The processing of experimental results on the silicon carbide of size 0,4-0,63 mm grinding in a vertical vibrational mill confirms sufficient deviation of experimental data from the logarithmic gamma distribution for higher reduction ratio values, that is explained by strong influence of particle size on its destruction probability.

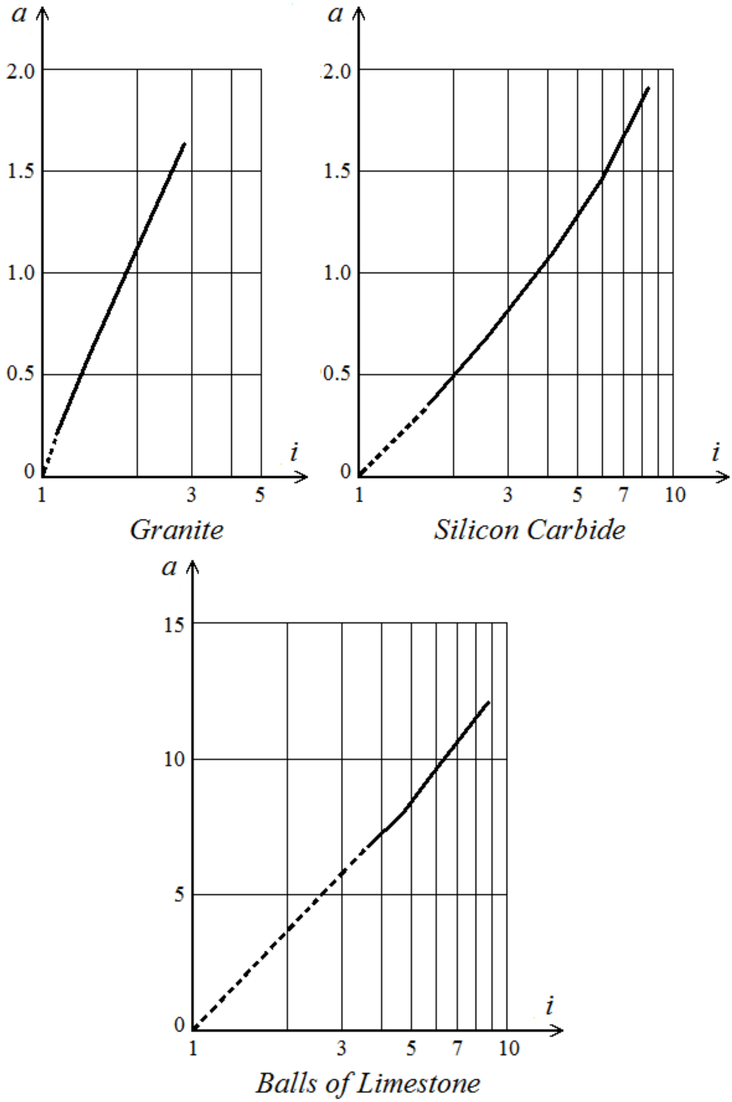


Fig. 8. Dependences of the shape coefficient of logarithmic gamma distribution of the material particle size on the current reduction ratio

The logarithmic gamma distribution is recommended for the description of destruction product of disintegrators having such working process, where the influence on material has the limited number of destruction acts regardless of the influence intensity.

The work results may be used for modelling of the disintegrator's product particle size distribution, especially for the finest particles, during destruction of solid lumps with high reduction ratio values, based on the analysis of splinters size distribution for lower reduction ratio values.

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UBSTANTIATION OF THE USE CONDITIONS SMALL-CONNECTING QUARRIES OVERBURDEN OF IRON QUARTZITE DEPOSITS FOR ARTIFICIAL BASES OF THE MINING AND CONCENTRATING COMPLEX OBJECTS

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Absract. The work is devoted to solving problems of waste utilization of the mining and concentrating industry and reducing the cost of building bases. For this purpose, the conditions for the use of the overburden from the extraction of minerals as a material of artificial bases have been investigated. The method of research in the field conditions of sealed and small-connecting overburden of iron quartz deposits has been developed. The method of laboratory determination of sealing parameters of these rocks is substantiated. Experimental research is planned and the method of evaluating the results of experiments on the basis of probability theory and mathematical statistics has been developed. According to laboratory and field studies of physico-mechanical characteristics of sealed insoluble overburden of iron quartzites quarry deposits and waste from the mining and processing industry, typical examples of the implementation with mathematical models probabilistic description of the distribution of random variables of the building properties of these sealed overburden for the arrangement of artificial bases are presented.

For the first time, the correct distribution laws for random variables of the physical (normal distribution law) and mechanical (the deformation module is best described by the logarithmically normal distribution law, and the internal friction angle and specific coupling, respectively, is a normal and logarithmically normal distribution law) of the properties of the sealed overburden of Iron quartz deposits as a material of artificial bases are grounded.

It is proved that waste formed during open mining of minerals in the mining and extraction industry, in particular, sealed and small-connecting overburden, should be used as a material for geotechnical structures.

Keywords: the quarry, small-connecting overburden, the seal, variance analysis, the angle of internal friction, specific clutch, deformation module, distribution law.

Introduction

To date, the construction of new facilities using floodplains, composed of weak soils. Under these conditions geotechnics recommends the installation of artificial arrays with better physical and mechanical characteristics than in natural state. Simultaneously, to solve the environmental problems associated with the recycling of industrial waste, and to reduce the cost of massive soil pillows, it is necessary to study the possibility of using the overburden remaining in the mining of minerals as a material of artificial bases.

At the same time, the inhomogeneous massifs are inherent in the parameters of which the random distribution of values of the characteristics of rocks can be accepted, the patterns of compaction of the medium in depth, and so on. These parameters depend on the type and natural properties of the material of artificial foundation, technological parameters of its construction. The existing normative approach evaluates only the quality of the seal and practically does not take into account the heterogeneity parameters of the bulk arrays. Methods for calculating soil pillows are deterministic, do not take into account the actual distribution of values of physical and mechanical characteristics compacted rocks, which leads to the laying of unjustified stocks of strength and deformability when they are erected.

Thus, the work is relevant for the possibility of overburden utilization as a material of artificial bases, expansion of the normative basis for designing and erecting soil pillows, taking into account the parameters of heterogeneity of soils of artificial bases of buildings.

A typical situation was at the construction site of the steel mill Vorskla Steel. It is planned to be built on the basis of the modern environmental and resource-saving technology of metallurgical production of the Austrian firm «VoestAlpineIndustrieanlagenbau» (VAI), ensuring the technical and ecological safety of natural resources. Quaternary deposits belonging to small-grained overlying rocks (mostly sand of medium to large size) of quarries of Yerristevsky and Lavrikovsky iron quartz deposits near the town of Horyshny Plavny of the Poltava region were used as material for the construction of an artificial array.

Therefore, for the purpose of the work, the estimation of the heterogeneity of the sealed insoluble overburden of quarries, as a material of artificial bases, is accepted.

1. Experience and problems of the use of waste mining and concentrating production as a material of artificial bases

A number of examples of artificial geomass, including islands, are known. The first in the world (island Dezim) was buried near the coast of Japan back in the XVII century. Today, this country has 9 islands with an area of 500 hectares (180-250 million m^3 of rocks). The most interesting of these is the island, which hosts the Kansai International Airport [1-3]. When it was created before the engineers there was a problem of high risk of earthquakes and typhoons. This geotechnical structure resisted the earthquake in 1995, as well as the very powerful typhoon in 1998, for which in 2001 it was recognized as a «Millennium Civil Construction Monument».

The experience of erection of the mentioned island has been successfully used for creation of other on muddy soils (airports of Kitakyushu, Kobe and Chubu). To preserve the ecosystem, the island of Chubu is constructed in the form of the letter *D*, which provided the circulation of water. Its shore is sometimes created by mountain masses, settled colonies of marine organisms on the slopes [4]. This is a unique example of solving the environmental problem. Also interesting is the experience of landfill waste from the island (436 hectares) in Tokyo Bay [1].

Ambitious project «Palm Islands» in the UAE is known in the world: PalmJumeira (2500 ha), PalmJebelAli (3700 ha), PalmDeira (7200 ha); Archipelago «World». It is by far the largest in the world [4]. In Southwest Asia, the island of Pearl-Qatar (400 hectares, 15,5 million cubic meters of soil) [4] is successfully exploited on the Persian Gulf coast in Qatar, in Bahrain, Lulu Island (600 hectares, 7,5 million cubic meters of soil). In Brazil, on an artificially named island, the Macau International Airport [1] is based. In Canada, the island of Notre-Dame is arranged in the middle of St. Lawrence.

To build such artificial bases (e.g, ground pillows), a significant amount of rocks is required, selected from appropriate quarries whose arrangement causes loss of fertile lands. This exacerbates environmental problems. On the other hand, large areas used for agriculture and forest plantations are covered with dumps in the mining and processing industry - dispersed rocks containing: poor ores; enrichment wastes (waste heaps of mines and quarries of sulfide ores of non-ferrous metals, oxide and silicate ores of ferrous and alloying metals); slimes and "tails" of ore-dressing factories; overburden [5].

According to V. Krutov, when 1 ton of iron is melted, an average of 1.2 tons of empty rock and 0.9 tons of slag, 1 ton of copper - respectively 4.2 tons of empty rock and 30 tons of slag are formed on average. To receive 1 ton of gold, about 23 million tons of minerals are processed. Due to active mining and technical work on the surface of the planet, the total volume of bulk rock is accumulated over 2000 billion cubic meters. Therefore, it is both environmentally and economically feasible to develop new ways of using these wastes for soil pillows. Norms do not prohibit their use. The composition of rocks is mainly due to geological conditions of the locality and human economic activity [6]. Depending on the composition and structure features, the waste heaps of the mining and concentrating production are classified in the «Dumps of Soils and Waste of Enterprises» group, which are homogeneous in composition but have uneven compressibility [6].

We know the experience of M. Zotsenko [7] using waste from the mining and processing industry as a material of pillows. The tailings of the Poltava Mining and Processing Plant in the town of Horyshniy Plavni has an area of about 1,400 hectares and contains over 156 million tons of waste of enrichment of ferruginous quartzites. 90% of the granulometric composition of these deposits is dust particles of silicate rocks. The use of these wastes in the embankments is complicated by minimal connectivity, which makes them unstable in dispersal and the effects of dynamic influences.

The diversity of composition and structure of rocks, the influence on them of engineering-geological processes, their polydispersity, polyphase and polymerisation, the nature of internal bonds are the causes of their heterogeneity in the natural state [8]. Consequently, if the factors affecting the formation of the properties of rocks are random, then the physical-mechanical parameters of the array have a random nature.

The heterogeneity of the rocks finds its expression in: the uneven distribution of matter in the array; the presence of defects in the structure, which determine the unevenness of internal bonds; a significant difference in the results of the actions of external forces and other factors on the same type of samples; essential spread of values of the characteristics of samples taken from the same layer [7].

In compacted form, rocks are more homogeneous in compression than in natural [7], but still field studies [9] recorded a significant heterogeneity of compacted soil, and its parameters are almost not investigated. M. Rat, V. Bondarik, I. Komarov, V. Romanchuk, K. Ruppenet and others [10] proposed a method for assessing the heterogeneity of soil bases.

Random values (RV), which include the properties of rocks, are completely characterized by distribution curves. Depending on the peculiarities of those or other RVs curves of their distribution may be of a different kind. According to M. Yermolayev [11], M. Maslov [12], V. Krutov [6], A. Pschenikin [13], the curves of the distribution of values of soil parameters correspond to the law of the normal distribution of Gauss. M. Goldstein [14] believed that for mechanical properties the most typical is the lognormal (logarithmically normal) distribution. According to V. Shilin [15], the stochastic properties of rocks describe the normal improved and Gram-Charlier laws of the distribution of RV. S. Matyus believes that RV of the angle of internal friction and specific traction of rocks is best approximated by normal or lognormal DL (distribution law).

As the criterion of variability (random distribution) of soil properties in compliance with Public Standard DSTU Б В.2.1-5-96, the coefficient of variation v_x ($v_x=\sigma_x/m_x$) is adopted, which is the ratio of the mean square deviation σ_x of the characteristic x and its mathematical expectation m_x. According to [16], the magnitude of the variation coefficient of the deformation module of natural sands does not depend on the coefficient of porosity and their size and varies within

12-20%, moreover, with an increase in the magnitude of *E*, a slight decrease in the coefficient of variation is observed.

For natural rocks, the variation coefficient of properties v_x depends on the number of tests *n* (table 1.1) [11]. It shows a significant variability of characteristics v_x=20-30%. There is a tendency of decrease v_x with increasing *n*. With an increase in the experimental area, the value of the coefficient of variation becomes sufficiently stable.

Table 1.1

The value of the variation coefficient v_x for the soil characteristics

Characteristic	The variation coefficient υ_x, %			
	sand	suppositories	loam	clay
Humidity, *w*	30-50/4.4-49	10-30/6.2-27.7	8-28/3.8-15.0	4-25/12.65
Porosity coefficient, *e*	3-13/1.1-6.7	6-12/2.3-16.5	6-25/3.5-14.2	3-22/19.3
Density, ρ	2-7.5/0.5-3.2	2-4.5/0.5-2.5	2.5-7.5/0.8-3.7	2-6/4.3
Particle density, ρ_s	-/0-0.3	-/0.2-0.65	-/0.2-0.6	-/0.8
Number of lashes, I_p		25-50/-	5-35/-	7-30/-
The limit of rolling, w_p		6-17/-	5-25/-	7-27/-
Yield line, w_L		5-16/-	5-20/-	5-20/-
Resistance to shifts, τ	-/-	9-27/-	6-29/-	-/-
Deformation module, *E*	-/-	-/-	15-35/18.6-65.4	-/-

In numerator data for M. Ermolaev [12]; in the denominator - according to O. Bugrov [18].

The spread of values of physical and mechanical characteristics of both natural and compacted soils is greater than the distribution in other materials of building constructions. In particular, in compliance with Public Standard DSTU Б В.2.1-5-96, the engineering-geological element is considered to be homogeneous if the coefficient of variation for physical properties does not exceed υ_x<15%, and for mechanical - υ_x<30%. In comparison, for example, with the prism strength of concrete on compression υ_x=13,5%, the strength of reinforcing steel at stretching υ_x=5%, the modulus of elasticity of concrete υ_x=4,4-9,2 %, etc.

Consequently, the rocks of densified arrays are characterized by a pronounced heterogeneity, which is reflected in the high variability of the characteristics of rocks (soils). For its parameters, it is expedient to accept a random distribution of values of soil characteristics, patterns of compaction of the medium in depth, anisotropy of their mechanical properties. Therefore, the task is to carry out complex investigations of the physical and mechanical properties of sealed, non-intersecting overburden of quarries of iron quartz deposits and to experimentally obtain the statistical data of the distribution of these characteristics.

2. Methods of researching the artificial bases properties from small- connecting overburden of quarries

Field research has obtained the most reliable results from the consolidation of small-leafing overburden. Defined: w_0 and $\rho_{d\,\max}$ when using a specific roller; the number of its passes followed to reach $\rho_{d\,max}$; the maximum thickness of the compacted soil with a homogeneous density; regularities of change ρ_d in the process of slipping. The control of the soil properties delivered to the site fixed the parameters of the roller, the mode of its work and the bandage, the number of passes one by one, measured the thickness of the layers of the pillow before and after the rolling.

To establish the relationship between the physical and mechanical properties, a genetically homogeneous material (in this case, Quaternary quartz deposits of Eristovsky and Lavrikovsky deposits) is needed. This material should be uniform in granulometric and mineralogical composition. According to the analysis of the rock formation, the rock formation is divided into: sand is shallow, homogeneous (60,6% of samples); a mixture of sand of shallow, homogeneous with a pollen plastic loamy sand (27,8%); sand of medium size, homogeneous (11.6%). For each of them it is revealed w_{opt}, at which it is achieved $\rho_{d\,\max}$ at the minimum number of roller passes N.

Samples taken in the cutting rings were taken to the laboratory. The granulometric composition in compliance with Public Standard DSTU Б В.2.1-2-96 was determined and its name was determined. Sieve analysis was used to determine the granulometric composition of the soil. Then in compliance with Public Standard DSTU Б В.2.1-17:2009 physical properties were determined: density ρ, soil skeleton density ρ_d; humidity w, - and for sandy admixtures, and also the humidity at the limit of rolling W_P and yield W_P, the ductility number I_P, the yield index I_L.

Deformation indices were determined in compression testing devices with a stepwise application of pressure $\sigma = 0{,}05; 0{,}10; 0{,}20; 0{,}30$ MPa to samples without their lateral expansion. Determined: the initial coefficient of soil porosity e_0; the coefficients of porosity e_i for the corresponding pressures σ_i; the corresponding compressibility coefficients m_0 and the relative compressibility m_v. After compression, the same specimens (in compliance with Public Standard DSTU Б В.2.1-4-96) were tested in a standard device with a shift in one plane, which determined the values of the internal friction angle φ and specific coupling c.

Laboratory data of the compacted sands properties to determine their normative values were statistically processed in accordance with DSTU Б В.2.1-5-96. Normative values of the physical characteristics of the soils X_n (humidity w, density ρ, soil skeleton density ρ_d) and the deformation module were taken as an arithmetic mean value $\overline{X}$

$$X_n = \overline{X} = 1/n \sum_{i=1}^{n} X_i, \tag{2.1}$$

n - number of definitions of the characteristic (not less than six); X_i - the values of the characteristics obtained from the results of individual (i) experiments.

To exclude possible errors remaining after the analysis of research data, statistical processing was performed. Excluded are the individual (maximal or minimal) values X_i for which the condition was fulfilled

$$\left|X_n - X_i\right| > \nu \cdot S, \tag{2.2}$$

ν - a statistical criterion, which depends on the definitions number n of the parameter; S - the mean square deviation of the characteristic, determined by expression

$$S = \sqrt{(1/n-1)\sum_{i=1}^{n}\left(X_n - X_i\right)^2}\,. \tag{2.3}$$

Again the values X_n and S were found, with the exception of the individual values of the parameter for the remaining experimental data.

Normative values of the internal friction angle φ and specific coupling c were found by statistical processing of individual values $tg\varphi_j$ and c_j. The number of individual values $tg\varphi_j$ and c_j should be at least six. For each j-point of the soil study, the individual values of $tg\varphi_j$ and c_j were determined by the least squares method based on the data of at least three determinations of the soil resistance of the cut τ_i at different stresses σ_i within the same range σ_i:

$$tg\varphi_j = \left(k\sum_{i=1}^{k} \tau_i \sigma_i - \sum_{i=1}^{k} \tau_i \sum_{i=1}^{k} \sigma_i \right) \Bigg/ \left(k\sum_{i=1}^{k} (\sigma_i)^2 - \left(\sum_{i=1}^{k} \sigma_i \right) \right)^2, \qquad (2.4)$$

$$c_j = (1/k)\left(\sum_{i=1}^{k} \tau_i - tg\varphi_j \sum_{i=1}^{k} \sigma_i \right), \qquad (2.5)$$

k - the number of determinations τ in each point of the soil layer.

By the values $tg\varphi_j$ and c_j, the standard values $tg\varphi_n$ and c_n were determined according to formula (2.1) and the mean square deviation of $S_{tg\varphi}$ and S_c in the expression (2.3). To exclude possible errors in the values $tg\varphi_j$ and c_j perform a similar statistical processing. A pair of values $tg\varphi_j$ and c_j were excluded if at least one of them fulfilled the condition (2.2).

As a result of field and laboratory research of the properties of sealed overburden, a statistically substantiated volume of definitions of soil characteristics (granulometric composition w,p_d,E,φ,c) and rolling parameters (h, Δh, static or vibratory mode) was obtained for establishing the relationship between physical and mechanical properties these rocks, taking into account the influence of the rolling parameters, namely:

- for fine, homogeneous sand: the humidity w, the soil skeleton density ρ_d - n=314; the internal friction angle φ and the specific clutch c - n=70; the deformation module E– n=314;
- for fine, homogeneous sand with a pollen plastic loamy sand: w and ρ_d - n=144; φ and c - n=39; E - n=144;
- sand of medium size, homogeneous: w, ρ_d and E, a n=61;
- number of measurements of moving the roller on one trace (Δh) n=45;
- number of measurements of thickness (h) of spilled layers kilkist zamiriv tovshchini dsipanih shariv n=45.

Such a technique is used for the analysis of RV experimental data [18]. 1. Determine the required amount of research data.

2. After obtaining experimental data in field and laboratory conditions, the RV in the form of a statistical series is analyzed in order to exclude gross errors: a) find suspect values x_{Max} and x_{Min}; *b*) determine the average value $\overline{x}$ and the mean square deviation $\hat{x}$;

c) calculate confidence intervals; *d*) reject false values and determine the homogeneity of observations; e) exclude, if necessary, from a statistical series x_{Max} or x_{Min} and obtain a new statistical series of new *n* members.

3. Calculate the statistical parameters of the new series: the mathematical expectation $\bar{x}$, the dispersion σ_2, the mean square deviation σ (standard) and the variation coefficient v.

4. Select the DL for the RV and check its adequacy.

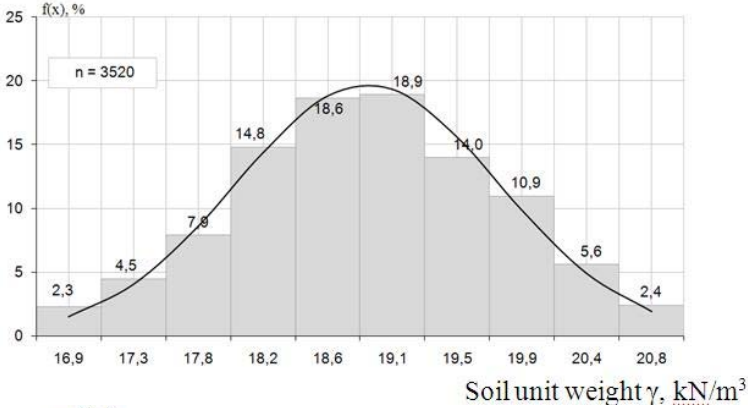
Thus, as a result of complex investigations of sealed small-necked overburden properties within the pillows, sufficient samples of random values of experimental characteristics and technological parameters were obtained. Each of the indicators of the soil′s physical and mechanical properties is characterized not by any single value, but by a certain set of random values. The higher this population, the more accurately the soil characteristics can be determined quantitatively for this indicator. Fully characterize random values distribution curves. After sorting out the errors, the statistical series is analyzed, the correct distribution is selected and its parameters are determined. According to the criterion of changing properties of soils pillows, the coefficient of their variation is taken.

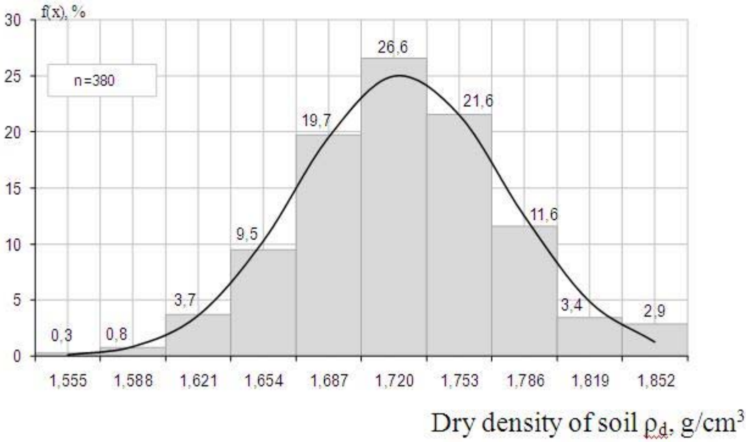
3. Results of research of heterogeneity the sealed insignificant small-connecting overburden in the composition of artificial bases

Passive experiment performed to study the distribution and statistical parameters (mathematical expectation, standard, coefficient of variation, asymmetry, excess), random values of physical and mechanical properties (specific gravity, humidity, skeleton density, deformation module, specific gravity, internal friction angle) of sealed, small-connecting overburden of pillows.

3.1. The distribution of random variables of the physical characteristics of the sealed, small-connecting overburden of artificial foundation

The statistical analysis of RV of physical properties of compacted overburden, in particular, humidity w, density ρ and soil skeleton density ρ_d, was carried out for the purpose of calculating statistical parameters, construction of experimental distributions and their approximation. The most typical experimental frequency histograms and graphs of analytical distribution for compacted rocks are shown in Fig. 3.1.





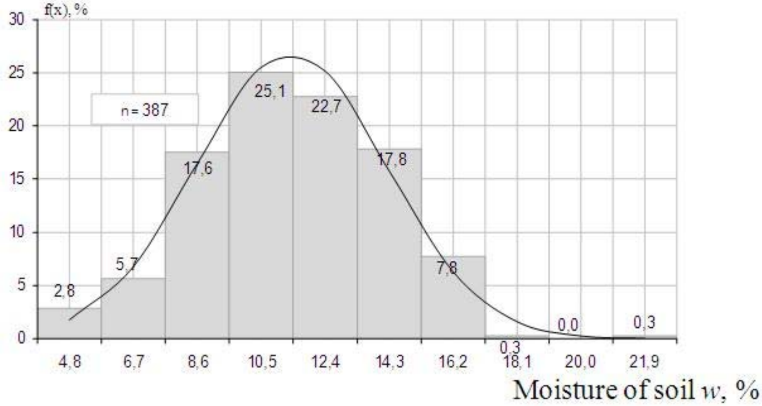


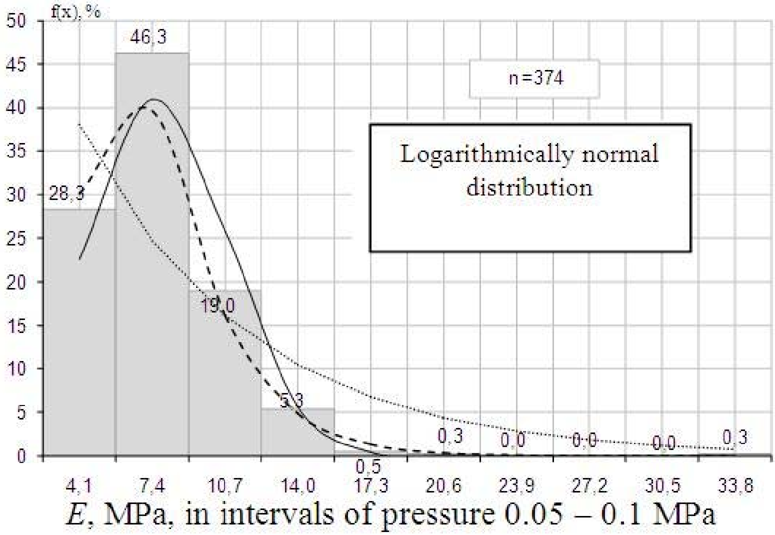
Figure 3.1. Density distribution diagrams of random variables of soil physical characteristics of cushions: *n* – number of random variables

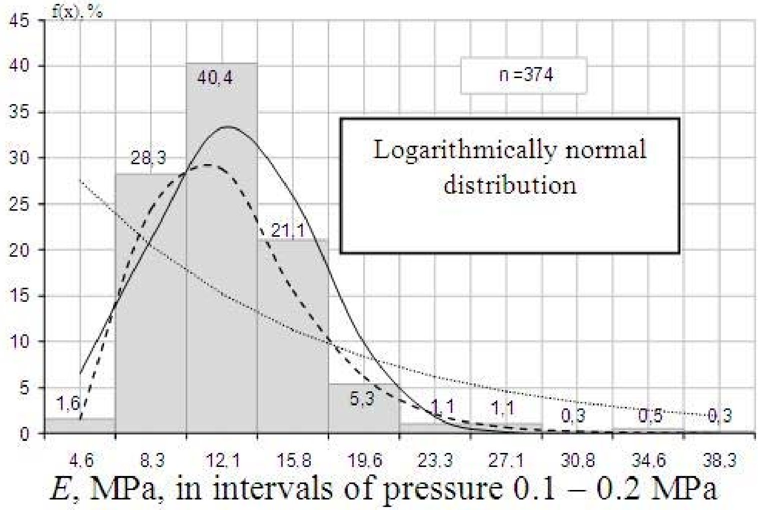
Thus, from the analysis of Fig. 3.1, we can conclude that for the approximation of experimental histograms of the distribution of the RV of the physical characteristics (w, ρ, ρ_d) of the compacted soils in most cases, the Gaussian law is the most correct.

3.2. Distribution of random variables of the deformation characteristics of sealed, small-connecting overburden of artificial foundation

According to the complex studies, experimental data of the deformation module of the sealed small-connecting overburden at different pressure intervals in the compression device (σ=0,05…0,1; 0,1…0,2; 0,2…0,3 MPa) were obtained. The most typical histograms and charts of the analytical distribution of the RV of the deformation characteristics, in particular the deformation module *E*, of the compacted soil, depending on the pressure intervals in the compression device, are given in Fig. 3.2.

Analyzing the experimental and analytical distributions of the RV of the deformation module *E* of the sealed, small-connecting overburden of pillows (Fig. 3.2), one can generalize that for them in most cases the logically-normalized DL is the most correct for approximation.


f(x), %
n =374
Logarithmically normal distribution
46,3
28,3
19,0
5,3
0,5
0,3
0,0
0,0
0,0
0,3
4,1
7,4
10,7
14,0
17,3
20,6
23,9
27,2
30,5
33,8
E, MPa, in intervals of pressure 0.05 – 0.1 MPa


f(x), %
n =374
Logarithmically normal distribution
40,4
28,3
21,1
5,3
1,6
1,1
1,1
0,3
0,5
0,3
4,6
8,3
12,1
15,8
19,6
23,3
27,1
30,8
34,6
38,3
E, MPa, in intervals of pressure 0.1 – 0.2 MPa

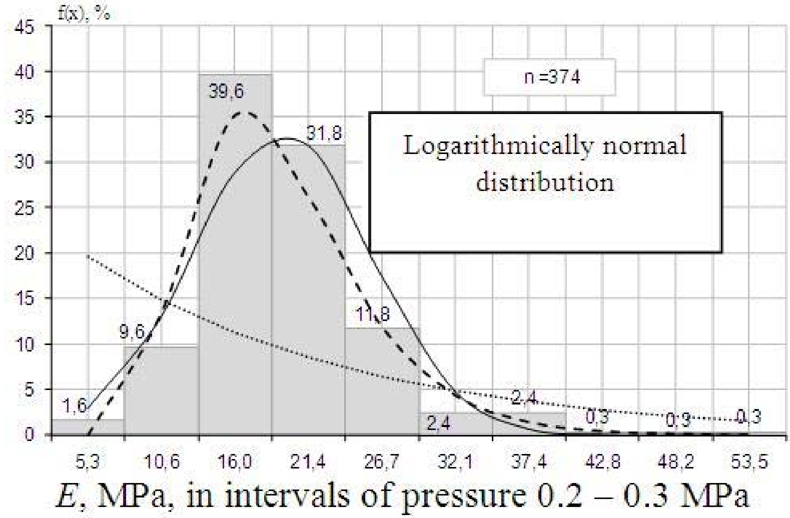


Figure 3.2. Density distribution of random variables of soil deformability characteristics of cushions: n – number of random variables

3.3. Distribution of the random variables of the strength characteristics of the sealed, non-connecting overburden of artificial foundation

As shown by the analysis of the RV of the strength characteristics of the sealed, non-connecting overburden (the angle of internal friction φ and specific adhesion c) of the pillows depend on each other, that is, the so-called random vector [19, 20, 21].

By the method [19, 20, 21, 22] of processing and selecting the function of the density of the distribution of random vectors, we must establish the distributions of the individual quantities. At the first stage of the analysis of the investigations of these properties of sealed, non-connected overburden, they were considered separately for the determination of statistical parameters and the approximation of experimental histograms. The statistical parameters of the distributions of the strength characteristics of the sealed, non-connecting overburden of pillows are summarized in Table. 3.1.

On the basis of experimentally obtained statistical data and DL parameters of the strength of the sealed, small-connecting overburden φ and c, a random vector is obtained, the graphic representation of which is shown in Fig. 3.3.

Table 3.1

Statistical parameters of the experimental distributions of the RV of the strength characteristics of the small-connecting overburden of pillows

Statistical parameters	M_1	M_2	M_3	M_4	$\overline{X}$	$\hat{X}$	μ_3	μ_4	σ	v, %	A	E
Specific grip of soil c, кPa	0,9 (0,6)	2,3 (2,3)	7,05 (4,15)	27,2 (14,3)	16 (3,1)	16,23 (0,08)	82,58 (0,01)	1308 (0,01)	4,03 (0,3)	25 (0,1)	1,26 (0,3)	1,97 (-0,7)
Angle of internal friction of the ground φ, ^{0}	-0,9	2,4	-7,1	26,1	31,3	10,92	-38,87	514,0	3,30	11	-1,1	1,31

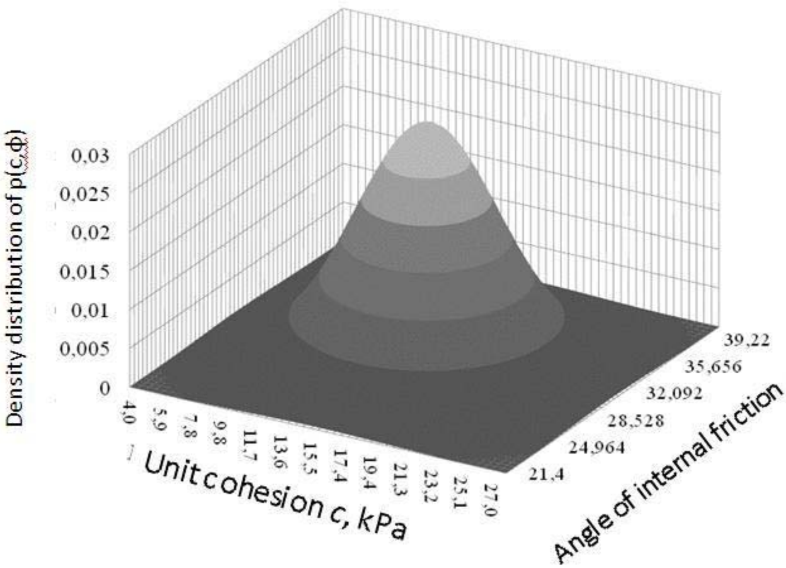


Figure 3.3. Density distribution of random variables of soil strengths characteristics of cushions

Thus, the interconnected parameters of the strength of the sealed, non-connecting overburden φ and σ can be described by a surface projected onto a plane parallel to the coordinate plane of the specific gravity of the soil c in the form of a logarithmically normal law curve with corresponding parameters at a fixed internal friction angle φ. On a plane parallel to the coordinate plane of the angle of internal friction of the soil, φ is the curve of the normal law with the corresponding parameters for its fixed coupling with.

According to the results of statistical analysis of experimental studies of the heterogeneity of the physical and mechanical characteristics of sealed, small-connecting overburden and their mixtures, the following generalizations can be made.

The distribution of explosives of physical properties of sealed, small-connecting overburden is expediently approximated by normal DL, and the skeleton density the of compacted soil mixtures - by polynomial-exponential. The coefficient of variation in the values of density and skeleton density of the soil varied within 2-4,4 %, and its moisture content was 23-36%. The correct distribution of the RV of the deformation module E of the sealed, non-connected overburden and their mixtures is logarithmically normal. The coefficient of variation E was 33-57%.

Conclusions. From the analysis of the results of complex studies with the volume of samples from 50 to 3000 RV, statistical data on the variability of the values of the characteristics of the sealed, non-connecting overburden of quarry deposits of iron quartz deposits has been obtained, which made it possible to draw such conclusions.

Wastes generated during the open mining of minerals in the mining and processing industry, in particular the sealed, small-connecting overburden, should be used as a material for geotechnical structures.

For the analytical description of the experimental distribution of the RV of the physical characteristics of the sealed, small-connecting overburden it is expedient to use the normal distribution law, and for the density of the soil skeleton of the compacted mixtures, it is polynomial-exponential. At the same time, the coefficient of variation in the values of soil skeleton density varied within 2-4,4%, its moisture content was 23-36%, and the specific gravity of the soil - 4-4.6%.

The modulus of deformation of sealed non-connecting overburden and their mixtures is best described by a logarithmically normal distribution law. The coefficient of variation of values of the deformation module is 33-57%. The internal friction angle φ and the specific gravity of the compacted soils and their mixtures are random vectors and are best described, respectively, by the normal and logarithmically normal distribution law. The coefficient of variation of the values of the angle of internal friction was 11%, and the specific coupling - 25%.

RV of the specific resistance of the penetration of sealed, non-connected overburden is best approximated by the exponential distribution law. The coefficient of variation of the values of specific penetration resistance was 57%.

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SETTLED DUST COLLECTION IN PROCESSING SHOPS OF ORE MINING ENTERPRISES

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Aim. The research is aimed at solving the problem of dust collection inside processing shops of ore mining enterprises.

Methods. Methods of research and publication analysis and target natural investigation into industrial premises were used to study, systematize and generalize dust sources and techniques to collect it.

Findings. Dust accumulation on the surface of industrial structures and equipment does not occur in the same manner. The amount of dust and intensity of its settlement in the upper part of an industrial premise are much smaller than those next to dust emission sources. That is the reason why dust collection intervals in a separate premise will depend on intensity of dust precipitation and accumulation on elements of a given object. This will enable enterprises to keep industrial premises clean by using industrial mobile dust collectors that could be transported to another object after cleaning the previous one.

Scientific novelty. Intensity of dust precipitation on various surfaces inside processing shops of mining enterprises makes from 0.01680 kg/m^2 to 0.1440 kg/m^2 per day. During long periods, on surfaces of construction structures, pipelines, equipment, etc. a great amount of dust is accumulated making up to 20 kg/m^2 in some places. The thickness of dust layers reaches 10 mm. The most intensive dust settlement is observed next to crushers and screens as well as at ground level of the crusher house floor дробильного корпуса.

Practical relevance. There is developed a mobile dust collector that can move independently to a required object within an enterprise. The problem of dust collection can be solved at the lowest cost by removing stationary pipelines, installing the machine at the most convenient places along the perimeter of an industrial object, fast handling of pipelines and high capacity. It can be used together with current dust suppression and collection means to maintain dust level of working places within sanitary standards.

Keywords: dust emission, dust settling/precipitation, dust collection, industrial dust collector.

Introduction. Technological processes of rock processing are accompanied by intensive dust emission which increases allowable values of dust content in the air of working places. Available methods of dust suppression such as aspiration, ventilation, dust binding by moistening processed materials, capsulation of trans-loading units, etc. are unable to localize dust sources completely.

Emitted dust spreads all over industrial premises and settles on various surfaces (construction structures, pipelines, cable trunks, equipment elements, etc.) making layers of various thickness. Under the action of various factors (equipment operation and vibration, repair works, aeration flows, etc.), settled dust can repeatedly turn into aerosols. Due to this, dust content of the air increases that can provoke vocational diseases of processing shop employees.

At the same time, the dust having settled on various surfaces makes equipment maintenance and repair more complicated, expedites wear of certain assemblies and parts, reduces maintenance periods, and disrupts the operation of control and weighing equipment.

That is why, solution of dust collection problem is urgent for processing shops of mining enterprises, it allowing them to facilitate working conditions, reduce vocational disease rates and cut down equipment wear.

1. Analysis of dust emission sources and dust content in the air of working places

Transloading units of bulk materials, grizzly and unbalanced-throw screens, medium and fine crushers, belt conveyors are basic and primary dust emission sources in processing shops [1-6].

While loading ore or waste materials from one equipment type to another, dust is emitted into the air of an industrial premise because of diffusion of a dust-air flow, kinetic power of dust particles imposed by moving mechanical parts, excessive air pressure inside transloading units (chutes) and receiving bins as well as inside protective shields of grizzly and unbalanced-throw screens. Excessive pressure makes dust enter the premise atmosphere through loading mouths or slots. Without protective shields of screens the air dust level reaches 800-1000 mg/m^3, and it reduces five-tenfold if they are available [5]. Dust emissions through loading mouths of crushers make 8-60 m/sec [6]. The way screens, inlet and outlet mouths of chutes and crushers are compressed and covered affects dust content in the air of working places.

A conveyor belt is another dust source. Humidity of transported materials and operation of liquid atomizers to suppress dust enhance dust adhering to the belt surface [5, 6]. If conveyor belt cleaners do not operate properly, part of dust materials stick to supporting rollers. In this place, intensive abrasion of ore dust occurs between a roller and the belt accompanied by intensive dust emission. Dust adhesion to supporting rollers happens along the whole length of the conveyor. The maximum dust adhesion is observed in the head part of the conveyor just after unloading. As moving away from the conveyer head, adhesion decreases [5].

Irregularity of materials on the dumping tract and adhesion of hauled materials on supporting rollers cause vibrations of the conveyor belt and dust emission.

When bulk materials move along unbalanced-throw screens, they make oscillatory movements vertically. It results in auxiliary dusting of materials that enhances fine-dispersed dust formation. When the material moves vertically, dust comes out of the flow and enters the premise atmosphere through slots of protective shields of the screen and the loading device. The dust level next to operating screens ranges from 20 to 150 mg/m^3 [4, 5].

When loading cone crushers of КСД-2200, КМД-2200m and КМДТ-2200 types, there is additional dusting of the rock mass flow moving along the cone towards the crushing bowl, this causing intensive emission of fine-dispersed dust. Because of excessive pressure in the loading space of the crusher, the dust comes out of the total flow and enters the air atmosphere of the shop. The most intensive dust emission is observed in case of using hammer mills with excessive pressure of 20-30 N/m^2 inside their housing [4, 5].

In hauling bulk materials from the receiving bin to the emergency storage, from one conveyor to another, there occurs adhesion of processed materials under screens. This causes production stoppages. These places are cleaned during inter-shift periods by means of pick hammers and compressed air. Such operations intensify dust content in the air that exceeds standard values by a factor of 15-20.

The dust settled on various surfaces is a source of secondary dusting. Some authors indicate that secondary dust sources are more difficult to eliminate than the primary ones [4, 5].

During their shifts, employees clean their working places by means of brooms, spades and scrapers. The floor surface is the main cleaning object, protective shields of equipment and construction structures are sometimes cleaned as well.

According to A.A. Kurnikov [7], while cleaning premises with brooms and scrapers, 40-50% of dust is left and about 10% rises into suspension. The residual concentration of dust is almost by 1.35 times bigger than background concentration.

Thus, both primary and secondary dust sources cause air pollution of industrial premises.

Table 1 presents the results of measuring the dust content in working places at the crushing and sizing plant of Rodina mine conducted by the industrial sanitary laboratory of the PJSC KZRK (Kryvyi Rih Iron Ore Works).

Table 1

Dust concentration in working places of the crushing and sizing plant of Rodina mine

Working place	Characteristics of a technological process	Dust concentration in the working place, mg/m^3
Belt runner	Monitoring and control	46,0
Screener	Screening	4,4
Crusher operator	Crushing	4,2
Bunkerman- signaler	Monitoring and control	5,4
Shift repairman	Monitoring and control	2,9-3,5

The maximum allowable concentration of dust containing from 10 to 90 % of crystalline silicon dioxide makes 2 mg/m3. The measurement results indicate that dust content in working places exceeds allowable values. Screens, crushers, transloading units and belt conveyors are the most intensive dust sources.

2. Numerical assessment of dust accumulation on various surfaces

The dust settled on various surfaces is characterized by different fraction composition. Fine-dispersed dust particles of 0-20 μm can be in suspension state for a long time. The most hazardous for health are dust particles of 0.2-5 μm causing acute respiratory diseases [5]. Particles of up to 200 μm enter the air atmosphere and precipitate on the surface of construction structures, cable trunks, equipment, etc. under gravity. Gradually dust accumulates on these surfaces. Part of dust becomes compressed while newly settled dust remains suspended for a while and from time to time is blown off the surface by air flows or leaks down under the action of different factors. Larger dust particles of 200 μm-1 mm fall onto the floor next to the dust source. The amount of settled dust decreases with distance from the dust source. Percentage of fine fractions in flying dust increases due to larger particles precipitating. If the dust source is high above the floor, emitted large-dispersed particles tend to reach the ground level. In most cases, dust settles on horizontal surfaces which happen to be on its way.

It is very difficult to get rid of settled dust completely using some simple cleaning means only. Application of brooms and brushes causes secondary intake of dust into the air creating increased dust content [7].

The most intensive dust precipitation on various surfaces results from crushing, grinding, screening and mechanical haulage of solid materials. Researches into intensity of dust accumulation at crushing and sizing plants enabled definition of the daily amount of settled dust.

There were selected places of maximum dust precipitation at the crushing and sizing plant of Rodina mine next to the most intensive dust emission sources. All the selected measurement points are on the floor next to the dust sources. The plant processes raw materials (iron ore) 18 hours a day and inter-shift stoppages and preventive repairs take 6 hours. It was noticed that dust precipitation occurs not only during plant operation, but also between shifts.

Observations were held within 24 hours. Samples of settled dust were weighed by means of analytical scales. Observation data are given in Table 2.

Table 2

Amount of dust settled on the floor next to the dust sources at the crushing and sizing plant

Measuring points	Amount of settled dust, kg/m^2 per day
1	2
Crusher КСД-2200 Б # 1	0,1152
Belt conveyor ЛК-3, ЛК-4 (drive)	0,0456
Crusher КСД-2200 Б No 2	0,1248
Belt conveyor ЛК-10	0,0696
Unbalanced-throw screens ГИТ-71 #1, 2	0,1416
Grizzly screen	0,1272
Belt conveyor ЛК-5	0,0168
Ground level of the crusher house	0,1440
Belt conveyor ЛК-2	0,0552
Gallery of Conveyors 8, 9 (tail)	0,0312
Gallery of Conveyors 8, 9 (head)	0,0408

The observation results indicate that if aspiration systems function properly, dust precipitation on the floor next to the selected dust sources make 0,0168-0,1440 kg/m^2 per day. The most intensive dust precipitation is next to crushers and screens and at the ground level of the crusher house making 0,1152-0,1440 kg/m^2 per day.

Part of the dust settled on the floor next to dust sources during the shift is cleaned by employees by means of scrapers and brooms. The rest of the dust in suspended state gradually precipitate on the surface of equipment, construction structures, pipelines, etc. As time goes by, dust accumulates on these surfaces making a layer of considerable thickness. This dust is not cleaned by conventional brooms and scrapers as this greatly increases the dust content in the air.

To determine the weight of settled dust on various surfaces of the crushing and sizing plant, there were chosen sites of 1 dm^2 on different heights on horizontal and spherical surfaces. A year later, the dust was collected there. The samples from each site were weighed and areal density of the settled dust was determined by re-computation. While sampling, dust layer thickness was determined as well. The results are given in Table 3.

Table 3

Results of determining dust density and thickness on various surfaces of the crushing and sizing plant of Rodina mine

Sampling places	Areal density of dust, kg/m^2	Thickness of the dust layer, mm	Height from the floor, m
1	2	3	4
The drive of the КСД-2200 crusher	3,88	1,5	1,5
The guard of the crusher charging chute	15,46	4,0	4,6
Column ties L75×75 of the crusher house	10,59	3,0	1,0
The repair platform near the КСД-2200 crusher (floor)	3,23	0,8	0
Drive guard of Conveyor 3	1,47	0,5	1,2
Metal structures of the ventilation installation	4,23	1,2	2,5
АТУ 2 pipeline	4,29	1,2	3,0
The floor under АТУ 2	41,23	10,8	0
The beam of the bridge crane of the crusher house	11,84	3,2	6

Continuation of table. 3

1	2	3	4
The frame of the laminal feeder	5,23	1,4	1,2
The guard of the charging chute of Conveyor 9 (tail)	18,01	4,7	1,2
Guard of charging chute of Conveyor 8 (tail)	22,67	5,9	1,2
The frame of Conveyor 9	7,74	2,6	0,7
The frame of Conveyor 8	5,46	1,4	0,7
Metal structures of the gallery of Conveyors 8 and 9	19,73	5,2	3,2
Cable runs of Conveyors 8 and 9	4,54	1,2	1,8
Metal structures of the gallery (ties)	11,22	4,3	1,8
Metal structures of the transloading unit	9,55	4,0	4,0
Metal structures of АТУ 4	22,51	6,0	3,0
The fire pipeline of ∅ 100 mm	3,12	0,8	1,2
The drive enclosure of Conveyor 9	17,33	4,6	1,5
The drive enclosure of Conveyor 8	21,98	5,5	1,5
The drive bottom of Conveyors 8 and 9	5,52	1,8	0
The cable runs along the gallery of Conveyors 8 and 9	8,89	2,3	1,8
The air duct of ∅ 108 mm (at EL of the storage hopper)	3,30	0,9	1,0
The window frame (at the same place)	0,30	0,1	0,8
Metal structures L100×100 (column ties)	12,34	2,6	5,2
The air inlet case of the storage hopper	22,14	5,6	4,8
Metal structures of the charging slot of the storage hopper	22,0	5,5	1,1
The pipeline of ∅ 159 mm	18,63	4,9	1,2
The cable run above the ГИТ-71 screen	6,05	1,6	1,8
The cable run at EL +4000 m	8,37	2,2	1,8
Metal structures at EL of the ГИТ-71 screens	11,51	3,0	4,1
Metal structures (column ties) L90×90	7,88	2,0	2,2

Continuation of table. 3

1	2	3	4
Metal structures of the industrial building (the ceiling beam I 30)	6,36	1,8	3,3
Metal structures (column ties) L75×75	25,52	6,6	2,4
Metal structures (the ceiling beam I30)	6,50	1,8	3,5
The air duct of ∅ 108 mm at EL +4.000 м	6,83	1,8	1,0
The air duct of ∅ 300 mm	21,93	5,8	1,8
The equipment at the repair platform at EL +4000 m	1,85	0,2	0
The ventilation pipeline of ∅ 450 мм	20,81	5,5	2,2
Cable runs of Conveyor 50	9,85	2,5	1,8
Cable runs inside the drive building of Conveyor 50	5,42	1,4	1,8
The fire main of ∅ 108 mm	10,28	2,7	1,8
The frame of Conveyor 50 (gallery)	2,97	0,9	0,7
The frame of Conveyor 50 (tail)	7,64	2,0	0,7
The frame of Conveyor 12	7,11	1,9	0,7
The frame of Conveyor 50	7,39	1,9	0,7
Ties of load-bearing columns (L90×90, left)	13,21	3,5	2,8
Ties of load-bearing columns (L90×90, right)	16,91	4,4	2,8
The repair platform	19,44	5,1	0
The air inlet of the charging slot of Conveyor 12	16,91	4,4	2,5
The air inlet of the charging slot of Conveyor 50	22,20	5,8	1,5
The air inlet of the transloading unit (the drive of Conveyor 12)	7,19	1,9	1,6
The air inlet of the transloading unit (the drive of Conveyor 50)	7,39	1,9	1,6
The air duct of ∅ 150 mm	15,73	4,1	1,5
The trans-loading unit from Conveyor 50 to Conveyor 12	3,64	1,0	1,1
Metal structures of the gallery of Conveyors 12 and 50	8,30	2,2	2,6
The same building of the trans-loading unit	6,45	1,7	2,6

Analysis of the results of determining areal density indicates quite a great amount of dust (up to 20 kg/m^2 per year) and in some places - over 40 kg/m^2 per year - accumulated during a long period of time on the surface of structures, pipelines, and cable trunks. On all flat surfaces, dust settles in equal layers with the exception of cylindrical and spherical surfaces of cable communications, air ducts and some other equipment. There, dust layers copy these objects outlines with the highest thickness in the central part of the surface. The amount of dust and intensity of its precipitation on the surfaces in the upper parts of a premise is much smaller than next to the dust sources. Therefore, intervals of dust collection in a separate premise will depend on intensity of dust precipitation and accumulation on elements of a given object.

3. Dust collection

Technologies of rock processing are noted for a variety of sources of intensive dust emission and great areas of its precipitation to be cleaned. Centralized industrial dust collectors can be applied to different surfaces in processing shops of mining enterprises [8-15].

Yet, these machines are stationary and applicable to just one premise. The stationary pipeline system when in long service tends to be polluted and requires either cleaning or demounting. Dust precipitation reduces the amount of air exhausted through dust-cleaning nozzles and changes aerodynamic indices and efficiency of the vacuum system as a whole.

The length of pipelines of stationary vacuum systems is conditioned by large areas to be cleaned and a great amount of equipment there. This requires highly efficient traction activators consuming much power.

Thus, a long stationary system of pipelines is the most significant disadvantage of all modifications of centralized industrial dust collectors. It should also be noted that one machine is applicable to one premise only. For large-scale enterprises, there should be several single-types machines, each of them having a stationary system of pipelines. It makes control over pipe blockage more complicated and is associated with higher maintenance costs of such a long system within one enterprise.

Dust accumulation on surfaces of industrial structures and equipment does not occur uniformly. It enables an enterprise to keep its industrial premises clean by using a mobile dust collector that is transported to another object after cleaning the previous one. Basic equipment of a mobile dust collector is in the body of a van outside a premise to be cleaned (Fig. 1).

Before starting the machine, some assembling operations are required to install vertical and horizontal sections of a pipeline.

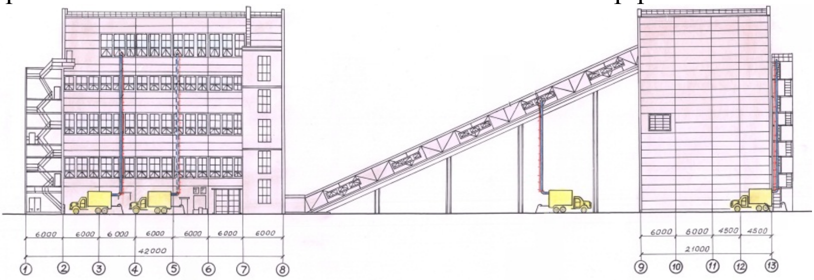

Fig. 1. Location of the mobile dust collector while assembling the vertical section of pipelines

As the industrial dust collector is mobile, assembly operations are conducted permanently when it is transported to another place. Before mounting the vertical section of a pipeline, the van is next to the premise where the pipeline system is to be installed at the set elevation (Fig. 1).

When the required elevation of the vertical section is reached, horizontal pipeline branching is mounted (Fig. 2).

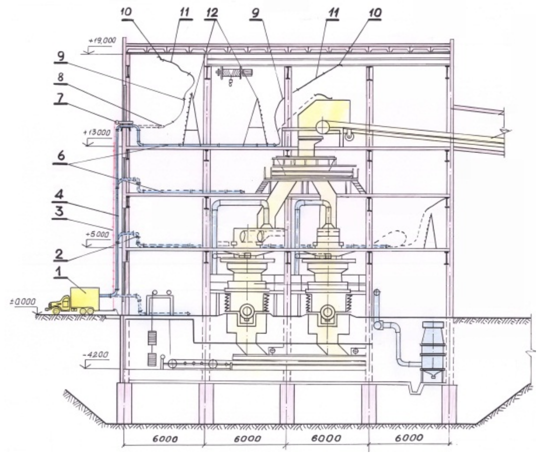

Fig. 2 Layout of pipelines inside a premise: *1* - mobile dust collector; *2* – pipeline elements (elbows, tees, etc.); *3* - winch cable; *4* - vertical pipeline; 5 - loading conveyors; *6* - horizontal pipeline; *7* - block-clamp; *8* - cone adapter; *9* - flexible hose; *10* - nozzle; *11* - lever; *12* - step-ladders

The mobile industrial dust collector is serviced by three people, one of them is a driver and the rest are dust-collecting operators. The driver controls basic equipment, while operators clean a premise. Both operators also deal with high-up surfaces and hard-to-reach places. One of them cleans the surface using a step-ladder, while the other watches out for him, switches a flexible hose from one pipe union to another, gives required nozzles, etc.

A mobile industrial dust collector is able to move independently to a required object of cleaning within a given enterprise as a traction activator and dust catching devices are inside a mobile van, while detachable pipelines and nozzles are mounted from separate sections during cleaning. It reduces the total length of the vacuum system and helps avoid blockage of pipelines in the long run. There is also an opportunity of cleaning hard-to-reach surfaces. As the van is located outside the cleaned object, the air does not re-circulate in heavily polluted premises.

Mobile dust collectors are applicable to shops with any appliance saturation. The machines are able to collect great amounts of dust and transport it to unloading points without intermediate loading-unloading operations. Dust unloading is performed outside the industrial premise to avoid its secondary dust pollution.

Conclusions

The most intensive dust sources in processing shops of ore mining enterprises are screens, crushers, transloading units and belt conveyors. Settled dust becomes a source of secondary dust emission. Both primary and secondary dust sources cause dust pollution of industrial premises with dust content in the air exceeding allowable standards in working places.

Intensity of dust precipitation on various surfaces in processing shops of ore mining enterprises makes from 0,0168 kg/m^2 to 0,1440 kg/m^2 per day. Over the course of a long period, a great amount of dust (up to 20 kg/m^2 in some places) precipitates on the surfaces of construction structures, pipelines, equipment, etc. Layers of settled dust can reach 10 mm. The most intensive dust precipitation is observed next to crushers and screens as well as at the ground level of the crusher house floor.

Dust accumulation on the surface of premise structures and equipment is not uniform. Dust quantity and intensity of its precipitation in the upper parts of a premise are much smaller than next to dust sources. Dust collection intervals in a separate premise will depend on intensity of dust precipitation and accumulation on elements of a given object. It will enable an enterprise to keep its industrial premises clean by using mobile industrial dust collectors transported to another object after cleaning the previous one.

A mobile industrial dust collector can independently move towards a required object within a given enterprise. Absence of a stationary pipeline system, an opportunity to be used in the most convenient points along the perimeter of a premise, quick mounting and demounting of pipelines, and high efficiency enables solution of the dust collection issue at minimal costs. If combined with current means of dust suppression and collection, the machine allows keeping dust content in working places within sanitary standards.

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INCREASE OF RESOURCE POTENTIAL OF AN ENTERPRISE AT AN EXPENSE OF AN INVOLVEMENT IN A MINING OF TECHNOGENIC DEPOSIT

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Abstract. The paper presents a procedure for creating technogeneous deposits on the base of tailing ponds in the mining and processing enterprises in order to increase resource potential of the country, and to provide rational usage of natural resources through involving of the technogeneous deposits, formed in such a way, into the field development. A list of researches is given which should be conducted before making decision on such deposit creation and which includes analysis of granulometric, chemical and mineralogical compositions and determination of certain physical and mechanical properties of the material to be stored. On the basis of previous industrial researches, parameters effecting on formation of technogeneous deposits are substantiated. Models simulating creation of technogeneous deposits at the stages of the tailing ponds designing and operation at the processing enterprises are presented.

The classification of technological schemes for the extraction of sludge from sludge storage by methods and means of excavation and transportation, as well as by physical and mechanical state of the contents with taking into account stage of exploitation are developed.

It is stated that technological schemes and complexes of machines and equipment for sludge excavation should be chosen depending on the parameters of the sludge storage, shape and type of the formed deposit, consistency of the content, conditions of transportation to the preparation factory and intended further use of the extracted raw materials.

Considered various options for technologies and technological schemes for the extraction of man-made minerals in existing sludge storages.

Introduction

The mining industry is an industry where the nonwaste production has not yet been achieved and the amount of generated wastes is the largest of all industries.

Ukraine is the country where the share of the mining industry in total production is 40-50 %.

As a result of imperfections in engineering processes a part of minerals get into wastes, the magnitude of which depends on the type of process and its technological level and the ability of minerals for preparation as well as the complexity of the extraction from the enclosing strata. Also as a rule by-product valuable raw materials (ore as well as nonmetallic) is contained in the sludge (tails) of preparation. As a result in the process of the sludge collector filling and segregation of enrichment wastes particles it takes place concentrating of mineral raw materials with specific grain-size composition. Places of technogenic raw materials concentrations represent as a technogeneous deposit. Operation life of sludge collector depends mainly on its capacity because it has limited dimensions. Because of the lack of additional space for new sludge collectors for capacity increasing during production process dams are built up or their clearing is performed. Sludge collector clearing gives possibility not only to free up capacity for the storage of newly generated wastes but also increases the resource intensity of resource consumption by wastes reprocessing with extract of mineral components.

The only one solution for the problem of the existence and sludge collectors influence on environmental is their elimination by content extracting. In this case there is a need for the implementation (use) or complete destruction of coal washing wastes that should be extracted.

Purpose of the work is to substantiate the method of creating man-made deposits in sludge tanks and technologies for the subsequent extraction of useful components from them to increase the resource potential of the enterprise and reduce the technogenic load on the environment.

1 Genesis of technogeneous deposits

Patterns of formation of technogenic arrays have some similarity to the formation of layers of rocks of the natural geological environment. By its very nature, technogenic arrays are sedimentary deposits, therefore, the stage of storage of waste from extraction and enrichment of minerals corresponds to natural sedimentogenesis. The difference between natural and technogenic sedimentogenesis consists of forces that influence the process - in nature, it is the forces of wind, water (transfer, weathering, erosion) and gravity, and in technogenic arrays - controlled mechanized storage, the transfer of particles by water and wind, segregation, sedimentation.

Since in the process of processing at concentrating factories, different types of rocks containing minerals are substantially crushed, erased, moisturized and mixed, then the resulting mass may well correspond to a new type of sediment.

Sedimentogenesis is considered as the first stage of the conversion of sediments into the rock. The concept of sedimentogenesis as the initial stage of lithogenesis was introduced by M.M. Strakhov, which distinguished it in three stages: mobilization of the initial for sediments of the substance in the crust of weathering, transfer of matter and sedimentation in catchment areas or in final reservoirs of drainage [1].

It is known that the material being stored differs according to the fractional and chemical composition, density and form. As a result of the deposition of each particle occurs in accordance with a certain regularity. Deposition of large particles takes place in accordance with the law of segregation, which in general can be described by the law of the normal distribution of Gauss [2]

$$F(r) = C \cdot exp\left[-\frac{(r - R_0)^2}{2c^2}\right] \quad (1)$$

where C and c are the normalizing coefficients that determine the dynamics of interaction between pieces of different size r and depending on the size of the average piece R_0.

Small, dispersed, and colloidal particles are deposited in a molded array according to the laws of sedimentation. In this case, it is important to determine the distance from which the release of such particles occurs, which is determined by the sedimentation rate of each elementary particle and is described by the Stokes law

$$v = \frac{2r^2}{9\mu}(D_1 - D_2) \cdot g \quad (2)$$

where r and D_1 - respectively, the radius and particle density; D_2 and μ are respectively the density and viscosity of the fluid in which the suspended particles are; g - acceleration of free fall.

Depending on the known rate of precipitation, you can find its duration by expression

$$T = \frac{h}{v} \quad (3)$$

where h is the height of deposition.

Thus, as a result of technogenic sedimentogenesis, technogenic sediments are formed. As a result of physical, chemical and biological weathering accumulated precipitations collapse, change, transform under the influence of the mechanical and chemical effects of the atmosphere, soil and surface water, as well as microorganisms.

When forming technogenic arrays of weathering, as an independent stage of lithogenesis, one can not allocate, because for active sludge it is accompanied by a continuous accumulation of sediments.

For conditions of the formation of technogenic massifs from hydraulic engineering objects (tailing storages), at the stage of sedimentogenesis, the process of coagulation is characteristic, when small particles of disperse systems are combined into larger ones.

Sedimentogenesis in nature is replaced by diagenesis. At the stage of diagenesis, the sediment is compacted under the influence of external pressure of accumulated new layers of sediment, dehydrated, cemented in a bundle with the presence of different size of sediment particles [3]. As a result of the interaction of various mineral and organic substances with water and oxygen, as well as among themselves, a number of chemical reactions, including decomposition, saturation with water, oxidation, enrichment, metabolism occur. At this stage crystallization and recrystallization of sediments, as well as the formation of nodules occurs.

For conditions of technogenic sediment, nodules can be scattered throughout the thickness of the sediment or be concentrated in a certain place. With a significant concentration of nodules containing useful components, they can become the object of industrial development.

The following stages of lithogenesis for technogenic massifs of Ukraine, whose age, as a rule, does not exceed 100 years, are not characteristic, but are possible in the future.

2 Models of creation of technogeneous deposits on the basis of the acting and forming drive of extraction waste and enrichment of minerals

2.1 Choice of the place of formation of technogeneous deposit in the reservoir of the sludge storage tank (tailings storage)

Managed creation of technogeneous deposits is possible due to knowledge of the mineralogical and granulometric composition of the wastes and the patterns of motion of individual particles in the pulp and in the stream of pulp released. In addition, the method of formation of technogeneous deposit is influenced by the way of the production of pulp, the method of weaving and the like.

During the hydrotransport tails in the pulp pipeline are stratified by the size of the coarse: small - in the upper part of the pipe, large - in the bottom.

In the case where the highest content of the useful component in the sludge enrichment accounts for a large proportion, effectively apply the bottom output. At the same time, in the bottom part of the pipeline, at a distance equal to half the diameter of the pipe, an additional hole is provided, which facilitates the deposition of large particles with high content of useful components in the dam [4]. This creates technogeneous deposits.

In connection with this, there is an opportunity to manage the creation of technogeneous deposits by adjusting the location of the production of the pulp in terms of and in relation to the water mirror in the capacity of the waste storage accumulator.

The choice of the method and place of the pulp production is based on the established indicators of the slimes (tails) and content in separate fractions of useful components.

In relation to the water mirror, the pulp can be ground (superficial) and deepened.

In this case, by adjusting the parameters l and h (respectively, the range of release from the dike and the depth of the dipping of the outlet pipe), it is possible in a specific place to create a technogeneous deposit of mineral raw materials for further development.

In the layout, the pulp output *2* can be faceted (Fig. 1, a) and frontal (Fig. 1, b and c) in relation to the dam *1*.

When changing the location of the production of pulp in the plan changes during the flow of pulp *3* (see Fig. 1), which contains particles of different sizes and, respectively, the weight. As a result, zones I, II and III with mineral resources sorted according to their size are formed (large, medium, small and dusty), the boundaries between which are determined by the possible range of R_i of different particles.

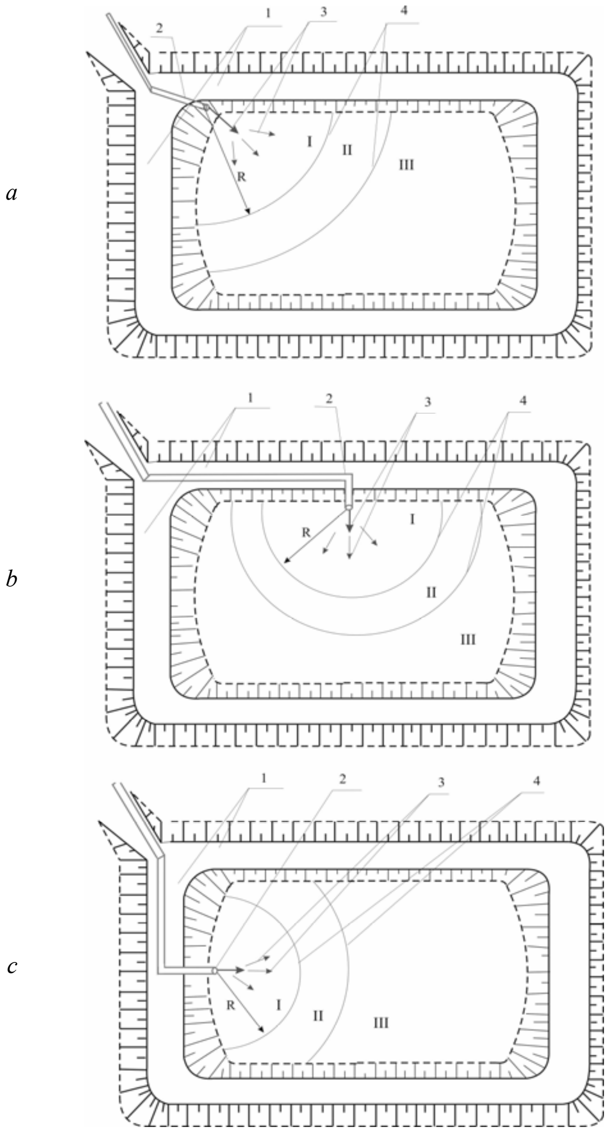

Fig. 1 - Options for location of the production of pulp and distribution areas of material in terms of grain size

Particle spill distance is determined by the following factors:
- the location of the outlet pipe in relation to the water mirror and

the dam - parameters *l, h*;

- the diameter of the pulp *d*;
- velocity of the pulp flow *v*;
- density of pulp (ratio of solid and liquid phases) - *g*;
- the depth of the storage capacity of the waste *H*;
- the deviation of the capacity of the waste container *i*;
- the presence (or absence) of previously constituted waste and their consistency - *x*.

Thus, the dispersion distance of the pulp in the waste container of the waste can be represented as a complex function of all of the parameters listed

$$R_i = f(1, h, d, v, g, H, i, x). \quad (4)$$

Determination of the R_i is accomplished by integrating the function (4) with all arguments.

In addition to these varieties of location of the pulley (Fig. 1) in practice, the option of placing several pulp releases in different parts of the sludge is used. This option is applicable to large and large areas of sludges and when operating their capacity with several enrichment plants.

In this case, when calculating the parameters of the distribution of the particles of the slimes (tails) by the size, it is necessary to take into account the width *B* of the stream of pulp *3* from each pulp output *2* and the zone of intersection of streams *C* (Fig. 2).

The pulp flow rate is determined by its flow rate and velocity

$$B = Q/\upsilon L \quad (5)$$

where *B, L* - respectively the width and length of the flow; *Q* - flow rate; *v* - flow rate.

The values of the intersections of the flows *C* (pos.*4* in Fig. 2) are determined by the distance between the pulse outputs *d*, the width of each B_i stream and the angle of distribution of the flow β_i.

In the zones of intersection of flows, the law of R_i (4) the distribution of particles of tails by size will be broken due to the twist of the flow. In this case, it is necessary to take into account the scheme of the presentation of the pulp on the pulp - a pressure or self-propelled, the height of the location of the pulp waste over the water mirror and slope of the surface of the sludge.

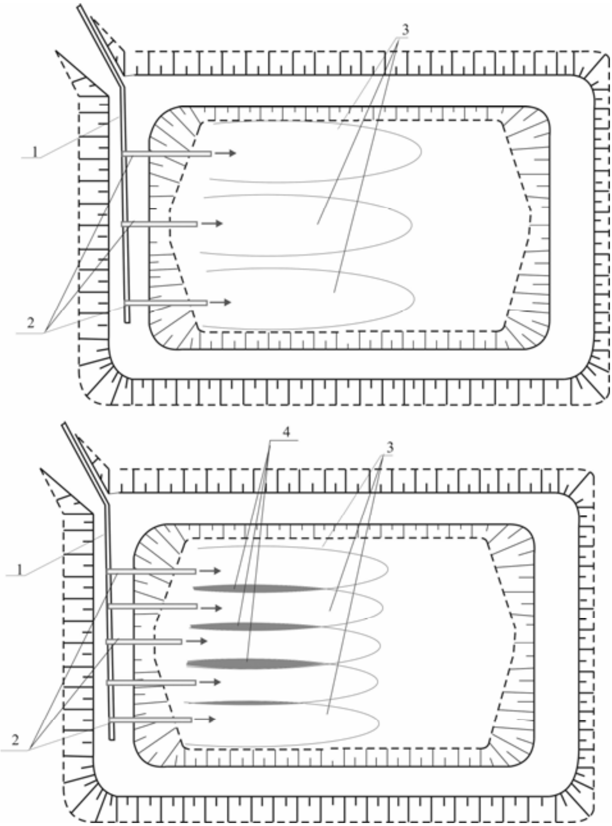

Fig. 2 - The location of several pulp releases and the zone of distribution of flows in the plan

The choice of the place of formation of technogeneous deposit is as follows:

- the mineralogical composition of the waste of the enrichment is determined;
- the parameters of the capacity of the sludge and pulp output are determined;
- fractions with conditional contents of the useful / valuable component are established;
- the type of release is selected;
- the range of pulp flux according to the expression (4) is deter-

mined;

- the rate of precipitation of the precipitate is determined according to the expression (2);

- the deposition time is determined by the formula (3);

- the model of location of the distribution zones of the material according to the size in the capacity of the sludge tailings storage is constructed on the basis of the performed calculations (for example, Fig. 1).

2.2 Formation of technogeneous deposit in one-section tailings storages

Since, as noted earlier, slurry particles (tails) are separated in the flow of pulp according to the size of a certain regularity, then knowing the fraction with the conditional content of the useful component can be regulated by the place of their accumulation, and accordingly the boundary of the t technogeneous deposit.

For conditions where the highest content of the useful component corresponds to the largest fraction of tailings, it is possible to apply the following variants of controlled technogenic formation.

1 option. Used in-depth pulp output. In the bottom part of the pipeline, at a distance equal to half the diameter of the pipe, there is an additional hole, which facilitates the deposition of large particles with high content of useful components in the dam.

Option 2. For a surface run at a distance equal to *D* (6), a droop is formed, the depth of which is determined by the flow rate and the specific gravity of the large particles in the flow by expression (7).

$$D = h \cdot tg(90 - \alpha) \tag{6}$$

where *h* - height of pulp distillation over the capacity of the sludge (tailing storekeeper); α - angle of incidence of the flow; *S* - flat fork; *W* is the capacity of the bump, which is determined by the expression

$$W = \frac{Q \cdot d_{\%} \cdot T}{K \cdot \gamma} \tag{8}$$

where *Q* - flow rate; $d_{\%}$ is the percentage of the contents of large particles; *T* - duration of operation; *K* - coefficient of filling zumpf (0,75-0,8); γ is the average volume density of the skeleton (dry) of the sludge.

For conditions where the content of the useful component corresponds to the average size of the slices (tails), the following scheme of the formation of man-made deposit in the sludge (tailings) is proposed (Fig. 3).

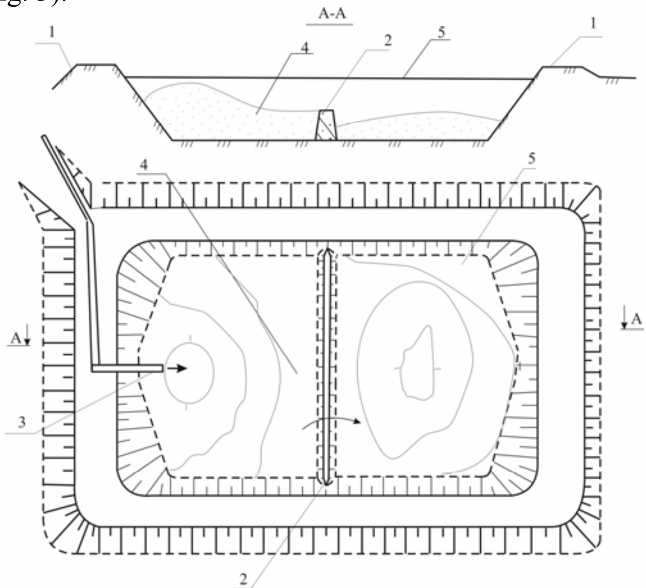

Fig. 3 – Scheme of the formation of technogeneous deposit in one-section sludge storage with the use of a sunken dam

The reservoir capacity is formed by fencing dams *1* (Fig. 3). Perpendicular to the pulp output, the sludge store is divided by a submerged dam *2* with a height of *2/3 h* (where h is the average depth of the drain) at a distance of *1/2 L* (where *L* is the length of the flow of the pulp) from the enclosing dam 1, where the place of release of the pulp *3* is located to ensure formation man-made deposit due to the segregation process.

When filling the sludge storage tank (tailings), as a result of segregation, the average particles of the sludge (tails) with high content of the useful component that forms the man-caused deposit accumulate in front of the submerged dam. Light and small slurry fractions are washed off by a stream of pulp and accumulate over the immersed dam. Thus, anthropogenic field *4* is formed. When the level of alluvial sludge is reached *5*, the filling of the reservoir is stopped.

After this, extraction of a useful component from the capacity of the sludge is carried out. If necessary, accumulated sludge in another part of the sludge storage tank can be cleared for use as building material.

2.3 Formation of technogeneous deposit in two-section sludge accumulators

As active enterprises are acutely concerned not only with the management of accumulated waste, but also the allocation of additional capacities for the storage of current ones, it is proposed, on the basis of the existing and emerging, waste depository of mineral resources, to create an engineering deposit in different sections and develop it (Fig. 4) [5].

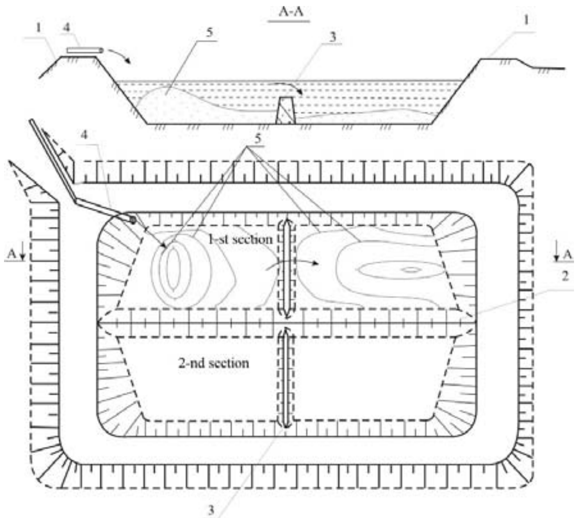

Fig. 4 - Scheme of formation of technogeneous deposit in the first section

For this purpose, the reservoir capacity is formed by fencing dams *1* (Fig. 4). On the length of the reservoir is a partition dam *2*, which divides the capacity into two sections of the section. Each section is transversely divided by a dam *3* with a height of 1/*3 h* (where h is the average depth of the drain) at a distance of 1/6 L (where *L* is the length of the section) from the enclosing dam 1, which places the place of release of the pulp *4* to ensure the formation of technogeneous deposits for account of the segregation process.

In the first stage, the 1st section is filled in (Fig. 4). As a result of segregation in front of the lowered dam *3*, the largest and most heavy particles with a high content of the useful component that forms the technogeneous deposit accumulate. The light and small fractions of the sludge are washed off by a stream of pulp and accumulate over the dam, in most of the section. Thus, an technogeneous deposit *5* is formed. When the level of alluvial sludge *6* reaches the filling of the section of the tailing storage section is stopped, and works are carried out in the second stage. Extraction of the useful component from the capacity of the sludge storage tank is carried out after filling one section. Extraction with the removal of a useful component is carried out in the smaller part of the section. If necessary, the accumulated in most sections of the sludge, can be cleaned for use as a building material.

At the second stage, the 2nd section is filled, and in the 1st section the production of the useful component *5* is carried out after drainage. For this purpose, the production of pulp *4* is sent to the 2nd section.

As a result of segregation, large and heavy particles with high content of a useful component are collected in front of the dam *3*. Thus, technogeneous deposit *5* is formed. Upon reaching the level *6* of the filling section, the sludge is transferred to the filling of the 1st section, the drainage of the 2nd section and the development of technogeneous deposits.

In this case, the capacities of the reservoir are used successively, the preparation of free tanks is carried out without build-up of dams and additional extraction of land, a technogeneous deposit is formed and a useful component is extracted.

Similar models can be developed for different output waste wastes. Depending on the size of the slimes having the highest content of the useful component, the height of the dam *3* (see Fig. 4) and the distance to its location vary.

2.4 Formation of technogeneous deposit in multisection slime accumulators

In multi-sectional sludge storage, the formation of technogeneous deposits can occur in a similar way to a two-sectional (see 2.3) or with the formation in each section of a hump or holes in the bottom part of the pulp for the accumulation of heavy slimes in certain places (see 2.2).

When storing multicomponent sludge multicomponent raw materials (waste wastes containing several useful components, each of which in industrial conditions is contained in the slices of different fractions), then it is promising to split the flow of the pulp before pouring into the capacity of the reservoir.

For this purpose, the following equipment or its complex can be used: robotic, hydrocyclone, classifier. The selected equipment *4* is installed on the dividing dam *2* (Fig. 5). The pulp from the pipeline 3 is fed to the hydrocyclone (classifier, crush) *4* and after separation by pipelines *5,6* by gravity is fed into different sections of the sludge. For example, through the pipeline *5* in the first section is fed sludge class greater than 0.5 mm, and in the 2nd section through the pipeline *6* is fed sludge (tails) classes of less than 0.5 mm.

After filling the equipment sections *4*, it is transferred to another dividing dam and the following two sections are filled. With constant characteristics of the pulp, it is possible to determine in advance the required volume of each section, which will allow them to be filled in at the same time.

The above scheme can also be used in the formation of technogeneous deposits from slimes (tails) that have only one useful component. In this case, one section after the classification is fed a pulp containing the fractions with the highest content of the useful component, and in the second section - all other pulp.

Also, when tailings is separated, a thickening of the pulp can be carried out directly on the dump of the sludge and in the prepared reservoirs it can be stacked in the form of a paste, and the separated water is fed into the third section for protection and lighting.

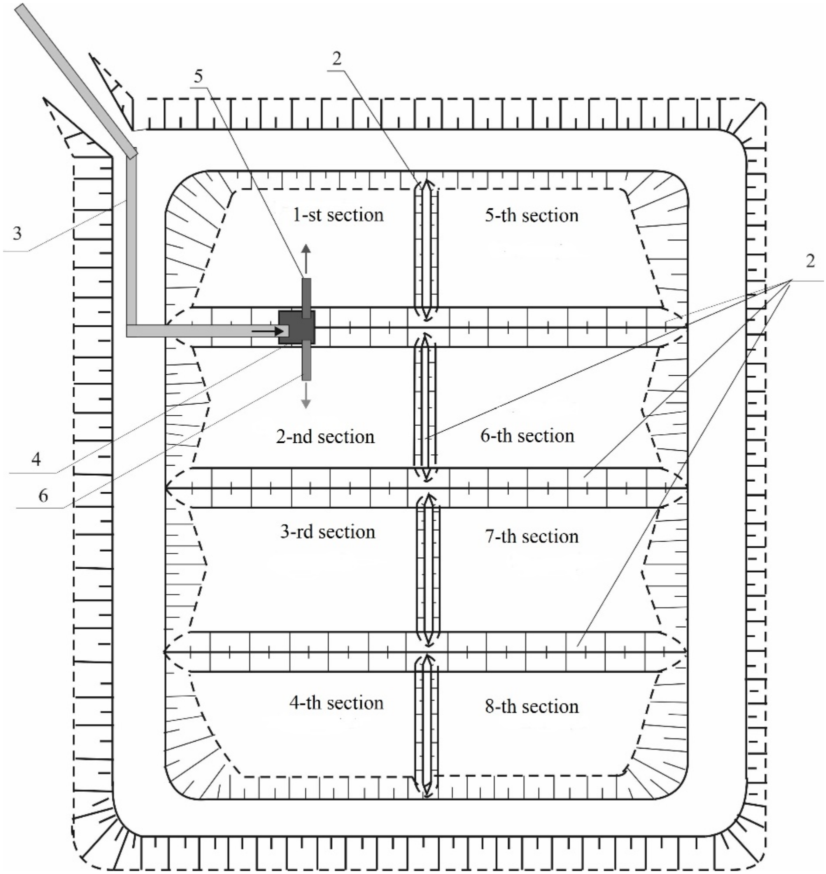

Fig. 5 - Scheme of the formation of technogeneous deposits by separating the pulp into classes on separating dams

3 Development of classification of systems for the extraction of sludge storage devices

According to the characteristics of sludge and their contents, one can draw the following conclusion: the development of slimes has common features in the development of natural deposits by open, underwater and hydroprocessing methods. It should be noted that regardless of the method of removing the useful component of the development system are characterized by such processes as removal, loading, transportation. The following are the main classification features of the systems for the development of reservoirs.

The consistency of the contents is due to the type of removable equipment. In turn, the method of removing causes the type of transport that can be used for the transport of contained sludge storage media.

The content of the sludge can be suitable for use as a building material only if it has the necessary properties and qualities. The same can be said about the recognition of composted slimes man-made deposits - this is possible provided that the accumulated waste of industrial conditions of useful components.

Since the storage of waste sludge enrichment is arbitrarily organized according to the size and content of the useful components, thus forming the highest concentrations in individual places, technologies should be allocated in the technological schemes for the production of sludge with a selective and continuous (gross) removal of the contained sludge storage media.

The stage of use of the sludge, availability, location and form of man-made deposit determines the technology of removing the contents:

- solid, if the sludge does not contain useful components and is used as a building material, or a useful component scattered throughout the capacity,
- selective, if the useful component is concentrated in a definite place;
- solid block (layer), if the withdrawal is carried out in a multi-sectional sludge or in that being exploited.

In accordance with the classification criteria of classification of systems for the development of sludge from slag storage devices are as follows:

- method of removal;
- mode of transportation;
- removal technology.

In fig. 6 shows the classification of technological schemes for the extraction of sludge from slag storage accumulators on the basis of accepted classification features.

It should be noted that the systems of development of sludge storage are classified simultaneously on all grounds, but each category can have only one value. Accordingly, it is possible to determine the possible number of systems for the development of slurry storage systems using the theory of graphs. Since all the classification features of sludge storage systems are interconnected, the pairs of formed graphs (I-II, II-III, III-I) are oriented, bound, not isomorphic. At the same time, the system of graphs I-II, II-III, III-I is congruent.

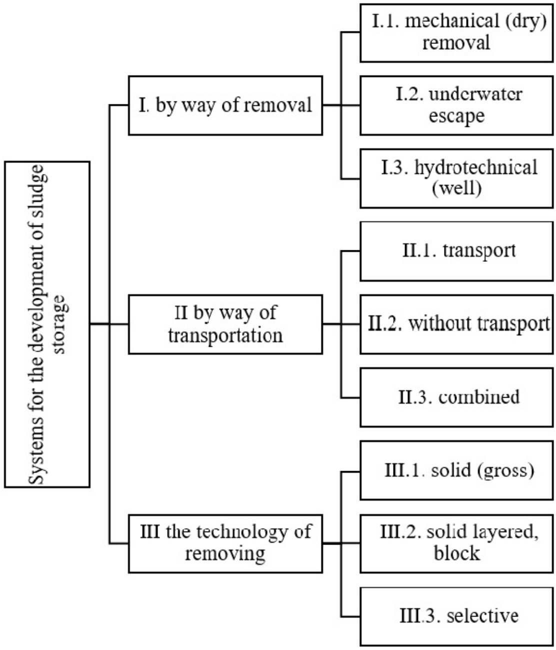

Fig. 6 - Classification of systems for the extraction of sludge storage

Thus, the definition of the degree of graphs is possible by the invariants, the form of which is given in Table 1.

According to the data in Table 1 there are 24 variants of systems for the extraction of sludge storage systems.

The choice of the treatment scheme is based on the classification of the sludge, the selection and alignment of the options for the development systems (Fig. 6, Table 1).

Table 1

Definition of options for systems for the extraction of sludge storage using graph theory

Sub-group systems	II.1			II.2			II.3			Degree of graph
	III.1	III.2	III.3	III.1	III.2	III.3	III.1	III.2	III.3	
I.1	+	-	+	+	+	+	-	-	-	5
I.2	+	-	-	+	+	+	+	+	+	7
I.3	-	-	+	-	-	+	-	-	+	3
I.4	+	+	+	+	+	+	+	+	+	9

Note: + connection available; - there is no connection

Technologies, technological schemes and complexes of machines and equipment for the development of man-made deposits are chosen depending on the form and type of the formed field, the consistency of the content and conditions of transportation to the enrichment plant [6].

4 Technologies for conducting work on the extraction of a capacious sludge dump

4.1. The technology of clearing the sludge storage with division into sections (Fig. 7)

The capacity of the filled sludge storage *1* (see Fig. 7) is divided into sections by temporary dams *2*. The division into sections can be both longitudinal and transverse, depending on the geometric parameters of the tank and the equipment used. Works are carried out in three sections simultaneously.

Waste enrichment (pulp) from the main processing plant *3* through the pipeline *4* is discharged into the tank *5* to its full. Preparatory work for clearing is carried out in advance in tank *6*: ramps are arranged for the operation of the equipment, a layer of dry sludge is removed by an excavator and transported by cars for further enrichment to the processing plant *3*. Wet sludge is extracted with a dredge *7* and fed the mined content through pipeline *8* to the processing plant *9*, which is the preliminary separation of the useful component of the stored waste. The selected useful component is fed to the warehouse *11*, from where the conveyor *12* is transported to the main processing plant *3*.

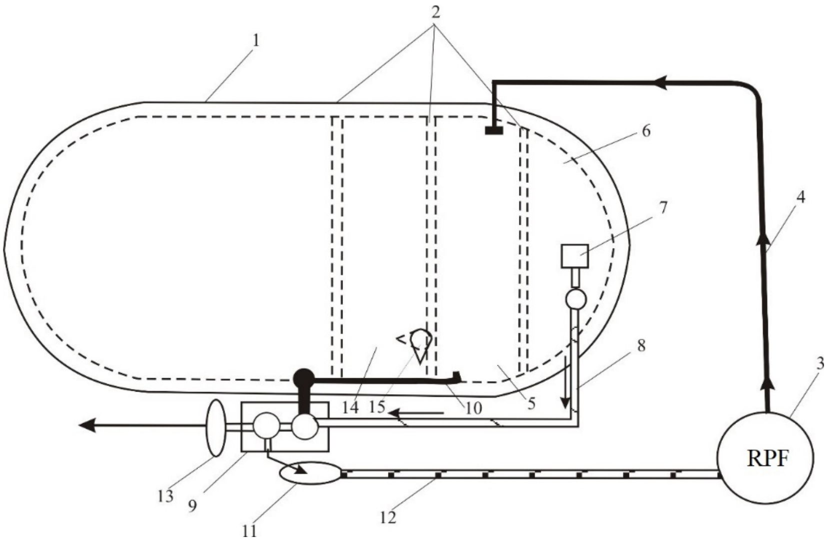

Fig. 7 - Schematic diagram of the clearing of the existing sludge storage with additional enrichment of sludge

The liquid component of the pre-enrichment waste is discharged through pipeline *10* to the tank *5*, and the solid component is fed to the warehouse (bunker) *13*, from where it is transported to the waste rock dump or to reclamation sites . In section *14*, a diking dam *2* is arranged by bulldozer *15* to prepare the section for clearance. The embankment dams are made of dry sludge and tamping, and rolling rollers. As a result of the clearing, section 6 becomes a reservoir for washing slimes, the clearing works are transferred to section *14*. Section *5* is left for natural drainage. The next section is being prepared for development. Thus, the entire area of the sludge storage will be in constant work for clearing and alluvium, which will allow to create additional free containers without land alienation, to obtain raw materials from waste. The device of the beneficiation plant *9* reduces the costs associated with transporting the contents of the sludge storage to the main processing plant *3*, followed by discharge of the waste from the enrichment through pipeline *4* into the sludge storage tank. When the factory *3* is located in the immediate vicinity of the sludge storage facility, the device of the factory *9* is inexpedient and the mined sludge is sent for further enrichment directly to the factory *3*.

4.2 The technology of clearing the sludge storage with simultaneous alluvium

The capacity of the sludge storage is divided into two parts: the part of the alluvium and the production site. Sludge is removed by blocks (panels) on a continuous development system. The parameters of the development system are determined on the basis of ensuring the minimum distances of the equipment and the transportation distances of the mined content to the processing plant.

The mining site is prepared for development as follows: the top layer (0.25-0.40 m) is removed by an excavator and placed in the shoulders with ramming along the border of two parts of the sludge storage to avoid spreading the pulp stream. Water-collecting grooves and drainage trenches are arranged along the development boundaries, congresses for mining equipment and vehicles. The upper (dry) part of sludge ledges (up to 2 m) is developed by auto-loaders for road transport. Sludge mining is carried out with a dragline-type excavator in the bottomhole hole, where they remain until the moisture content permissible for loading into dump trucks is reached. Directly near the exit there is a place for turning trucks and loading sludge into them. The forklift removes sludge in its face and delivers it by road to the car loading area. The work continues until the dragline and the loader lift the refit. Provides shuttle scheme of the forklift. At the same time, the platform for loading and turning cars is not transferred along the work front, since the bottom of the forklift approaches it, and the excavation of sludge by the dragline is removed, but at the same time the minimum transportation distance is ensured. Loading dehydrated sludge into dump trucks can be made by dragline. At the same time, it moves along the front from the bottom of the face and wet sludge to dehydrated. This leads to a loss of its performance.

Then the dragline and forklift make the excavation of the contents in the sludge storage in the next stopping. This creates the next exit and a platform for loading the contents into the sludge storage in dump trucks.

Transportation of the extracted sludge is carried out to the warehouses of the OB.

Depending on the depth of the sludge storage, subsequent excavation of the contents with dredgers or dredges is possible.

When the mining site for sludge mining is fully completed, the jetting stream is transferred to the arranged tank, and the alluvium site is dried and prepared for mining.

4.3 Technological scheme of mining sludge by cards

Depending on the content of the sludge storage and its physical properties, the possibility of developing its maps is very diverse: maps are containers of various shapes, sizes in terms of and depth limited by dams from the solid component of sludge or other rocks.

The map parameters depend on the equipment used for the development, the required degree of excavation, the quality and properties of the mined sludge and can be either small tanks or large ones. The smaller the capacity, the greater the capital costs of developing the sludge storage and preparing additional containers.

This article does not consider any of the possible options in view of the fact that there is no possibility to present any generalizing scheme without specified conditions.

The advantages of using technologies of parallel development of sludge storages with continuous filling with sludges are:

- the environmental damage caused by sludge storage facilities when building dams is significantly reduced;

- improving the environmental situation by maintaining a constant water level in the sludge storage;

no additional land acquisition is required to expand the sludge storage and create additional tanks;

- the costs of creating complex hydraulic structures are reduced;

- the possibility of using the waste after its excavation and dehydration increases;

- the income of the enterprise from receiving raw materials from waste increases.

The disadvantages of technology are the additional costs of equipment (one-time), production and transportation of dehydrated sludge (permanent).

Sludge (tailings) can be used after drying and dewatering:

- for the production of building materials as an additive to the main raw materials;

- as a component of the mixture for road pavement;

- as a building material for building dams, dumping and strengthening the slopes of the ridges;

- for laying in the required stratigraphic sequence for the reclamation of rock heaps and surfaces of quarries.

Conclusions

Thus, as a result of controlled storage of waste of enrichment in the capacity of sludge storage, the occurrence of natural and technogenic processes of lithogenesis and segregation of particles of pulp mixes, it is possible to form a technogenic massif, the properties and mineral composition of which are determined by the raw material, and the position of the deposit of the useful component can be regulated.

Parameters of the sludge, its structure, location of the pulp and technogenic deposits are determined by calculations for the source data by slimes.

The formed technogenic deposit can be developed by a different complex of equipment, the composition of which is determined by the properties of the array.

Thus, at present, it is recommended to clear all existing sludge dumps, to carry out scientific research on the possibilities of reuse of the content.

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IMPROVING MANAGEMENT OF MINING ENTERPRISES ON THE BASIS OF MODERN INFORMATION TECHNOLOGIES FOR THE PURPOSE OF OPTIMIZING THE USE OF RESOURCE POTENTIAL

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Abstract. An important problem in the production activities of mining enterprises operating in complex areas is the need to ensure the necessary technical and economic results of their production activities, which is primarily determined by the efficiency of use and the rationality of the use of production resources. An effective way to solve this problem is to automate the management processes of enterprises based on the use of specialized information management systems. The implementation of these systems in enterprises with a specific type and nature of production is associated with a number of difficulties, the main one of which is the creation and effective use of the information space of the enterprise, corresponding to the nature and specifics of its production activities.

The authors described the principles and basic approaches to the creation of such an information space, described its necessary structural elements, their purpose and the nature of the functions. In the work, a flow chart of information containing the main resource of the enterprise Knowledge is formed, on the basis of which the structure and procedure for managing information flows and a specific professional nature of production resource flows between business units should be developed. The main tasks of the information systems of enterprise management in the conditions of the Information Economy are formulated: the transformation of the organizational structure of the enterprise; implementation of strategic planning based on the forecast of the state of national, international, global markets; decentralization of enterprise management; staff motivation to increase personal competence and efficiency. Based on the described approaches, specific information technologies and management systems should be developed in the future, improving and developing

the information space of an enterprise to bring this space to the level of ensuring the solution of strategic enterprise management tasks in ensuring rational use of resources, competitiveness, and expanding areas of interaction with resource suppliers and the production system and consumers of products of the enterprise.

Introduction. Currently, the mining enterprises of Ukraine operate in difficult conditions, which are characterized by: high dynamism; the influence on the production process of a large number of factors having a stochastic nature; tough market competition; institutional instability; increase in the cost of resources; the difficulties of ensuring the stability of the supply of enterprises with productive resources. The peculiarity of the production activities of mining enterprises in terms of their provision and use of resources is that they have two types of resources. The first type includes the resources that are necessary for the implementation of the production process itself (explosives, compressed air, electricity, drill bits, explosives, etc.), the second type of resource is the natural mineral reserve itself, which also represents the subject labor and production resource, the extraction and processing of which leads to the production of marketable products of the mining enterprise.

This situation leads to the emergence of a particularly acute task for the implementation of the production activities of a mining enterprise, in which the maximum rationalization of the use of production resources must be combined with the rational use of natural resources, which together represent the resource potential of the mining enterprise.

Under such conditions, one of the main methods for solving this complex task is the automation of the management processes of the mining enterprises, since it is the management system that solves all the tasks of rationalization and optimization of the use of resources.

Effective management is one of the most difficult tasks of production management. This is due to the specifics of modern mining production, which includes a large number of interrelated elements (divisions, shops, services, non-production units, direct and reverse links between them). The functioning of this structure requires interaction with a large number of external elements (suppliers, repairmen, consumers).

The organization of a balanced operation of this system with the fullest use of the potential of the resource potential requires the speed of control actions; making management decisions based on a detailed analysis of emerging situations at the system level of production; solving complex planning tasks, designing, organizing, monitoring, correctly assessing the production and commercial activities of an enterprise.

This function is assigned to the *Enterprise Management System* (EMS - *Enterprise Management System*) [1], which represents one of its most important divisions. Automation of EMS (adoption and implementation of management decisions) provides the opportunity to achieve maximum efficiency of the enterprise, in which the optimization of the use of its resource base is one of the main aspects. This is achieved through the use of operational management tools that implement object-oriented *information technology* (IT) forming the *enterprise management information system* (PMIS) [2].

At present, a large number of PMIS based on different principles have been developed; allowing to solve different types of management tasks; ways to implement management functions; the specifics of the formation of the information space of the enterprise.

Along with this, it should be noted that the introduction of these PMIS is associated with one serious *problem*, the essence of which is as follows. The implementation of the PMIS is a complex and lengthy process that requires serious preparation, transformation of the information environment of the enterprise, changes in the principles of the formation and management of information flows, and training specialists in the use of the PMIS.

All these questions relate to the field of formation of the *Information space of the enterprise* [3], which represents the basic component of the PMIS. All information processes in the process of the enterprise are carried out precisely in the information environment using specialized means of work in this space, which are also its elements.

Along with this, the theory of the development and functioning of the PMIS presents, and especially these systems in the field of specific mining production is developed extremely insufficiently, which creates more problems in the field of the functionality of such systems.

Analysis of recent research and publications. A review and analysis of available literature on the theory of functioning of the

PMIS in the field of organization of the information space of enterprises and the management of its elements showed that there is very little information on this issue. The available publications are non-systemic, scattered in nature and mainly boil down to describing the functions and capabilities of the PMIS, but not the functioning of the enterprise information space. Therefore, the issues of structural changes in the enterprise management system, due to the use of the PMIS, remain obscure. This makes it difficult to fully implement the capabilities of existing PMIS and expand their functionality for implementation in specific areas of production.

Objective. In order to eliminate this drawback, the author of this publication conducted research on theoretical issues of implementation and practical implementation of the EMIS capabilities that change management approaches and technology, and allow improving, on this basis, the methodology for managing enterprises in the direction of streamlining resource use.

The presentation of the main material of the work. One of the features of the current stage of development of social production is the formation of the so-called *Information Economy* [4]. This concept implies a form of economic activity, in the management of which modern information and communication technologies lie.

Decentralization of enterprise management, as an element of the information economy. The main direction of the reorganization of the management structures of enterprises in the information economy is primarily to maximize decentralization of management, that is, increasing the role of middle and lower levels in making management decisions, and in the field of mining production the management structure is extremely complex and ramified. This removes the need for solving local tasks (tasks of a relatively low level in the hierarchical structure of management) from top-level authorities, allowing them to focus on solving strategic problems [5].

Such an approach enhances the management flexibility and adaptability of enterprises to rapidly changing conditions while minimizing the losses caused by the process of such changes, and gives greater freedom to the management apparatus of the middle and lower levels in the manifestation of management initiative. However, this also imposes a great responsibility on the effectiveness of the decisions made.

Management processes in the information economy are based on integration processes that ensure interaction between elements of the management structure not only vertically (the head is a slave), but also horizontally (between employees of different departments of the same hierarchy level). These processes generate a new structure, defined by the notion *Enterprise without borders*. This approach transforms the management system from a closed (closed system) using traditional management methods (bureaucratic, hierarchical, mechanistic) to open systems based on *Network methods and management technologies* [6].

The basic task of the PMIS. The purpose of the development and implementation of the PMIS is to automate the solution of tasks related to production management, the rational allocation and use of resources.

Among the enormous aggregate of these tasks, there is one, which by its nature is basic. The special role of this task consists in the fact that the success of the enterprise and the solution of all other tasks absolutely depend on the success of its solution.

This task is to develop a plan of production activities of the enterprise, which must be correct, accurate and realistic to perform.

The development of such a plan and the determination of its parameters should be carried out on the basis of the implementations the provisions of the so-called Deming-Shewhart cycle (PDCA, Plan-Do-Check-Act–planning→action→verification→adjustment) [7] for managing complex systems with the function of continuous improvement activities.

A simplified diagram of the PDCA cycle is shown in Fig.1. This cycle is an algorithm for managing the process and achieving its goals.

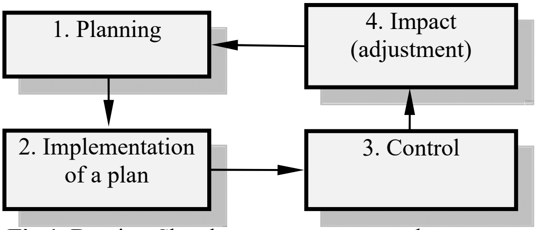

Fig.1. Deming-Shewhart management cycle structure

The PDCA cycle includes a series of sequentially executed processes. Let us explain the essence of the work that should be performed in the PMIS when implementing the PDCA and the tasks that should be solved using this system.

1. Planning of the production process consists in developing a plan in the form of a network model and a Gantt chart and also its mathematical and semantic description.

This process is quite complicated. The scope of work for the development of a network model of the plan is as follows. Before building a model, the entire planned production process should be studied and analyzed.

In the analysis should be taken into account: the purpose of the process for its desired result; the parameters to which the executions result of the process should correspond (the volume of production, the productivity of production, the cost of production, profit, profitability, etc.); scope of work that must be performed to achieve the goal; structure of work (their interrelationship and interdependence).

Then, based on the results of the analysis, a network model of the process should be developed. The network model includes a graphical model (sequence of work and the relationship between them) and a mathematical description of the process (in numerical form – time, material, labor, financial costs). On the basis of the network model, a Gantt chart is being developed - a linear model of time parameters for the implementation of the plan and work. In the PMIS, the Gantt chart is developed automatically based on the network plan already developed, the role of a specialist in the development of this type of model is reduced only to checking the correctness of the chart elements and making the necessary corrections if necessary.

Planning is one of the most difficult stages (according to the nature of the tasks to be solved) and responsible for the results of the implementation of the plan. The complexity of this process is explained by the fact that it is necessary to use the method of variations when forming the network model of the plan. This method consists in shaping the structure of the model by means of “tests” and “errors”, that is, making decisions on specific elements of the plan, calculating the expected results of the plan based on these decisions and checking them for compliance with certain criteria.

Successfully and quickly solve the problem of planning can only be using specialized software tools that allow you to visually form, calculate and test solutions.

At the next stage, the resulting network model should be analyzed and optimized according to the time parameters of the plan implementation, resource costs, and the dynamics of their spending. In this operation, information systems play an especially important role, since the solutions to optimization and resource-use problems are complex due to the specifics of mathematical methods of multi-criteria optimization. Solving the problems of optimizing the use of the resource potential of an enterprise is possible only by automated information and analytical tools.

After receiving the results of optimization and the formation of the final version of the plan with its parameterization, specific tasks are set for the plan's implementers.

It should be noted that the planning process does not end there. Planning operations, or rather plan adjustments, will also need to be carried out at the implementation stage. In this part, the specifics of the work of the PMIS is as follows. During the implementation of complex processes, deviations from the original planning model are usually identified. Such deviations are taken into account, their influence on the general planned model is revealed. If deviations are critical, rescheduling the model.

The revised network model is again analyzed and optimized, and new tasks are brought to the performers. Thus, the planning and management stage continues from the beginning of the implementation of the complex of works to the moment the plan is fully implemented.

Functions of the PMIS. The introduction of a full-scale PMIS consists in the reorganization of the structure of enterprise management with mastering the model of electronic management of production and business. The basis of the management strategy is the expansion and development of communications, as well as new organizational interactions. In this situation, besides the standard functions of ERP and CRM, the following is implemented: EDI (*Electronic data interchange*); the formation of a single chain "supplier - production - consumer"; B2B, B2C, B2G technologies (*commercial relations management systems*); conducting electronic trading (Smart

Tender); electronic payments Internet banking, and the introduction of AMM (*Advanced Manufacturing Management, production management system*) is especially important. Under this option, the management functions are changing and expanding the information space of the enterprise. This approach allows you to create a fully functional automated enterprise management system - *the Automated Enterprise Management System* [8], which is already characterized by a clear specialization.

The role of PMIS in the modernization of production. It is necessary to emphasize the role of the PMIS in addressing such an important issue as the modernization of production [9].

Modern enterprise should not represent a frozen production structure that has mastered a certain technology, technology, organization and management methods, but a dynamically developing and improving system. This need is explained by the fact that a modern enterprise operates in a dynamic environment with ever-increasing demands on consumer properties of products, the tendency of its obsolescence and obsolescence of the means of production, and fierce competition in the market.

Ensuring the competitiveness of enterprises in such conditions requires its periodic modernization based on more advanced technologies, means of mechanization and automation, improving the organization and improving management efficiency, one of the main goals of which is to optimize resource consumption. Lagging behind in this area can have extremely negative consequences, including the crowding out of an enterprise from the market by competitors who more fully control its trends, the development of technology, technology, tools and management methods. Such activity requires the collection, systematization and analysis of huge amounts of information in different directions coming from various sources, which predetermines the scale of the information space of the enterprise.

Problem-oriented PMIS in this area are an indispensable means of operational management, ensuring the possibility of a timely response to changes in environmental conditions.

Network information communications in the PMIS. One of the most important elements of the information space of an enterprise is information communications. The basis of these communications is provided by information networks of different levels [10], namely:

– global networks (*Internet; GENnetwork; WideArea NetWork; Global network*) provide an opportunity for the enterprise to operate at the level of the international division of labor, with global coordination of its work, finding new markets, suppliers, and investors, thus ensuring broad business opportunities on a global scale;

- regional networks (*Datapacnetwork; EURONETnetwork, Fidonetnetwork*) provide coordination of the enterprise's work within a closed structure, for example, within the state, region, industry;

- internal network of the enterprise (*Ethernet*) joint, brigade work, coordination of work outside the boundaries of individual structural divisions of the enterprise, thus reducing the cost of operational management;

- distributed computing network - related working groups, each of which has the necessary knowledge and information tools. Their goal is to streamline business processes and support an optimal level of decentralization of management;

- portable computing (*VO - Virtual Organization*) their work is not tied to a specific location and is mobile. Knowledge and information are delivered to where they are needed at a specific time.

Stages of development and implementation of PMIS. One of the important aspects of the information environment of enterprises is the issue of composition and interaction with information technology (IT), which are implemented in it. The development of this interaction and the change in the composition of information media goes through several stages.

As practice shows, the presence of these stages is explained by the fact that such a complex system as a PMIS cannot be developed and implemented at one time. The development of the PMIS software in its entirety for a particular enterprise, its adaptation to the working conditions of this enterprise and the development of this system by the enterprise personnel requires a certain time and gradual (stage-by-stage) implementation and development by the employees of the enterprise. This is especially manifested in the fact that as you master the understanding of the intricacies of the PMIS and its elements, specialists will become more interested in solving problems of an ever higher level, requiring more complex solution methods, as well as subsystems with greater functionality, which may not be original versions of the PMIS. Thus, there is a tendency to the development of the PMIS and the expansion of their functionality and information space.

The essence of these stages and the level of tasks is as follows.

- at the first stage, the enterprise uses the PMIS, as a rule, only at the level of obtaining operational information about the state of the enterprise and the course of its processes. This information is analyzed by employees of the management system and make management decisions on the tasks of maintaining and developing the future activities of the enterprise;

- at the second stage, the PMIS is already used to analyze situations (in production, market, resource supply) and to propose solutions in specific situations. Management personnel make decisions based on the evaluation of these options, that is, the PMIS is already used in the management decision-making mode [11];

- at the third stage, the PMIS are already oriented toward solving strategic problems in order to ensure the competitive advantage of the enterprise, its adaptation to changing conditions, and monitoring of processes in the external environment. Management employees perform the function of monitoring and adjusting decisions made by the system at the level of an expert in management and for this purpose there are already expert IT [12].

Principles of development and operation of the PMIS. The development and improvement of the management system and the development of the information space of an enterprise should be based on certain principles that take into account the information and communication component of management. These principles are as follows:

1. Information integration, mastering integrated management models.

2. Transformation of the administrative structures of enterprises from structures with vertical integration into structures with horizontal integration, with the minimum number of levels between top management and performers.

3. Decentralization of management functions. Semi-autonomous or autonomous subsidiaries, strategic business units responsible for their production and financial and economic activities are being created.

4. Network forms of interaction between the enterprise and other enterprises, for example, by creating internal markets (network methods of interaction).

5. Standardization of business processes, products, services avoiding narrow functional specialization in the content and nature of management activities and management style.

6. Increasing the competence of staff by equipping them with innovative methods and means of carrying out professional activities and their theoretical and practical training.

The implementation of these principles requires the formation of a single information space of the enterprise, which contributes to the interaction of a large number of subjects involved in the management of the organization of the enterprise.

Information resources and information products of the PMIS. Forming the information space of an enterprise, the PMIS implies the emergence of such a concept as *Information Resources* (IR) of an enterprise [13]. IR is what should circulate in the information space and is subject to processing and management: search, analysis, logical and mathematical processing, transfer, accumulation, storage. PMIS, providing information services to employees of the enterprise, converts the information resources into *information products* (IP). This process should occur systemically. It is the organization of such a system (strict functional sequence and structuredness of the information processing process) that is provided by the PMIS [14].

Thus, the PMIS is a multifunctional cybernetic system that combines control, production and communication services and provides interaction between it. These services are occupied by employees whose activities are subject to management. The management of the services is carried out by the management system of the enterprise, and the PMIS provides for this means and information resources, as well as the means of implementing management decisions.

The structure of the PMIS. One of the most important issues that arises in the process of creating an PMIS for a particular enterprise is the development of the structure of this system. This structure must comply with the scheme of the enterprise management system and take into account the organization of relations within the enterprise and with the external environment. The task of the PMIS is to ensure the functioning of these links at the necessary level of automation.

In fig. 2. shows, developed by the authors, the general scheme of the structure of enterprise management, taking into account the above provisions and demonstrating the work of the PMIS in the field of management of the mining enterprise.

The main element in this structure is the *Enterprise Management System* (EMS), which has its own internal structure and hierarchy.

The EMS solves strategic problems, which requires knowledge and data that are already accumulated in the knowledge base of the information center of the enterprise, as well as information from the external information space. Obtaining all this information and communication is provided by the *Enterprise Information Center* (EIC) and its External and *Internal Information Management Subsystems* (this information may be confidential).

According to the results of information processing in the EMS, a decision is made, on the basis of which the heads of structural divisions form the tasks of their divisions. These tasks are brought to the attention of employees of the link of *the middle management level* (heads of workshops, services, departments). The transfer of this information is also organized by the information center. The leaders of this link themselves must develop the composition and content of the tasks of their units.

It should be noted that they must have access to the external information space and the knowledge base in accordance with the scope of their professional activities in the conditions of decentralization of management and obtaining large rights in management.

After making management decisions at the middle management level, information about these decisions is communicated to the managers of the lower management level (section heads, groups). At this level, management decisions are also made and communicated to the executives.

The task of the information system does not end there. This is explained by the fact that in the process of implementing management decisions, it becomes necessary to monitor the implementation of tasks in accordance with the plans; management decisions adjustments.

This task requires the organization of feedback between the lower level and higher level management link.

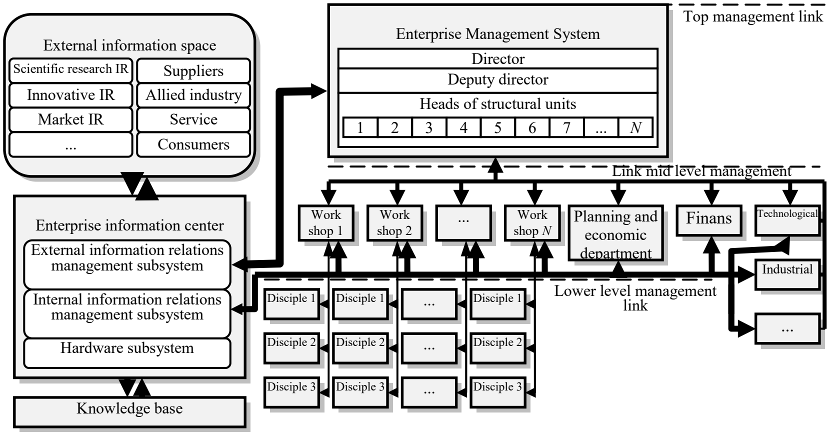

Fig. 2. Scheme of the enterprise information system

Feedback represents a stream of reporting documents, which requires appropriate standardization of documents, timing and procedure for their transfer.

Thus, on the whole, this entire structure, consisting of its elements, types of information being transmitted, the nature of direct and reverse links, opportunities and modes of access to information resources, means of receiving and processing information, forms a full-scale information system of an enterprise and its information space.

Now let's stop another important aspect of the PMIS, which is that these systems are, above all, a powerful tool for knowledge management. It is knowledge that represents the basic resource and the basis for ensuring the competitiveness of an enterprise [15].

The tasks of knowledge management of the PMIS consist in providing each employee with the opportunity to gain corporate knowledge; their preservation; sharing current and retrospective knowledge; providing tools for processing and knowledge management.

Naturally, the PMIS, as a system for managing knowledge and actions determined by the results of their processing (these results also represent knowledge), has a certain basic flow of information containing knowledge. On the basis of this scheme, the ICIS develops traffic patterns for information of a specific professional nature and its movement control subsystem. The structure of the basic scheme is shown in Fig. 3

In accordance with this scheme, the PMIS acts as an intermediary link between the Communicator, generating a certain type of information, the Knowledge Base in the system memory, the External Environment and the Recipient (information consumer). The management system outlines the following contours: A – processing (filtering and entering information into the knowledge base), B – providing information (notification and advertising), D – preparing documents. The knowledge base represents the field of knowledge of employees.

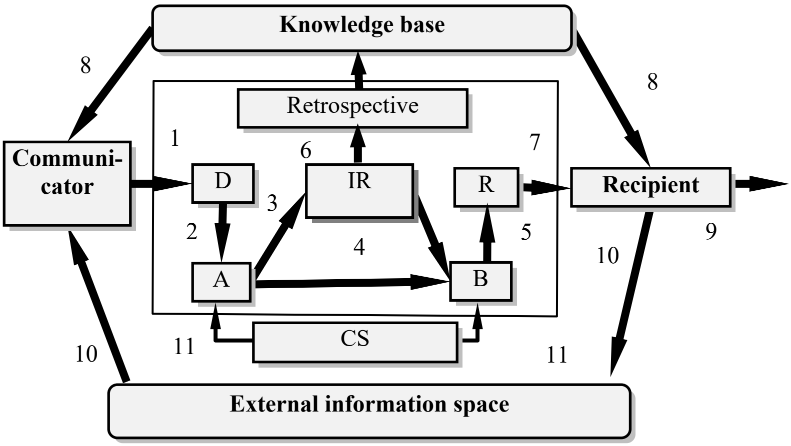

Fig. 3. Information flows diagram representing corporate knowledge

The external environment serves as an object of control and cognition, which is carried out by an employee acting as a Communicator (information flow 10). In order to make their knowledge and conclusions from the analysis of the state of the environment available to workers who need this information, the Communicator draws up its knowledge in a form convenient for familiarization (electronic document) with the means of subsystem D and sends it to the information processing service A (streams 1, 2). Service A in accordance with its algorithm and instructions of the control system (CS) (vector 11) selects the further direction of the document. If the content of the document is recognized as important for the enterprise, the document is recognized as IR (stream 3).

Processing services have hardware and software for automated processing and replication of documents, which are carried out in accordance with the professional knowledge of the staff of this service and instructions of the management body (stream 11). As a result, the processing of documents appear IR for public use (stream 6). IR form the information environment of the enterprise. The socialized information circulates in the contours of the enterprise (streams 8), reaching employees. In if the AI at the moment does not matter much, then it gets into the archive (Retrospective memory) and waits for its actualization. Another information flow is formed by the publication service B. The IR is the object of study (stream 4) by service staff B, who, in accordance with their knowledge and directives of SS (vector 11), form intermediary products (P) (catalogs, abstracts, lists). (stream 5). After that, the service delivers this information to the Recipient (stream 7).

As a result of studying the PI, the Recipient receives new knowledge and generates control solutions that are sent to the appropriate services (9). Thus, there is an increase in intangible assets of the enterprise by generating IR and IP.

Risk management in PMIS. In conclusion, we note the following. Information processing in the PMIS and the content of information flows are associated with two types of events – planned and sudden. Planned events - their type and time of occurrence, as well as the company's reaction to them are planned as planned. Sudden events – it is impossible to foresee. However, under certain conditions, the degree of risk of their occurrence can be determined and even managed by these risks, for this there are special mathematical methods, and corresponding information systems, for example, KGRisk.

What is important in the PMIS is that they allow the enterprise to respond quickly to unforeseen situations and for which, using such information tools, the necessary reserve of resources (financial and material) can be calculated. Addressing issues related to risks in production and business activities currently represents a whole scientific and practical direction [17].

Conclusions and directions for further research. Based on the results of the studies performed, the following conclusions can be drawn:

1. One of the important problems that arise in the process of production activities of industrial enterprises operating in difficult modern conditions is the need to ensure the necessary technical and economic results of their production activities. An effective way to solve this problem is to automate the management processes of enterprises based on the use of specialized information management systems.

2. The implementation of these systems in enterprises with a specific type and nature of production is associated with a number of difficulties, the main of which is the creation and effective use of the information space of the enterprise, corresponding to the nature and specifics of its production activities.

3. The author describes the principles and basic approaches to the creation of such an information space, describes its necessary structural elements, their purpose and the nature of the functions.

4. The work contains a flow chart of information containing the main enterprise resource Knowledge, on the basis of which the structure and procedure for managing information flows of a particular professional nature should be developed by departments of the enterprise and ensuring efficiency and activity.

5. The main tasks of the information systems of enterprise management in the conditions of the Information Economy are formulated: the transformation of the organizational structure of the enterprise; implementation of strategic planning based on the forecast of the state of national, international, global markets; decentralization of enterprise management; staff motivation to increase personal competence and efficiency.

6. Based on the described approaches, specific information technologies and management systems should be developed in the future, improving and developing the information space of an enterprise to bring its information space to the level of solving the tasks of strategic enterprise management to ensure its competitiveness and expand areas of interaction with suppliers, the production system and consumers.

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PHEOLOGICAL PROPERTIES HYDRATES OF THE HYDROCARBON GASES

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Authors studied the features of forming of gas hydrate mass by a compression during an experiment. Expedience carry out of operations forming gas hydrate mass after freezing are substantiated. A considerable manifestation of the creep effect of gas hydrate mass is fixed. Its influence on the parameters of process forming of gas hydrate blocks is investigated experimentally. Manifestation of the deformation creep is finding after apposition a constant force to the gas hydrate samples. Nature of deformation of the gas hydrate sample is established during its formation. The processes that occur during the formation of gas hydrate mass are substantiated. Limits of the rheological parameters such as initial and long deformation modules and viscosity coefficient are established for gas hydrate mass. Rheological model for describing the process of hydrate mass formation as for elastic-viscous body is proposed. In addition, dependence of gas hydrate mass plasticity with a temperature is analyzed. Dependence of influence the time of force application on the absolute deformation of creep for gas hydrate mass is established. Dependence of change pressure necessary for the gas hydrates formation from the time application effort is established.

Keywords: pellets, gas hydrate blocks, gas hydrate, porosity, compression, creep, dissociation, self-preservation.

Introduction

The reserves of natural gas is rapidly exhausted. Nearly 80% of deposits opened into category small and medium-sized distant. However, many there are undeveloped, as traditional transportation technologies are often ineffective. At present gas is transported mainly with pipelines or LNG-tankers and storage - in underground storage. However, in recent years, technology based on the ability of gas molecules and water form gas hydrates, is actively developed. The $1m^3$ gas hydrate contains of gas 160 m^3. In the composition of the gas hydrate, considerable volumes of gas can long be stored at atmospheric pressure and a slight negative temperature (Sloan 1998).

Today the technology of gas transportation in the form of hydrates is considered as a serious alternative to traditional technologies and granular hydrates - as the main form of transportation.

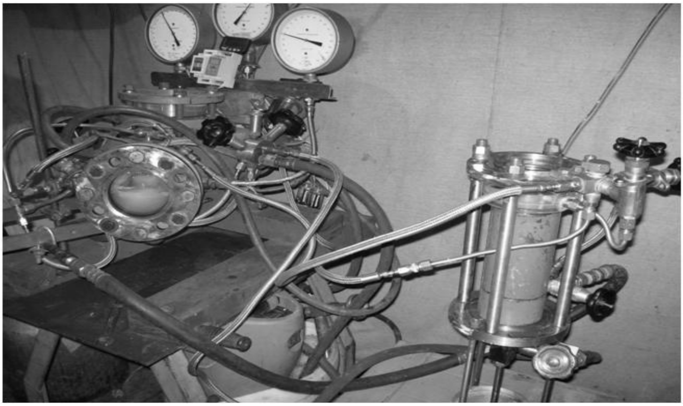
This technology has undeniable prospects for implementation in the near future, but needs improvement and testing of its elements.

At present several concepts the transportation of gas in hydrate form are considered. The technology for the transportation of non-equilibrium conditions (small negative temperature and atmospheric pressure) is the most attractive. It needs the production of gas hydrates in the most stable form under these conditions. At present granules gas hydrate is proposed transport today (Gudmundsson 1996).

However, over time freezing granular hydrates, complicating unloading (Dawe et al. 2003). Also granulated hydrates only 78% fills the volume vehicles or storage (Gudmundsson & Graff 2003). Besides, much of the total surface area of the granules and system of open channels between channels granules stimulates the process of volumetric dissociation of the gas hydrate mass. Keeping them stable at atmospheric pressure needs additional costs for cooling to temperatures below a 258 K. Monolithic blocks of large size are a good option. However, today the industrial technology of production does not exist (Yakushev et al. 2008). Due to thermal properties and features process of transportation and storage, gas hydrates is proposed to produce in the form of gas hydrate of blocks maximum cooled large, preserved with the layer of ice and refrigerated in the production process to the desired level (Gudmundsson & Graff 2003).

This technology provides intensive production synthetic gas hydrates. Moreover, it must have a maximum gas content (up to 160 m^3/m^3). According to the research, the formation of hydrates proposed to perform in the contact devices based on jet devices (ejectors with elongated mixing chamber or jet devices with free stream).Their application allows increasing efficiency and simplify the process of technological design of technology (Pedchenko & Pedchenko 2014). Possibility of organization continuous the gas hydrates production when using the jet devices as contact devices also substantiated and confirmed experimentally (Figure 1).

a



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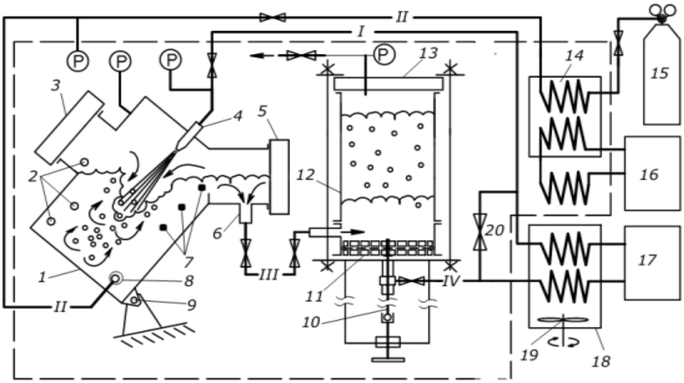


Figure 1 – The laboratory unit for continuous production of gas hydrates: *a* - photos; *b* - scheme; *1* – reactor; 2 – temperature sensors; *3*, *5*, *13* – observation windows; *4* – inject apparatus; *6* – branch pipe; *7* – LEDs; *8* – bubbler; *9* – hinge; *10* – rod; *11* – plunger with filter; *12* – separator; *14*, *18* – exchanger; *15* – gas bottle; *16* – refrigerator; *17* – pump; *19* – mixer; *20* –valve; flows: I, IV – water; II – gas; III – mixture of hydrate and water

In addition, the problem of section of water in the process of separation from the produced hydrates of free water (captured between the crystals and the film) was recorded during the research. Therefore, after the separation the gas hydrates requires draining by transferring of the balance of free water into the composition of gas hydrates (with stirring the mixture is blown cooled gas). Blocks will formed from the previously cooled mixture of crushed and granulated hydrates of minimum porosity in the respective proportion. This solution allows obtain blocks of uniform density and waive the need them cooled (Figure 2).

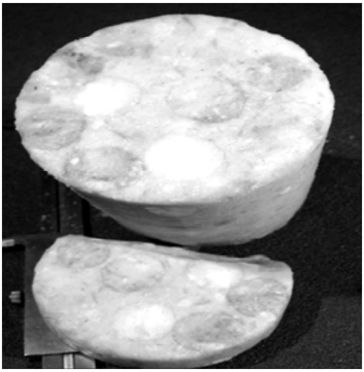


Figure 2 - Transverse section of the gas hydrate block after the formation

The production of gas hydrate blocks, as well as granulated gas hydrate, consists of many technological operations. Among them, their formation is the most responsible.

With the formation of gas hydrate mass external forces perform work directed to breaking the link between the crystals, to reduce the volume of pore space and oppose the forces of cohesion between gas hydrates and ice crystals. However, deformation (rheological) properties of gas hydrate mass in the process its formation are studied insufficiently. Therefore, the purpose of the work was to investigate the mechanical properties of the gas hydrate mass, to check the adequacy of the proposed method in the patent (Pedchenko & Pedchenko 2014) and the production of gas hydrate blocks.

Main part

In the sample internal stresses arises as resistance to external forces. It manifests itself in the form of internal friction and be aimed in the direction of reducing the influence on the sample. Some the work during deformation of the sample is consuming for its heating. Therefore, the residual deformation arises.

Partial the dissociation of gas hydrate the sample occurs during its deformation. In addition, this process occurs and at low and at high temperatures. Destroying the integrity of the crystal lattice and micro-cracks cause the dissociation. The formation of water, the growth of nuclei of hydrate formation occurs at a temperature higher than 273 K. Water migrates to inter-crystalline of space and causes it to fast deformation when compression of the sample (Durham et al. 2003). However, the technology of gas hydrate blocks of large size (Pedchenko & Pedchenko 2014) suggests that their temperature must not be above the 258 K.

Taking into account the physical properties of the gas hydrate, the duration of the cooling process of the manufactured blocks (and the time of their presence in the installation) does not correspond to the conditions for the continuous technological process. Therefore, in the patent (Pedchenko & Pedchenko 2013) it was proposed that the gas hydrate mass will cooled to the time of its formation into blocks. A series of experiments was executed to establish the basic rheological parameters of the gas hydrate mass. We studied the process of formation of the hydrate block under influence of compressive loading.

Rheological properties of hydrate masses we investigated using the device shown in Fig. 3. The experiment performed on the hydrate of natural gas with composition (CH_4 – 92,8% C_2H_6 - 5,1%, C_3H_8 - 2,1%) and an initial temperature of 238 K.

Gas hydrate was made on a laboratory installation, pictured on fig. 1. By the time of experiments, gas hydrate were stored at a temperature of 238 K. Its porosity was at an average of 31%. The accumulated gas hydrate was crushed before the experiment (up to 1 mm) and carefully mixed. Therefore, its initial porosity was 51%.

Samples for the experiment were prepared from crushed (particle size not more than 1 mm, porosity of about 51%) and cooled to 238 K of gas hydrate mass. The cylinder 3 (Fig. 3) was filled with gas hydrate and pre-compacted it to a porosity of 35-40%. Before the experiment, the temperature was raised to 253 K. The sample in the metal cylinder 3 was placed between the pistons 2 and 4 in the device. The piston 2 is connecting with a hydraulic press 1 and the piston 4-5 a dynamometer.

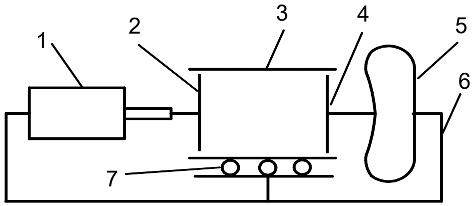


Figure 3 – Scheme of the device for the study of deformation of gas hydrate mass during its formation in blocks: *1* - hydraulic press; *2*, *4* - pistons; 3 - cylinder; *5* - dynamometer; *6* - frame; *7* - mobile support

The parameters of deformation (settling) of gas hydrates in gradually increasing of load and its dependence on the porosity and the initial temperature of the sample were with objects of study.

The experiment method is based on the variation of the parameter *h* (deformation values), at stepped and constant values of the internal voltage σ* as a one-factor experiment. The essence of the experiment was to fix the applied force using a dynamometer 5 and reduce the sample by position the pistons 2 and 4 when compressing the sample.

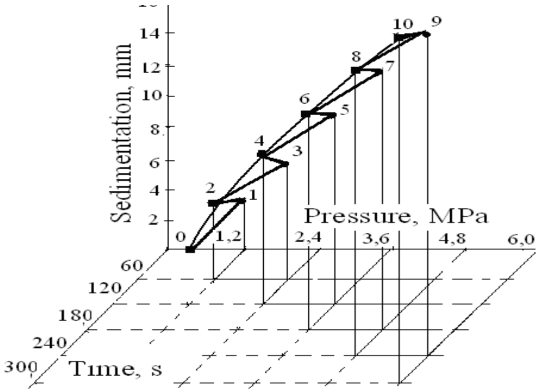
During the experiment, we applied the model to the efforts of 6 to 48 kN with a stepwise increase 6 kN. At each transfer of force of the press to the piston it movement was stopped now when the external force are equal with the internal stress of the sample. After this the piston is connected to a press, we fixed, but inner tension of the sample was equal through the second piston with the springs of dynamometer. In this case, the dynamometer fixed the initial values of effort. After each of rapid (instantaneous) compression through every 10 seconds for one minute, we recorded deformation (settling) of the sample (Fig. 4). After a minute was repeated of compression, increasing efforts on 6 kN.

The results of the experiment we found the dependence of sedimentation of the sample created of the stepwise load

$$P = a \cdot h^{b} = 0.103 \cdot h^{1.3} \qquad (1)$$

where P - pressure, MPa; a and b - empirical coefficients; h - sedimentation, mm.

a



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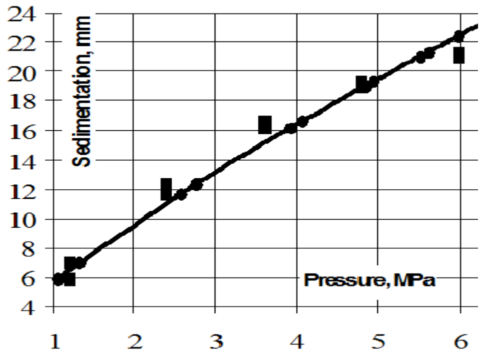

Figure 4 – The deformation of the sample of gas hydrates during the creation of stepwise load: *a* – diagram; *b* – graph

As a result of uniaxial of compressive compression of the samples to the porosity 0,07 their length reduced by 21,4 mm. The rheological diagram compaction of gas hydrates at a constant stress is shown in Fig. 5. (σ^*_i =const). The dependence of stress σ^* of the instantaneous deformation ε_0, has the form

$$\sigma^* = E_0\varepsilon_0 \tag{2}$$

where σ^* - internal stress, Pa; E_0 - module instantaneous irreversible deformation, Pa; ε_0 - instantaneous deformation.

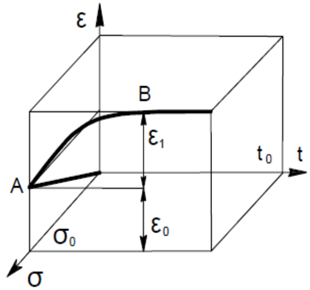

Figure 5 – Rheological diagram of compaction of gas hydrates at constant load (σ_0=const)

As shown in Fig. 4 and 5, damped with time reducing of creep is observed at constant stress at the sample.

Dependence of the relative deformation ε_1 from time τ (Fig. 6) has the form

$$\varepsilon_1 = A' \cdot (1 - e^{-B'\tau}) = 0.0089(1 - e^{-0.076\tau}) \tag{3}$$

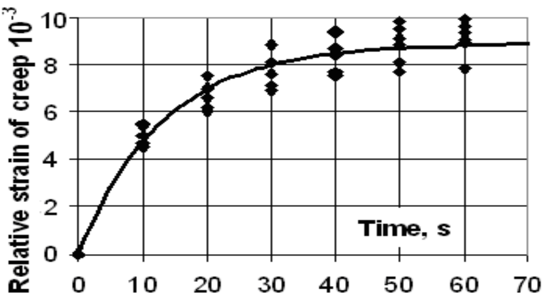

Figure 6 - Relative strain of creep of the sample of gas hydrates with time at constant load

where A', B' - empirical coefficients; ε_1 - relative deformation of creep; τ - time.

The absolute strain creep with time is nonlinear dependence (Fig. 7).

$$h_1 = Ae^{B\tau} = 7.429 \cdot e^{-0.047\tau} \tag{4}$$

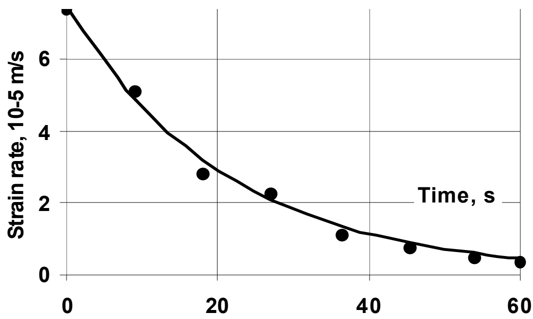

Figure 7 – Experimental and calculated rates of absolute deformation of creep the sample of gas hydrates with apposition of constant load

where *A,B* - empirical coefficients; h_1 - absolute strain of creep, m.

Deformation rate gradually is reducing with time, towards constant value – $0.5 \cdot 10^{-5}$ m/s, which agrees with the results of given in (Durham et al. 2003, Vaysberg 1982).

Analysis of the known of rheological models showed that similar to equation (3) change of deformation with time is described in the model of elastic-viscous of the body the Kelvin – Voigt (parallel connection of elastic and viscous elements)

$$\sigma^* = \varepsilon_1 E_1 + \eta \frac{d\varepsilon_1}{d\tau}, \tag{5}$$

where E_1 - module of prolonged deformations, Pa; η - coefficient of viscosity, Pa·s.

After substitution into formula (3) $\varepsilon_1 = v(\tau) \cdot u(\tau)$ and distribution of variables are

$$\varepsilon_1 = \frac{\sigma^*}{E_1} + C \cdot e^{-\frac{E_1}{\eta}\tau} \tag{6}$$

,

where *C* - the constant of integration, which is determined at point *A* of rheological diagrams with initial condition τ=0, ε_1=0 and is equal to $C = \sigma^* / E_1$.

Then on the interval A-B (Fig. 3) value of relative deformation of creep is

$$\varepsilon_1 = \frac{\sigma^*}{E_1}(1 - e^{-\frac{E_1}{\eta}t}) . \quad (7)$$

Assuming that $A'=\sigma^*/E_1$, (with formulas (3) and (7)) we received module of prolonged creep 674 MPa, coefficient of viscosity 1781 MPa.

General deformation is the sum the instantaneous of deformation and creep

$$\varepsilon = \varepsilon_0 + \varepsilon_1 \quad (8)$$

The artificial gas hydrate is similar to the snow visually and its properties (Vaysberg 1982). Therefore, similar to the model of Hoenemzer - Prager for compaction snow (Economides et al. 2006), the model of compaction of gas hydrate mass can be submit as series connection of linearly deformed and elastic-viscous of bodies (Fig. 8). Equation of the model Hoenemzer – Prager has the form (Zocenko et al. 2004).

$$E_0 n \frac{d\varepsilon}{d\tau} + \varepsilon H = n \frac{d\sigma^*}{d\tau} + \sigma^* , \quad (9)$$

where $n = \frac{\eta}{E_0 + E_1}$ - relaxation time; $H = \frac{E_0 \cdot E_1}{E_0 + E_1}$ - prolonged module of deformation.

In the case of constant pressure σ^*=const equation (9) takes the form

$$E_0 n \frac{d\varepsilon}{d\tau} + \varepsilon H = \sigma^* . \quad (10)$$

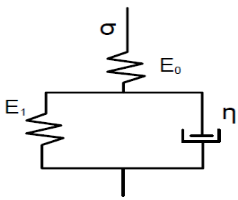

Figure 8 – Rheological model of the stress-strain state of elastic-viscous the body, which describes process of forming of gas hydrate mass

Having solved equation (10), we obtain an expression that describes the dependence of deformation of the body to time

$$\varepsilon(\tau) = \frac{\sigma}{H} + \left(\frac{\sigma}{E_0} - \frac{\sigma}{H}\right)e^{-\frac{H}{En}\tau} . \qquad (11)$$

As follows from formula (11), intensity of the relative deformation at apposition of constant effort decreased from the initial value $\varepsilon(\tau)=\sigma/E_0$ (in case of $\tau=0$) at a rate ($d\varepsilon^2/d\tau^2<0$) to $\varepsilon(\tau)=\sigma/H$, when $\tau\to\infty$, which corresponds to obtained experimental data. Necessary pressure was calculated for formation of hydrate blocks required of porosity and time was set for exposure of applied force (Fig. 9) with using the established of model of viscous-elastic of deformation of body (11).

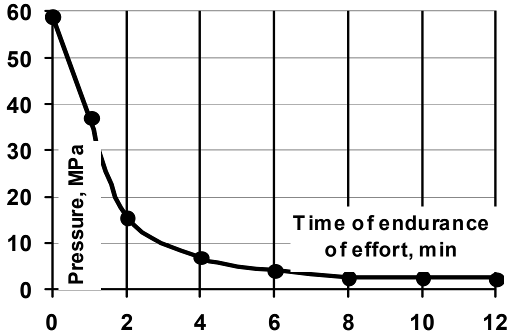

Figure 9 – The dependence of necessary pressure for the formation of hydrate of the sample with time of endurance of effort

As seen from the graph (Fig. 9), the pressure necessary for compaction of the gas hydrates sample (to the porosity 0,08-0,1), significantly is reducing in case of the extension time of application effort. Time of formation of within the limits of 420-540 s is optimal.

In this case, exposure during for 480 s can reduce the pressure of formation of gas hydrates with 57,2 to 2,7 MPa (compared to short-term processes (e.g., the blow)). Length of the sample was reducing by 33-36% only due to deformation of creep. The process occurs without the application of additional effort. This plastic deformation leads to a decrease of elastic and growth of plastic properties. Thus, the quantity of energy required for compaction of gas hydrates decreases by an order.

Following processes occur, during compression of the gas hydrate sample: the destruction the crystal lattice of gas hydrate and ice, the conversion of mechanical energy into thermal energy, dissociation of certain parts of the gas hydrate and separation of gas and supercooled water, absorption of energy in the dissociation certain parts of gas hydrates, the formation (but in a different place) of the gas hydrate and release of energy of hydrate formation.

The predominance of a process depends on the initial temperature of the sample and the gas pressure in the pores.

Conclusions

1. Expedience carry out of operations forming gas hydrate mass after freezing are substantiated.
2. Manifestation of creep deformation for the first time it was discovered and experimentally investigated under conditions of applying constant force to the gas hydrate sample.
3. It was found that during the application of constant force, an instantaneous deformation occurs at first, and then there is a deformation of creep.
4. Rheological model for describing the process of hydrate mass formation as for elastic-viscous body is proposed.
5. Deformation of creep is an important part of the sealing process.
6. The process consists a series of successive phase transitions at the boundary of structural units.
7. It cannot take place immediately and subordinate certain laws.
8. Consideration of creep deformation allows to optimize parameters of gas hydrate blocks formation.

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DEFINITION OF THE GROUND SURFACE DEFORMATIONS AND CONSTRUCTIONS IN THE ROCK BREAKAGE ZONE

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Annotation

The article discusses the issue of improving the efficiency of surveying and geodesic observations of deformations of areas worked up by underground mining. The authors made an analysis of the current state of the issue and based on this, it was proposed to use different approaches to solving the problem, taking into account the features of the observed object. The use of electronic tachometers for performing linear measurements using observation stations to study the processes of displacement of rocks and the earth's surface deserves attention. In this case, benchmarks can be determined from leveling. In studies of deformations in large areas of dumps or quarries, it is advisable to use the method of coordinate reference frames using GPS. The coordinates are calculated not only the magnitude of the deformation, but the direction of displacement. Modern digital methods allow you to explore inaccessible objects that are dangerous to perform work on them by traditional methods. The article provides an example of processing a digital survey made from an unmanned aerial vehicle. The possibility of using this method to monitor deformations has been proved, and to improve the accuracy of determining the values of deformations, it is advisable to improve the accuracy of image processing by including the use of additional characters in standard programs.

Introduction

In 2010, there was a collapse of the ground surface in the mine of Ordzhonikidze, which was undermined by underground mining, which caused human losses and material losses. To prevent further similar incidents, for this purpose, and near the lying areas, a project has been created to upgrade the observation station, which is being improved every year. It should be noted that the observation station in this area has existed since 1954.

A modern observation station includes 20 profile lines that run along the earth's surface and main buildings. 123 ground points are fixed with metal rods 150-180 cm long. In addition to ground benchmark, which are used to study the ground surface, sixty wall marks were laid, which serve to observe the deformations of the buildings of a substation, a garment factory, and other important objects located in the same territory.

Instrumental observations are conducted in accordance with the requirements of the instructions [1]. It is known that the classical methods involve the implementation of linear measurements of the distance between the benchmarks and the measurement of the excess between the benchmarks. At first, geodesic roulettes were used for linear measurements, then a range finder, and now an electronic total station. The excess between the benchmarks is determined by the results of high-precision leveling.

Leveling on closed passages is performed in one direction, and leveling on open passages is in the forward and reverse directions. The permissible discrepancy in closed and double passages does not exceed $15\sqrt{L}$ mm, where, as is known, L is the travel length in kilometers. As a result of the leveling of the leveling passages, it was established that the actual values of the residuals are less than f_{per}.

The lengths of the intervals between ground benchmarks are measured with an SET 630R total station with a standard error m_S = 2 mm. The centering of the total station is carried out with the help of optical centers, and the centering of the reflector using the tripod. The lengths of the lines are measured in the forward and reverse directions, which provides a relative error in the measurement of the length of the intervals between the benchmarks of the profile lines no coarser than 1:10000.

According to the results of the research performed, it was established that it is impossible to perform the work strictly following the provisions of instructions that are twenty or more years old. Old documents are focused on the use of routine instruments and inefficient measurement processing technologies. Considering the modern achievements of science and technology, it is necessary to constantly improve the classical methods of performing work, increasing their efficiency.

1. The use of electronic tachometers in the observation of deformations

One of the significant achievements in geodetic instrument making is the development and manufacture of an electronic tachometer. Electronic tachometers allow increasing the efficiency of instrumental observations of various deformations, including the displacement of rocks.

With the help of the total station SET 630R, work is carried out to determine the length of the intervals between the benchmarks with sufficient accuracy. In addition to the dignity of this device with respect to its accuracy, it can significantly reduce the time taken to make measurements. It is often possible to choose a station in the alignment of the profile line, from which all the benchmarks are visible, and from one such point to measure the lengths of all intervals. Using a total station allows you to determine the length of the intervals from a point located not in the alignment of the profile line. To do this, it is necessary to perform linear and angular measurements, which are convenient for solving the given task of the station, which are necessary for determining the lengths of intervals from linear-angle constructions.

Based on the equal influence of linear and angular measurements, with a relative accuracy of linear measurements of 1:10000, it is assumed m_β=20" at m_S=2 mm. The SET 630R total station has an angular measurement accuracy of 6".It was previously established that to determine elevations with a mean square error $m_h = \pm 3$ mm at m_β=6″, S= 0 m and at tilt angles from 1° to 10°, the lengths of the lines should be measured with an accuracy of 15 cm to 2 mm, which allows electronic total station SET 630R.

Considering the difficult conditions of the territory where works are performed to monitor the deformations of the ground surface and buildings located near the collapse zone, it was decided to use other methods in parallel. Therefore, special polygonometric strokes were laid along the profile benchmarks. Prior to this, observations on the reference lines of the profile line were performed using the measurement of distances and elevations and the values of horizontal and vertical deformations were determined.

Using this technique, the results of observation of deformations on the reference points did not exceed the critical values [2].

As a result, the determination of the coordinates of the benchmarks on the initial and subsequent dates, they found significant values of deformations that exceed the critical ones. The calculated differences of the coordinates of benchmarks characterize the magnitudes and directions of displacement for a certain period of time.

In practice, there was a case when the distance between the benchmarks from adjacent series of observations changed by 12 cm. and the coordinates of the benchmarks, between which this distance was determined, changed respectively by 76 cm and 57 cm. If the coordinates of the benchmarks had not been determined, then such significant values of the benchmark offset would not have been detected. In complex areas of land that are undermined by underground mining, such cases are not alone.

When performing work on the observation of deformations of the earth's surface and buildings in the territory located near the collapse zone, the electronic SET 630R total station was used to determine the planned-high-altitude position of the profile line reference points.

In practice, the correctness of the selected measurement characteristics was confirmed when studying the deformations of the ground surface and structures located above or near the developed space. The method of measuring the lengths of intervals or their determination from linear-angle constructions using electronic total stations was applied on the profile lines of the observation station in the territory in the mine of Ordzhonikidze in Kryvyi Rih [3-5].

2. Coordinating working benchmarks with GPS

As mentioned above, in practice there are often cases when classical methods for monitoring the movement of rocks do not give reliable results. Coordinating the benchmarks of the profile lines of an observation station on specific dates of various series of observations allows to obtain a complete description of the deformation processes occurring in a given area.

When using GPS to determine the X, Y, Z coordinates, the known formulas for calculating the values of horizontal and vertical deformations are respectively transformed [6]. Linear elements - the lengths of the intervals, which are used in the formulas, are calculated from the coordinates of the frames, and the directions of the deformations are determined from the results of the analysis of coordinate increments and from which the directional angles of the directions are calculated.

Previously, the authors considered the issue of improving the efficiency of work on monitoring the deformations of the ground surface and the structures located on it, by constructing special networks in the form of quadrangles or triangles, at the vertices of which there are deformation frames. This allows to obtain the strain values for large areas, and not only in the directions of the laid profile lines.

3. The use of digital photogrammetry in deformation studies

Recently, there has been significant technological progress in the field of digital technologies, which has led to a significant improvement in the quality of digital cameras and their resolution.

A positive factor affecting the widespread use of digital imaging techniques in mining is the improvement in digital image quality, along with a significant decrease in the cost of digital cameras. As a result, the surveying service received an effective measuring tool that can be used to solve a wide range of surveying tasks for providing open-pit mining at a higher level.

Particularly noteworthy is the variety of programs that are designed to perform filming with unmanned aerial vehicles. The programs allow you to process data from surveys, create 3D models, calculate the volume of mineral extraction and the area of objects.

The advantages of many programs are that they can be used in conjunction with any unmanned aerial vehicle and process images taken with any DSLR or GoPro digital camera; allow you to get the final result with high speed and accuracy; just importing images from the camera into the program for processing; after constructing a three-dimensional digital model, you can calculate the volume of a certain area and solve other tasks.

Aerial photography of the area, which is carried out by unmanned aerial vehicles, is today relevant and cost-effective in solving a wide range of tasks of mine surveying and mining. In 2017, the EBEE SENSEFLY unmanned aerial vehicle was presented on the territory of the career № 2 of CGOK, LLC TNT-TPI. Based on the results of the survey, pictures were taken of a part of career № 2.

The processing of survey results was carried out using several software products, one of which is Agisoft Metashape Professional. Due to the large amount of data obtained as a result of a quarry survey, there are certain requirements for the processor and the graphic card. For processing the obtained images, the processor power, the characteristics of the graphics core and the video card are important. Snapshot processing was performed using the fourth generation 4790K Intel® Core ™ i7 processor.

Despite the high performance of the processor, the processing lasted more than 12 hours. Due to adverse weather conditions, of the 210 images, 130 were taken for processing. Fig.1 shows the characteristics of the camera.

Type	Resolution	Focal Length	Pixel Size
Frame	**5472 x 3648**	**10.2 mm**	**2.4 x 2.4 μm**

	Value	Error	F	Cx	Cy	B1	B2	K1	K2	K3	K4	P1	P2
F	**4416.46**	0.12	1.00	0.03	-0.62	0.01	-0.00	-0.13	0.04	-0.03	0.04	0.02	0.28
Cx	**-32.2697**	0.048		1.00	-0.04	-0.08	-0.04	0.02	-0.02	0.02	-0.02	0.69	0.02
Cy	**-9.93983**	0.049			1.00	-0.02	-0.08	0.02	0.02	-0.03	0.02	-0.04	0.13
B1	**0.804109**	0.02				1.00	0.00	0.00	-0.01	0.01	-0.01	-0.02	0.03
B2	**0.059513**	0.021					1.00	-0.02	0.03	-0.02	0.02	0.01	-0.05
K1	**-0.101064**	8.8e-005						1.00	-0.96	0.91	-0.85	0.01	-0.06
K2	**-0.351346**	0.00067							1.00	-0.98	0.95	-0.00	-0.02
K3	**0.674133**	0.0019								1.00	-0.99	-0.00	0.03
K4	**-0.353447**	0.0018									1.00	0.01	-0.04
P1	**-0.000786635**	1.7e-006										1.00	0.00
P2	**-0.000304893**	1.9e-006											1.00

Fig. 1. Elements of camera calibration and correlation matrix

In fig. 2 shows a digital model with a color characteristic of heights.

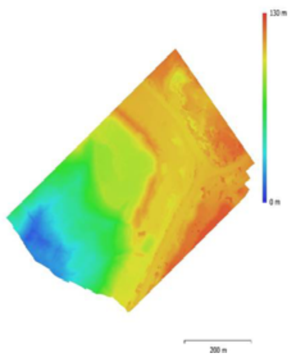

Fig. 2. Digital model with a density of 161 points per m^2

The graphic window can be represented as a sparse or dense cloud of points, as a model or a tile model. In fig. 3-5 are some perspectives of the resulting model.

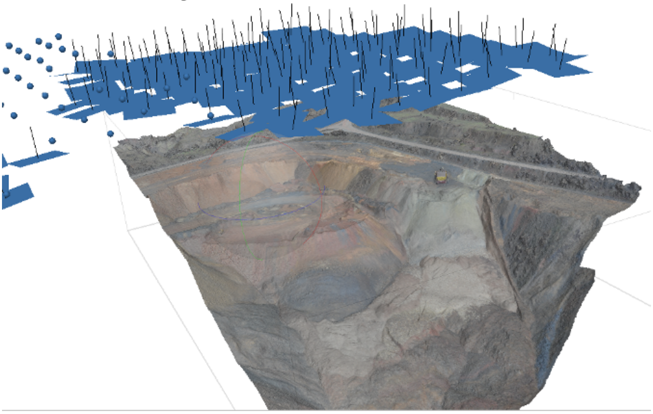

Fig. 3. Perspective 1

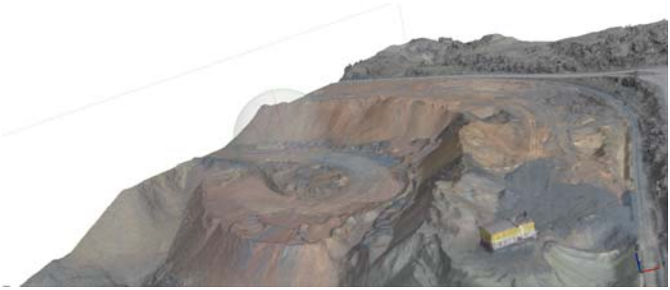

Fig. 4. Perspective 2

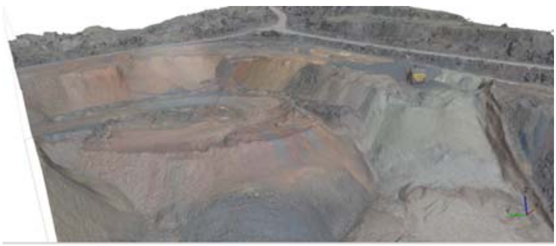

Fig. 5. Perspective 3

In fig. 6 shows a model with contours.

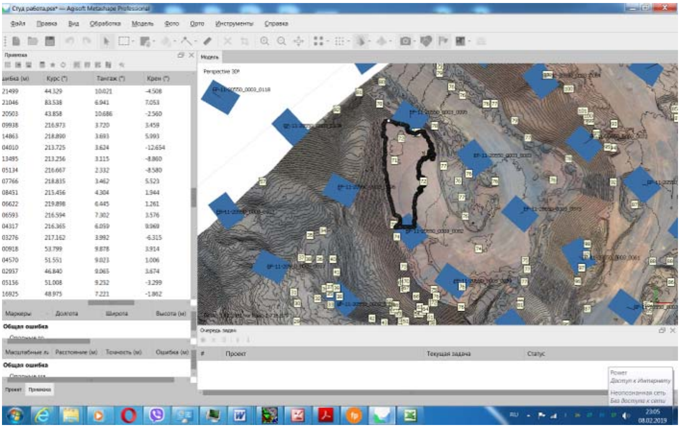

Fig. 6. The model combined with contour

The program also allows you to build a model in the form of contour or orthophotomap with a binding file. The use of digital imaging, made from an unmanned aerial vehicle, can be used to monitor deformations in areas that are dangerous for people to find on them (landslides, collapse zones, etc.).

Conclusions

Currently, unmanned aerial vehicles are widely used in remote sensing, in drawing up and updating plans, and in solving a wide range of mine survey tasks. The widespread introduction of this type of survey in mine surveying is facilitated by the use of a GPS system installed on board a UAV, which allows you to perform a planned reference with centimeter accuracy. It does not always require a planning and high-rise justification on the set, which is important when working under the conditions of the open method of field development.

The disadvantage is the dependence on weather conditions and insufficient accuracy for observing deformations, the magnitude of which is less than the accuracy of the determinations. In addition, the equipment has a low reliability.

When creating plans on a scale of 1: 500 and when observing deformations, digital imaging can be used in conjunction with satellite and tachometric surveys.

The problem of improving the accuracy of digital shooting from unmanned aerial vehicles can be solved by improving the image processing process, which should be based on the use of a sufficient number of identifications.

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ASSESSMENT OF THE EFFICIENCY WORK OF SURFACE MINERS OF THE MILLING TYPE ON IRON ORE PITS

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Subject research. A significant part of the costs associated with ore mining belongs to drilling and blasting operations. Therefore, one of the ways to improve the performance of mining enterprises is to improve drilling and blasting operations or the use of a non-blasting method of preparing rocks for excavation using surface miners. To obtain the correct results of the use of surface miners, it is necessary to assess their effectiveness in the conditions of open development of iron ore deposits, as an alternative to drilling and blasting.

Methodology. The analytical method of research, which takes into account the influence of different indicators on the effectiveness of the use of combine layer milling, is used. Based on the mathematical method takes into account the patterns and dependencies that enable you to compare the two ways of weakening the rock massif, covering the theoretical and practical parameters of their work.

Purpose. To evaluate the efficiency of milling-type surface miners on the basis of their comparison with traditional drilling and blasting operations in the development of half-rocky rocks and rocky rocks.

Conclusion study. The presented method of obtaining an assessment of the efficiency of surface miners in the development of iron ore, makes it possible to compare the non-blasting method of rock development and the method of softening the rock massif using drilling and blasting. On the basis of the obtained assessment it follows that the implementation of the combine method of mining has lower energy costs and better performance of mining operations.

Introduction

Increasing the intensity of production of iron ore requires reducing the cost of technological processes of mining timely to perform drilling and blasting operations. Therefore, there is a need to improve and develop more significant technological and technical solutions regarding the method of preparation of rocks for excavation.

In recent years, due to the environmental situation of mining enterprises and the constant increase in energy prices, career harvesters around the world have become increasingly important [1-12].

Experience in the operation of milling-type mountain combines in the development of rocks with a fortress coefficient of up to f=15 with the presence in the rock mass of various inclusions with a strength factor of f>20 on the scale of M. Protodyakonov, which indicates the possibility of using them quite effectively in iron ore pits.

Research and solving problems on the effective use of technology of layer-by-layer milling of rocks requires the study of the nature of the influence of physical and mechanical properties of rocks on the working body of the combine and the conditions for the development of rocks combine layer-by-layer milling.

Complex geological and mining conditions for the development of iron ore quarries when reaching significant depths of working off the rock mass with the use of combines layer-by-layer milling improves the technical and economic performance of the open pit.

To evaluate the efficiency of milling-type surface miners on the basis of their comparison with traditional drilling and blasting operations in the development of half-rocky rocks and rocky rocks. Achieving this goal is formed on the consideration and solution of the following tasks:

to carry out the analysis of preparation of rocks for excavation by explosion;

to analyze the destruction of rocks by open pit combines of milling type and the nature of rock softening taking into account their physical and mechanical properties;

to analyze and justify the indicators that affect the formation of a qualitative assessment of the effective use of career combines milling type.

To determine the quantitative characteristics of the reliability of modern removal and loading equipment need statistics that can be obtained in the implementation of the operation of the combine or take the results of the analysis studies.

Obtaining statistical data of harvesters is problematic due to the lack of the ability to test them in the right operating conditions and the difficulty to obtain statistical data during bench tests, due to the lack of alternative materials.

The analytical method of research, which takes into account the influence of different indicators on the effectiveness of the use of combine layer milling, is used. Based on the mathematical method takes into account the patterns and dependencies that enable you to compare the two ways of weakening the rock mass, covering the theoretical and practical parameters of their work.

Further evaluation of the effectiveness of the application of surface miner milling type will be formed on the basis of the work of such scientists as: Yu.I. Anistratov, I.A. Tanaeva, I.J. Repik and L.I. Baron.

Some theoretical and practical results of operation of career combines of layer milling at the mining enterprises are considered. The analysis of the obtained characteristics of harvesters in the conditions of open development of mineral deposits, indicates a sufficient level of effective use of milling combines and makes it necessary to justify the production technology of layered milling in iron ore pits.

The definition rational type surface miners

Based on the calculation of energy costs, it is possible to determine the power of the power plant of a milling type surface miner, which is required to obtain the required size of a piece of softened rock. To determine the specific energy consumption required to perform layer-by-layer milling of rocks by a combine, it is necessary to compare them with the specific energy consumption for the destruction of rocks by roller drilling machines. We use the Kirpichev-Kik law, the formula of which has the following form, kV·h/m^3

$$E_{sm} = \frac{E_b}{ln(\frac{d_{ave}}{d_b})}, \tag{1}$$

where d_{ave} - the average size of a piece softened rock milling combine, mm;

E_b - specific energy consumption of roller drilling, kV·h/m^3;

d_b - particle size softened rocks rolling cutter drilling wells (d_b=5 mm).

The value of the energy intensity of rock drilling roller machines SBSH-250, according to research I.A. Tangaev, is a characteristic of the strength properties of rocks and is determined by the formula, kV

$$N_b = N_p + N_x, \tag{2}$$

where N_b - total engine power needed for the weakening of the breed of the rotary drilling machine, kV;

N_p - the required motor power for the weakening of the breed of the rotary drilling machine, kV;

N_x - idle power system of the drilling rig (N_x=8÷10 kV at the SBSH-250MH), kV.

When the formula (1) is transformed, it will have the following form, kV·h/m^3

$$E_b = \left(\frac{E_p}{k_u} \right) + \left(N_x \cdot \frac{t_b}{V_{wel}} \right), \tag{3}$$

where E_p - the specific energy absorption of the breeds in the process of performing drilling, kV·h/m^3;

t_b - the time of the drilling of the well, h;

V_{wel} - the volume of the borehole, m^3;

k_u - efficiency of energy use in the performance of roller drilling (about 0.6%).

Determination of the value of E_p is performed by the formula proposed by Yu.I. Anistratov, kV·h/m^3

$$E_p = 2{,}78 \cdot 10^{-7} \left(\frac{\sigma_{com}{}^2}{2E} \cdot lg \frac{d_{wel}}{d_p} + g \cdot \rho \cdot \frac{l_{wel}}{2} \right), \tag{4}$$

where σ_{com} - the tensile strength of rocks in uniaxial compression, Pa;

E - modulus of elasticity, Pa;

ρ - rock density, kg/m^3;

d_{wel} - bore diameter (d_{wel}=250 mm when using SBSH-250), m;

d_p - the average size of sludge particles in rolling cutter drilling (d_p=0.005 m), m;

l_{wel} - length of the blast hole (for iron ore quarries l_{wel} = 18 m at the height of the ledge 15 m), m;

g - acceleration of gravity (g=9.8), m/s^2.

Determine the volume of the well according to the following formula, m^3

$$V_{свр} = 0{,}25 \cdot \pi \cdot d_{wel} \cdot l_{wel} \,. \tag{5}$$

We determine the duration of the net drilling of one well by the formula, h

$$t_b = \frac{l_{wel}}{v_b}, \quad (6)$$

where V_b - technical speed of drilling, m/h.

To determine the value of the drilling speed of the well, you can use the formula B.N. Kutuzov

$$v_b = \frac{14400 \cdot P_o \cdot n_o}{P_k \cdot D_d}, \quad (7)$$

where P_o - the value of the axial force on the bit in the implementation of drilling soft and medium strength rocks, MPa;

n_o - the frequency of rotation of the drill bit, s^{-1};

D_d - bit diameter of the machine SBSH-250, m;

P_k - contact strength of the rock, proposed by L.I. Baron, MPa.

When using drilling rig SBSH-250 accepts the following parameter values: P_o=0,22 MPa; n_o=3,0 s-1; D_d=0,244 m.

To determine the value of the P_k can be calculated using the empirical dependence of the proposed N.Ja. Repin, MPa

$$P_k = 1{,}9 \cdot \sigma_{com}^{\;1{,}5}. \quad (8)$$

In the future, substituting the corresponding values of the parameters in the formulas (1-8), we obtain the values of the specific costs of the development of iron ore put surface miners. According to the results of calculations, the specific energy intensity of rock softening by surface miner will be determined. Taking into account the specific energy consumption of E_{sc} and the technical performance of the milling combine Q_{sc}, it is possible to determine the necessary power of the power plant by the following formula, kV

$$N = 1{,}15 \cdot E_{sc} \cdot Q_{sc}. \quad (9)$$

Speed of movement of the milling combine at performance of working off of a layer of breeds of weight can be defined by the formula, m/h

$$V_w = \frac{N}{1{,}15 \cdot E_{sc} \cdot B_s \cdot h}. \quad (10)$$

We choose career combines of milling type 4200SM, which have N=1193 kV, B_s=4200 mm, h=0.1÷0.6 m.

Determination productivity milling-type surface miner

An important indicator of the successful operation of the mining enterprise is the maintenance and improvement of the level of productivity of mining equipment, as it affects the further coherence of the complex of technological processes of mining.

Taking into account the necessary data, we determine the performance of the milling combine according to the following formula, m^3/h

$$Q_h = \frac{60 \cdot h \cdot B_s \cdot L_w}{T_o + \frac{L_w}{V_w} + T_t + T_n}, \tag{11}$$

where h - the depth of the rock milling layer, m;

B_s - maximum width of rock milling entering, which corresponds to the length of the milling drum of the quarry combine, m;

L_w - the average length of the front of the working stroke of the surface miner, m;

V_w - speed of movement of the surface miner performing layer-by-layer milling of the massif of rocks, m/min;

T_o - the duration of lowering the working body of the surface miner to the beginning of the layer-by-layer milling of rocks combine (T_o = 0.5), min;

T_t - it takes time for the implementation of the rotation surface miner in the opposite direction to continue testing of rocks (T_t = 6÷8) min;

T_n - the duration of raising the working body of the mining combine prior to performing layer-by-layer milling of rocks combine (T_n=0.3), min.

To determine the operational performance of a mining combine during the change, we use the following formula, m^3/ch

$$Q_{ch} = Q_h \cdot T_{ch} \cdot k_1 \cdot k_2 \cdot k_3, \tag{12}$$

where T_{ch} - the duration of the work shift, h;

k_1 - the factor of time spent on maintenance aggregates mining combine (k_1 = 0.83);

k_2 - factor of the account of expenses of time for implementation of replacement of cutters of the working body of the combine (k_2 = 0.95);

k_3 - time factor for installation and replacement of dump trucks for loading (k_3=0.7).

To determine the annual production capacity of a mining combine is possible by the formula, t/year

$$Q_{year} = Q_{ch} \cdot N_{ch} \cdot N_d \cdot k_u, \tag{13}$$

where N_{ch} - number of shifts per day;

N_d - number of working days in a year;

k_u - coefficient of use of surface miner during the year (k_u = 0.75).

After determining the annual productivity of a milling type surface miner in the specific conditions of the development of iron ore, the required number of combines is determined to ensure a given performance of the open pit.

Comparison of rock mining methods

Application of technology of layer-by-layer milling of rocks by surface miners in comparison with traditional technology of preparation of rocks for excavation by blasting has characteristic advantages and disadvantages.

Comparison of two methods of rock mass softening can be made on the basis of taking into account the energy absorption of rocks, which is necessary for the actual softening of rocks under the influence of the destruction factor [13,14].

Specific energy absorption in the performance of explosive crushing and formation of the collapse of the softened rock massif can be determined by the following formula, J/kg

$$E_b = \left(\frac{\sigma_{com}^2}{2 \cdot E \cdot \rho} \cdot lg\, n^I + \frac{l_{wel}}{2}\right) \cdot N, \tag{14}$$

where σ_{com} - tensile strength of rock to uniaxial compression in the implementation of drilling, Pa;

E - modulus of elasticity, Pa;

ρ - density of rock, kg/m^3;

l_{wel} - well depth, m;

N - the ratio between the volume of drilling operations and the volume of the blasting unit.

The resistance of the rock to compression, and N is determined by the formula, m

$$n^{I} = \frac{d_{wel}}{d_p}, \tag{15}$$

where d_{wel} - borehole diameter, m;

d_p - diameter piece softened rocks in the drilling process, m.

Specific energy absorption in the destruction of rocks can be determined by the following formula, J/kg

$$E_v = \frac{\sigma_{com}^2}{2 \cdot E \cdot \rho} \cdot lg\, n^{II} + \Delta + l_c, \tag{16}$$

where Δ - the specific energy providing the necessary degree of softening of the blasted rock mass for effective excavation, J/kg;

l_c - the amount of offset of the center of gravity of splitting array of blasted block to the center of the collapse of the rock mass, m.

In the case of using the blasting to prepare the rocks for excavation, the destruction of the rock mass occurs as a result of overcoming the resistance of the rock to stretching σ_{str}. Therefore, the energy absorption of the process of destruction of rocks is proportional to the degree of grinding of rocks, namely the ratio of the size of the cracks of the natural separation of d_{sep} to the desired average size of a piece of softened rock d_{awe} for their effective excavation

$$n^{II} = \frac{d_{sep}}{d_{awe}} \tag{17}$$

The value of LC is determined by the following formula, m

$$l_c = \frac{(c + h \cdot ctg\alpha) \cdot (k_p \cdot h - h_p)}{2 \cdot h_p}, \tag{18}$$

where c - the distance between the first row of wells and the upper brow of the ledge, m;

α - the angle of slope of the ledge, degree;

k_p - coefficient of loosening of rock massif;

h_p - the permissible height of the collapse of the rock mass in the face, m.

We determine the ratio between the volume of drilling operations and the volume of the explosive block according to the formula

$$N = \frac{V_b}{V_{bl}} = \frac{n_{wel} \cdot S_{wel} \cdot l_{wel}}{h \cdot B \cdot L_{bl}}, \tag{19}$$

where V_b - drilling, m^3;

V_{bl} - the volume of the explosive block, m^3;
n_{wel} - the number of wells of the explosive block;
S_{wel} - area of the blasting well, m^2;
h - the height of the ledge, m;
B - width of the blasting block, m;
L_{bl} - length of the blasting block, m.

The no-vibration method for the development of rocks by milling combines requires specific energy absorption of E_{cut}, which is determined by the following formula, J/kg

$$E_{sc} = \frac{\sigma_{cut}}{2 \cdot E \cdot \rho} \cdot lg\, n^{III} , \qquad (20)$$

Combine how to develop a rock mass fulfills the weakening of rocks due to the resistance of rocks under shear σ_{cut}. Moreover, the degree of grinding of rocks n^{III} is determined by the ratio of the size of the separation of the array to the average size of a piece of waste rock, which is an essential condition for the effective use of mining rock massif reinforced cutters working body of the milling combine.

Conclusions

The results of the study indicate that the presented method of obtaining an assessment of the efficiency of milling combines in the development of iron ore, makes it possible to compare the method of development of rocks and the method of softening the rock massif using drilling and blasting. On the basis of the obtained assessment it follows that the implementation of the combine method of mining has lower energy costs and better performance of mining operations.

Further research will be devoted to the implementation of the introduction of milling type surface miners in the development of iron ore pits on the basis of appropriate technical and economic calculation, taking into account all possible features of the mining enterprise.

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GENERALIZING SOLUTIONS OF APPLIED PROBLEMS OF THEORY OF ELASTICITY

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Abstract
On a basis of argument functions method and a method of functions of a complex variable, generalizing solutions of a plane problem of theory of elasticity in a classical formulation are obtained. Boundary conditions in stresses are analyzed. Use of a trigonometric substitution that connects integral characteristics of a stressed state with components of a stress tensor is shown. A fundamental substitution is suggested. Argument functions of basic variables are introduced. When substituting into differential equations, operators are formed, which are characterized by these argument functions and that perform a role of special search regulators. As a result of this, dependencies of existence of solutions in a form of the invariant Cauchy-Riemann conditions and Laplace's equations are determined. Getting a new result is associated with a complication of a problem, by adding additional variables to a solution structure. The solution uses generalizing relations in a differential form for specific functions - functions of harmonic type.

Introduction

There is a necessity of their theoretical and experimental ground at working mine of minerals, planning and constructing of mining machines and mechanisms. Using the substantive provisions of geomechanics, prognostication of the stress state of soils comes true in the conditions of all-round compression and realization of road heading technology. In the process of exploitation of mining machines and complexes there are a lot of considerable local and general load. It results in unpredictable breakages of the used equipment. Estimation of strength characteristics in the conditions of static and dynamic influence of separate details and complex of extractive mechanisms is related to determination of working tensions, by their ability to react on the changing terms of external loading. In this connection, in a theoretical plan aspiration appears in being of decisions in the theory of elasticity, various regional tasks related to realization. Such statement of problem can be carried out with the use of generalizing approaches at being of decisions of tasks of mechanics of continuous body. In literature such generalizing tendencies are known.

Analysis of literary data and problem statement

Most papers are devoted to a determination of a stress state in zones of elastic loading. The main approaches to their determination are formulated in papers [1]…[7]. In the analysis of papers on the theory of elasticity, it has been shown that many studies are aimed at identifying and studying generalizing forms and approaches of finding conditions for the existence of a complex of solutions of equations satisfying different boundary conditions. Such generalizations include structures of solutions of problems [8], integral ratios for estimating kinematic perturbations [9], closing parameters determining the general form of a gradient solution [10], interaction problems, on the basis of generalized theory of elasticity [11]. In many papers the conditions of integrability of systems of linear differential equations for partial derivatives are shown, in particular, this concerns paper [12]. It should be emphasized that in paper [12], not the final result is shown, but the integrability conditions that determine the general approaches to obtaining a set of solutions that satisfy different variants of applied problems. In papers [13], [14] differential ratios that impose restrictions on sought functions, expanding the possibilities of obtaining the final result, are defined for certain types of differential equations. The authors showed not the result, but the conditions of its existence. The same approach was used in papers [15], [16]. Unknown argument functions are introduced, which, according to the conclusion, close the problem with certain restrictions characterizing the solutions in a form of the Cauchy-Riemann conditions.

An analysis shows that such generalizing approaches in the decision of tasks of theory of plasticity, elasticity are possible with the use of the offered method argument of functions.

Appears expedient to get certain generalizations of the decided flat tasks of theory of elasticity so that to educe the terms of existence of decisions as the differential correlations, which are restricted on the different required function.

Methodology of research and development of theory of elasticity with the use argument of functions complex variable

A well-known formulation of a plane problem of a classical theory of elasticity is used. It includes: two differential equations of equilibrium, an equation of joint deformations, boundary conditions in stresses in trigonometric form. Unknown argument functions are introduced into consideration, both for fundamental and trigonometric basic functions, which simplifying a decision and close the solutions of a plane problem. In this case, the simplifying trigonometric and fundamental substitutions must be confirmed in the process of obtaining of the final result. The differential ratios between the argument functions, shown by the problem solution, represent the Cauchy-Riemann conditions that occur in analytic functions of complex variables, i.e.

$$\theta_x = -A\Phi_y, \theta_x = A\Phi_x$$

where θ_x=-$A\Phi_y$, θ_x=-$A\Phi_x$ - partial derivatives of argument functions by coordinates.

Going over to the second derivatives and summing, elliptic Laplace's equations are obtained

$$\theta_{xx} + \theta_{xx} = 0, \quad A\Phi_{xx} + A\Phi_{yy} = 0$$

An original invariant connection that is fixed in basis of mathematical transformations with the use of method of complex variables is established between real functions. The feature of this approach is that the correlations got as a result of conclusion allow to lock a decision, bringing him over in accordance with border terms.

Suggested solutions, as a rule, begin with a problem statement and formulation of boundary conditions. For a plane problem

$$\frac{\partial \sigma_x}{\partial x} + \frac{\partial \tau_{xy}}{\partial y} = 0, \quad \frac{\partial \tau_{yx}}{\partial x} + \frac{\partial \sigma_y}{\partial y} = 0, \quad \nabla^2 (\sigma_x + \sigma_y) = \nabla^2 (2\sigma_0) = 0 \quad (1)$$

Boundary conditions in stresses can be reduced to the form [3]

$$\tau_n = -\frac{\sigma_x - \sigma_y}{2}\sin 2\varphi + \tau_{xy}\cos 2\varphi \quad (2)$$

where σ_0 - average normal stress or hydrostatic pressure.

It is necessary to underline that the given above the statement of elastic problem is acceptable both plane-strain and plane-stress states of material, [2]. An analysis shows that with the purpose of simplification of decision and border terms it is necessary to take advantage of trigonometric substitution of kind:

$$\tau_{xy} = T_i \cdot sin\left(\mathrm{A}\Phi\right), \qquad \sigma_x - \sigma_y = 2 \cdot T_i \cdot cos\left(\mathrm{A}\Phi\right), \quad (3)$$

that in future must be confirmed by the solving of problem and satisfaction of boundary conditions. Putting (3) in (2), we have:

$$\tau_n = -T_i \cdot sin\left(\mathrm{A}\Phi - 2\varphi\right), \quad (4)$$

where $T_i = T_i\left(x, y\right)$ - function of coordinates of the deformation zone, coinciding in terms of functional purpose with the intensity of tangential stresses; A – a constant coefficient that determines the elastic state of a deformed medium; Φ – a coordinate function characterizing contact tangential stresses, and one of the introduced argument functions; φ – tilt angle of a cross-section.

To consider that the system of differential equations is linear, it is possible to use a fundamental substitution at determination of intensity of tangential stresses, [17]. It should be noted that in paper [17] the linear dependence on coordinates is accepted as the index of quantity in exponent. It is suggested to enter second argument function θ, i.e. index of exponent, as an arbitrary continuous function of coordinates, the value of that on this stage of decision is unknown. As a rule, elastic strain determines the stress state such at that stress intensity is the size of variable. In this connection it is expedient to enter in consideration the three-dimensional factor of influence, related to the coordinates of strain zone, i.e.:

$$T_i = C_\sigma \cdot exp\left(\pm\theta\right) = C_\sigma \cdot \left[ch\left(\theta\right) \pm sh\left(\theta\right)\right]. \quad (5)$$

It is necessary to add that an index of exponent θ is unknown dependence and is the second argument of function. Considering remarks (3-5) we have:

$$\tau_{xy} = C_\sigma \cdot exp\left(\theta\right) \cdot sin(\mathrm{A}\Phi), \ \sigma_x - \sigma_y = 2 \cdot C_\sigma \cdot exp\left(\theta\right) \cdot cos\left(\mathrm{A}\Phi\right). \quad (6)$$

In expressions Eq. 6 two basic functions are considered, trigonometric and exponential, and two unknown argument functions (θ and $\mathrm{A}\Phi$), which in many ways are defining in solving the problem. If there is a mathematical connection between them, for example, the Cauchy-Riemann condition, then it becomes sufficiently certain to obtain an analytical solution and a possibility of establishing the condition for existence of a solution of the system of equations in Eq. 1.

Boundary conditions show that the difference in normal stresses Eq. 6 must have a specific mathematical content, which is confirmed by the solution of a plane problem of theory of elasticity. According to Eq. 1 and Eq. 6, considering the deviatoric component for normal stresses [18], the following can be written down

$$\sigma_x = -\int \frac{\partial \tau_{xy}}{\partial y} dx + \sigma_0^{'} + C, \qquad \sigma_y = -\int \frac{\partial \tau_{xy}}{\partial x} dy + \sigma_0^{'} + C,$$

then

$$\sigma_x - \sigma_y = -\int \frac{\partial \tau_{xy}}{\partial y} dx + \int \frac{\partial \tau_{xy}}{\partial x} dy = 2 \cdot C_\sigma \cdot exp(\theta) \cdot cos(\mathrm{A}\Phi).$$

This is possible if

$$\int \frac{\partial \tau_{xy}}{\partial y} dx = -C_\sigma \cdot exp(\theta) \cdot cos(\mathrm{A}\Phi), \; \int \frac{\partial \tau_{xy}}{\partial x} dy = C_\sigma \cdot exp(\theta) \cdot cos(\mathrm{A}\Phi),$$

or

$$\begin{aligned} \sigma_x &= C_\sigma \cdot exp(\theta) \cdot cos(\mathrm{A}\Phi) + \sigma_0^{'} + f(y) + C, \\ \sigma_y &= -C_\sigma \cdot exp(\theta) \cdot cos(\mathrm{A}\Phi) + \sigma_0^{'} + f(y) + C. \end{aligned} \tag{7}$$

Requirements in Eq. 3...Eq. 7 are imposed on a solution of the problem from the side of boundary conditions in Eq. 2.

From these positions consider average normal stress, which is used in the equation of joint deformation. This parameter should be given particular attention because it is present in a formulation and a solution of the problem. Considering Eq. 7, the following can be written down

$$\begin{aligned} \sigma_x^{'} + \sigma_y^{'} &= C_\sigma \cdot exp(\theta) \cdot cos(\mathrm{A}\Phi) + \sigma_0^{'} + C + \\ &+ \left[-C_\sigma \cdot exp(\theta) \cdot cos(\mathrm{A}\Phi) + \sigma_0^{'} + C \right] = 2\sigma_0^{'} + 2C. \end{aligned}$$

If $2\sigma_0^{'} + 2C = 2 \cdot \sigma_0 = 0$ or is equal to a constant, $2 \cdot \sigma_0 = const$ then the equation of continuity of deformation is identically satisfied. However, these are not the only options for solving the equation of continuity. The integral ratios $C_\sigma \cdot exp(\theta) \cdot cos(\mathrm{A}\Phi)$, shown above in Eq. 7, pose interest.

By analogy with the stress circle of Mohr, deviations of the stress state are possible along abscise axis toward negative or positive values, due to average stress σ_0. In order to the process of deviation was obvious, it is necessary to present hydrostatical pressure in a comparable kind with the "kernel" of decision, in Eq. 6-7. To set, at what constraints the argument of functions will be satisfied equation of continuity of strains.

The necessity to determine the conditions of existence of a solution for Laplace's equation appears

$$\nabla^2\left(\sigma_0\right)=\nabla^2\left(C_\sigma\cdot exp\left(\theta\right)\cdot cos\left(\mathrm{A}\Phi\right)\right)=0. \tag{8}$$

Analyze dependencies in Eq. 6, Eq. 7, of the average normal stress by solving the problem in a statement Eq. 1. We have σ_0 in form

$$\sigma_0=C_\sigma\cdot exp\left(\theta\right)\cdot cos\left(\mathrm{A}\Phi\right).$$

Results of research of the stress state of elastic body

We will return to the system of differential equations.

Considering that in Eq. 6, Eq. 7 there are exponential dependencies, including complex, write the tangential stress through a function of a complex variable, in a form

$$\tau_{xy}=C_\sigma\cdot\frac{exp\left(\theta+i\mathrm{A}\Phi\right)-exp\left(\theta-i\mathrm{A}\Phi\right)}{2i}\ . \tag{9}$$

To determine normal stresses, it is necessary to substitute the expression for tangential stresses Eq. 9 into equilibrium equations. Partial derivatives have a form

$$\frac{\partial\tau_{xy}}{\partial y}=C_\sigma\cdot\frac{\left(\theta_y+i\mathrm{A}\Phi_y\right)\cdot exp\left(\theta+i\mathrm{A}\Phi\right)-\left(\theta_y-i\mathrm{A}\Phi_y\right)\cdot exp\left(\theta-i\mathrm{A}\Phi\right)}{2i},$$

$$\frac{\partial\tau_{xy}}{\partial x}=C_\sigma\cdot\frac{\left(\theta_x+i\mathrm{A}\Phi_x\right)\cdot exp\left(\theta+i\mathrm{A}\Phi\right)-\left(\theta_x-i\mathrm{A}\Phi_x\right)\cdot exp\left(\theta-i\mathrm{A}\Phi\right)}{2i}.$$

After substituting the derivatives into differential equations of equilibrium and separating the variables, the following is obtained

$$d\sigma_y=-C_\sigma\cdot\frac{\left(\theta_x+i\mathrm{A}\Phi_x\right)\cdot exp\left(\theta+i\mathrm{A}\Phi\right)-}{2i}\dots$$

$$\dots\frac{-\left(\theta_x-i\mathrm{A}\Phi_x\right)\cdot exp\left(\theta-i\mathrm{A}\Phi\right)}{2i}dy.$$

Using a condition of analyticity of functions in parentheses

($\theta_x=-\mathrm{A}\Phi_y$), ($\theta_y=\mathrm{A}\Phi_x$) the following is obtained

$$d\sigma_x = -C_\sigma \cdot \frac{\left(A\Phi_x - i\theta_x\right)\cdot exp\left(\theta + iA\Phi\right) -}{2i}\ldots$$

$$\ldots\frac{-\left(A\Phi_x + i\theta_x\right)\cdot exp\left(\theta - iA\Phi\right)}{2i}\cdot dx,$$

$$d\sigma_y = -C_\sigma \cdot \frac{\left(-A\Phi_y + i\theta_y\right)\cdot exp\left(\theta + iA\Phi\right) -}{2i}\ldots$$

$$\ldots\frac{-\left(-A\Phi_y - i\theta_y\right)\cdot exp\left(\theta - iA\Phi\right)}{2i}dy.$$

At such statement of a question integrands are written in one variable. It can be shown that

$$A\Phi_x - i\theta_x = \frac{\theta_x + iA\Phi}{i}, \qquad A\Phi_x + i\theta_x = -\frac{\theta_x - iA\Phi_x}{i},$$

$$-A\Phi_y + i\theta_y = -\frac{\theta_y + iA\Phi_y}{i}, \qquad -A\Phi_y - i\theta_y = \frac{\theta_y - iA\Phi_y}{i}.$$

After substituting the ratios obtained above and integrating

$$\sigma_x = C_\sigma \cdot \frac{exp\left(\theta + iA\Phi\right) + exp\left(\theta - iA\Phi\right)}{2} + C,$$

$$\sigma_y = -C_\sigma \cdot \frac{exp\left(\theta + iA\Phi\right) + exp\left(\theta - iA\Phi\right)}{2} + C.$$

Going over to real dependencies, there are

$$\sigma_x = C_\sigma \cdot exp\,\theta \cdot cos\,A\Phi + C, \qquad \sigma_y = -C_\sigma \cdot exp\,\theta \cdot cos\,A\Phi + C. \qquad (10)$$

If integration was used not to determine stresses, but stress deviators $s_x = \sigma_x - \sigma_0$, $s_y = \sigma_y - \sigma_0$, according to [18], then:

$$\sigma_x = C_\sigma \cdot exp\,\theta \cdot cos\,A\Phi + \sigma_0,\ \sigma_y = -C_\sigma \cdot exp\,\theta \cdot cos\,A\Phi + \sigma_0, \qquad (11)$$

where $\theta_x = -A\Phi_y$, $\theta_y = A\Phi_x$, $\theta_{xx} + \theta_{yy} = 0$, $A\Phi_{xx} + A\Phi_{yy} = 0$.

It can be underlined, that expression Eq.11 corresponds to expressions Eq.6 that required the statement of the problem. By substituting the deviatoric component

$s_x = \sigma_x - \sigma_0 - f(y),\ s_y = \sigma_y - \sigma_0 - f(x)$, the following is obtained

$$\sigma_x = C_\sigma \cdot exp\,\theta \cdot cos\,A\Phi + \sigma_0 + f(y), \qquad (12)$$

$$\sigma_y = -C_\sigma \cdot exp\,\theta \cdot cos\,A\Phi + \sigma_0 + f(x).$$

Above-mentioned Cauchy-Riemann conditions of argument functions completely close the solution of the problem both by boundary conditions Eq. 6 and equilibrium equations Eq. 1. Unknown functions θ and $A\Phi$ are introduced into consideration and are determined by Laplace's equations, according to Eq. 13, what introduces sufficient certainty for their finding. Differential relations

$$\theta_x = -A\Phi_y,\ \ \theta_y = A\Phi_x,\ \ \theta_{xx} + \theta_{yy} = 0,\ \ A\Phi_{xx} + A\Phi_{yy} = 0 \qquad (13)$$

are invariants of argument functions that limit the solution of the problem. When using Eq. 13, a tool for obtaining additional possibilities of an analytical and numerical solution appears. A whole class of argument functions is found, hence new dependencies that satisfy boundary conditions and equations of equilibrium of the system in Eq. 1.

However, the problem is not completed, because average normal stresses included in Eq. 7, Eq. 11, Eq. 12 are not determined, through the condition of joint deformations Eq. 8. In this case, a problem is formulated: what values of argument functions satisfy the continuity condition in Eq. 8. Write down Eq. 8 in terms of a function of a complex variable

$$\nabla^2\left(C_\sigma \cdot \frac{exp\left(\theta + iA\Phi\right) + exp\left(\theta - iA\Phi\right)}{2}\right) = 0.$$

Write down the derivatives with respect to coordinates

$$\frac{\partial^2\left[C_\sigma \cdot \frac{exp\left(\theta + iA\Phi\right) + exp\left(\theta - iA\Phi\right)}{2}\right]}{\partial x^2} =$$

$$= C_\sigma \frac{\left[\left(\theta_{xx} + iA\Phi_{xx}\right) + \left(\theta_x + iA\Phi_x\right)^2\right] exp\left(\theta + iA\Phi\right)}{2} +$$

$$+ \frac{\left[\left(\theta_{xx} - iA\Phi_{xx}\right) + \left(\theta_x - iA\Phi_x\right)^2\right] exp\left(\theta - iA\Phi\right)}{2},$$

$$\frac{\partial^2\left[C_\sigma \cdot \frac{exp\left(\theta + iA\Phi\right) + exp\left(\theta - iA\Phi\right)}{2}\right]}{\partial y^2} =$$

$$= C_\sigma \frac{\left[\left(\theta_{yy} + i\mathrm{A}\Phi_{yy}\right) + \left(\theta_y + i\mathrm{A}\Phi_y\right)^2\right] exp\left(\theta + i\mathrm{A}\Phi\right)}{2} +$$

$$+ \frac{\left[\left(\theta_{yy} - i\mathrm{A}\Phi_{yy}\right) + \left(\theta_y - i\mathrm{A}\Phi_y\right)^2\right] exp\left(\theta - i\mathrm{A}\Phi\right)}{2}.$$

After substituting the derivatives into the equation of continuity of deformations and simplifications, the following is obtained

$$exp\left(\theta + i\mathrm{A}\Phi\right)\cdot$$

$$\cdot\left[\left(\theta_{xx} + \theta_{yy}\right) + \left(\mathrm{A}\Phi_{xx} + \mathrm{A}\Phi_{yy}\right)\cdot i + \left(\theta_x + i\mathrm{A}\Phi_x\right)^2 + \left(\theta_y + i\mathrm{A}\Phi_y\right)^2\right] +$$

$$+ exp\left(\theta - i\mathrm{A}\Phi\right)\cdot \tag{14}$$

$$\cdot\left[\left(\theta_{xx} + \theta_{yy}\right) - \left(\mathrm{A}\Phi_{xx} + \mathrm{A}\Phi_{yy}\right)\cdot i + \left(\theta_x - i\mathrm{A}\Phi_x\right)^2 + \left(\theta_y - i\mathrm{A}\Phi_y\right)^2\right] = 0.$$

Operators near the exponents contain the same second derivatives with respect to coordinates and non-linearity. If, for some reason, the operators are zero, then there is an identity. Show this. Write non-linearities in the operators and regroup them

$$\left(\theta_x + i\mathrm{A}\Phi_x\right)^2 + \left(\theta_y + i\mathrm{A}\Phi_y\right)^2 =$$

$$= \left(\theta_x + \mathrm{A}\Phi_y\right)\cdot\left(\theta_x - \mathrm{A}\Phi_y\right) + 2i\left(\theta_x \cdot \mathrm{A}\Phi_x + \theta_y \cdot \mathrm{A}\Phi_y\right) +$$

$$+ \left(\theta_y + \mathrm{A}\Phi_x\right)\cdot\left(\theta_y - \mathrm{A}\Phi_x\right),$$

$$\left(\theta_x - i\mathrm{A}\Phi_x\right)^2 + \left(\theta_y - i\mathrm{A}\Phi_y\right)^2 =$$

$$= \left(\theta_x + \mathrm{A}\Phi_y\right)\cdot\left(\theta_x - \mathrm{A}\Phi_y\right) - 2i\left(\theta_x \cdot \mathrm{A}\Phi_x + \theta_y \cdot \mathrm{A}\Phi_y\right) +$$

$$+ \left(\theta_y + \mathrm{A}\Phi_x\right)\cdot\left(\theta_y - \mathrm{A}\Phi_x\right).$$

Assuming one of the multiplied parentheses equal to zero, evade from nonlinearity, then $\theta_x = -\mathrm{A}\Phi_y$, $\theta_y = \mathrm{A}\Phi_x$, which was the case when solving differential equations of equilibrium. The expression for both operators is automatically converted to zero

$$\theta_x \cdot \mathrm{A}\Phi_x + \theta_y \cdot \mathrm{A}\Phi_y = -\mathrm{A}\Phi_y \cdot \mathrm{A}\Phi_x + \mathrm{A}\Phi_x \cdot \mathrm{A}\Phi_y = 0.$$

The equation of continuity Eq. 14, is substantially simplified and takes the form

$$exp\left(\theta+i\mathrm{A}\Phi\right)\cdot\left[\left(\theta_{xx}+\theta_{yy}\right)+\left(\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}\right)\cdot i\right]+$$
$$+exp\left(\theta-i\mathrm{A}\Phi\right)\cdot\left[\left(\theta_{xx}+\theta_{yy}\right)-\left(\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}\right)\cdot i\right]=0.$$

From the Cauchy-Riemann relations we define the second derivatives, which show, see above, that $\theta_{xx}+\theta_{yy}=0$, $\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}=0$, i.e. the equation of continuity of deformations is identically satisfied.

Consequently, the solution of the equation of continuity of deformations is

$$\sigma_0=n\cdot C_\sigma\cdot exp\theta\cdot cos\mathrm{A}\Phi \tag{15}$$

where $\theta_x=-\mathrm{A}\Phi_y$, $\theta_y=\mathrm{A}\Phi_x$, $\theta_{xx}+\theta_{yy}=0$, $\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}=0$, n - any number.

Solution in Eq. 15 is subjected to the same limitations as in Eq. 11, Eq. 12, for those parameters.

It should be emphasized that the solution of the equation of continuity of deformations allows a presence of two exponents with opposite signs of argument function θ simultaneously in the expression of average normal stress. Then

$$\sigma_0=n\cdot C_\sigma\cdot exp\left(-\theta\right)\cdot cos\mathrm{A}\Phi=n\cdot C_\sigma\frac{exp\left(-\theta+i\mathrm{A}\Phi\right)+exp\left(-\theta-i\mathrm{A}\Phi\right)}{2}.$$

Substituting into the equation of continuity of deformations, there is

$$\begin{gathered}exp\left(-\theta+i\mathrm{A}\Phi\right)\cdot\\ \cdot\left[-\left(\theta_{xx}+\theta_{yy}\right)+\left(\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}\right)\cdot i+\left(\theta_x-i\mathrm{A}\Phi_x\right)^2+\left(\theta_y-i\mathrm{A}\Phi_y\right)^2\right]+\\ +exp\left(-\theta-i\mathrm{A}\Phi\right)\cdot\\ \cdot\left[-\left(\theta_{xx}+\theta_{yy}\right)-\left(\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}\right)i+\left(\theta_x+i\mathrm{A}\Phi_x\right)^2+\left(\theta_y+i\mathrm{A}\Phi_y\right)^2\right]=0.\end{gathered} \tag{16}$$

Comparing Eq. 14 and Eq. 16, it's clear that the operators in front of exponents, in comparison with the function θ of the opposite sign, from a point of view of the solution practically did not change.

Group

$$\begin{gathered}exp\left(-\theta+i\mathrm{A}\Phi\right)\cdot\\ \cdot\left[-\left(\theta_{xx}+\theta_{yy}\right)+\left(\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}\right)\cdot i+\left(\theta_x+\mathrm{A}\Phi_y\right)\cdot\left(\theta_x-\mathrm{A}\Phi_y\right)-\right.\\ \left.-2i\cdot\left(\theta_x\mathrm{A}\Phi_x+\theta_y\mathrm{A}\Phi_y\right)+\left(\theta_y+\mathrm{A}\Phi_x\right)\cdot\left(\theta_y-\mathrm{A}\Phi_x\right)\right]+\end{gathered}$$

$$+exp\left(-\theta - i\mathrm{A}\Phi\right)\cdot$$
$$\cdot\left[-\left(\theta_{xx}+\theta_{yy}\right)-\left(\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}\right)\cdot i+\left(\theta_{x}+\mathrm{A}\Phi_{y}\right)\cdot\left(\theta_{x}-\mathrm{A}\Phi_{y}\right)+\right.$$
$$\left.+2i\cdot\left(\theta_{x}\mathrm{A}\Phi_{x}+\theta_{x}\mathrm{A}\Phi_{x}\right)+\left(\theta_{y}+\mathrm{A}\Phi_{x}\right)\cdot\left(\theta_{y}-\mathrm{A}\Phi_{x}\right)\right]=0.$$

Using the Cauchy-Riemann conditions, $\theta_x = \mathrm{A}\Phi_y$, $\theta_y = -\mathrm{A}\Phi_x$, the continuity equation is substantially simplified and takes a form

$$exp\left(-\theta + i\mathrm{A}\Phi\right)\cdot\left[-\left(\theta_{xx}+\theta_{yy}\right)+\left(\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}\right)\cdot i\right]+$$
$$+exp\left(-\theta - i\mathrm{A}\Phi\right)\cdot\left[-\left(\theta_{xx}+\theta_{yy}\right)-\left(\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}\right)\cdot i\right]=0.$$

The established differential connection between the argument functions allows determining the second derivatives

$$\theta_{xx} = \mathrm{A}\Phi_{yx}, \qquad \theta_{yy} = -\mathrm{A}\Phi_{xy}, \qquad \mathrm{A}\Phi_{xx} = \theta_{yx}, \qquad \mathrm{A}\Phi_{yy} = -\theta_{yy}.$$

Substituting the derivatives into the equation, it can be noted that it turns into an identity.

From a single point of view, a plane problem of the theory of elasticity is formulated and solved, generalizing relations Eq. 13 are identified that determine the conditions for existence of a given class of solutions through invariants of differential ratios of argument functions.

In general, the following can be written down

$$\sigma_x = \pm C_\sigma \cdot exp\left(\pm\theta\right)\cdot cos\mathrm{A}\Phi + \sigma_0 + f(y) + C,$$
$$\sigma_y = \mp C_\sigma \cdot exp\left(\pm\theta\right)\cdot cos\mathrm{A}\Phi + \sigma_0 + f(x) + C,$$
$$\tau_{xy} = C_\sigma \cdot exp\left(\pm\theta\right)\cdot sin(\mathrm{A}\Phi),$$
$$\sigma_0 = \pm n\cdot C_\sigma \cdot exp\left(\pm\theta\right)\cdot cos\mathrm{A}\Phi, \tag{17}$$

where $\theta_x = \mp\mathrm{A}\Phi_y$, $\theta_y = \pm\mathrm{A}\Phi_x$, $\theta_{xx}+\theta_{yy}=0$, $\mathrm{A}\Phi_{xx}+\mathrm{A}\Phi_{yy}=0.$

The analysis shows that the solution Eq. 17 can be further reinforced and presented in the form

$$\sigma_x = \pm exp\left(\pm\theta\right)\left(C_1 cos\mathrm{A}\Phi - C_2 sin\mathrm{A}\Phi\right) + \sigma_0 + f(y) + C =$$
$$= \pm\left[ch\left(\pm\theta\right)\pm sh\left(\pm\theta\right)\right]\left(C_1 cos\mathrm{A}\Phi - C_2 sin\mathrm{A}\Phi\right) + \sigma_0 + f(y) + C,$$
$$\sigma_y = \mp exp\left(\pm\theta\right)\left(C_1 cos\mathrm{A}\Phi - C_2 sin\mathrm{A}\Phi\right) + \sigma_0 + f(x) + C =$$

$$= \mp\left[ch(\pm\theta) \pm sh(\pm\theta)\right]\left(C_1 cos\mathrm{A}\Phi - C_2 sin\mathrm{A}\Phi\right) + \sigma_0 + f(x) + C,$$

$$\tau_{xy} = exp(\pm\theta)\cdot\left(C_1 sin\mathrm{A}\Phi + C_2 cos\mathrm{A}\Phi\right) =$$

$$= \left[ch(\pm\theta) \pm sh(\pm\theta)\right]\left(C_1 sin\mathrm{A}\Phi + C_2 cos\mathrm{A}\Phi\right),$$

$$\sigma_0 = \pm n \cdot exp(\pm\theta)\cdot\left(C_1 cos\mathrm{A}\Phi - C_2 sin\mathrm{A}\Phi\right) =$$

$$= \pm n \cdot \left[ch(\pm\theta) \pm sh(\pm\theta)\right]\left(C_1 cos\mathrm{A}\Phi - C_2 sin\mathrm{A}\Phi\right), \qquad (18)$$

where $\theta_x = \mp\mathrm{A}\Phi_y,\ \theta_y = \pm\mathrm{A}\Phi_x,\ \theta_{xx} + \theta_{yy} = 0,\ \mathrm{A}\Phi_{xx} + \mathrm{A}\Phi_{yy} = 0.$

As a particular option, expression Eq. 18 can be considered as a function of stresses and it can be used for comparative analysis. Indeed, the biharmonic equation for a plane problem can be represented

$$\nabla^4\phi = \nabla^2\left(\nabla^2\phi\right) = 0\ .$$

As was shown above $\nabla^2\left(\sigma_0\right) = 0,$ therefore, $\nabla^2\left[\nabla^2\left(\sigma_0\right)\right] = \nabla^2\left[0\right] = 0.$

In [19], solutions of a plane problem with a help of trigonometric series are shown. Stress function ϕ has a form

$$\begin{aligned}\phi &= sin(\alpha x)\cdot \\ &\cdot\left[C_3\cdot ch(\alpha y) + C_4\cdot sh(\alpha y) + C_5\cdot y\cdot ch(\alpha y) + C_6\cdot y\cdot sh(\alpha y)\right].\end{aligned} \qquad (19)$$

Reduce the expressions Eq. 18 and Eq. 19 to a comparable form, i.e. $C_5 = C_6 = 0$, $\mathrm{A}\Phi = \alpha x$, $\theta = \alpha y$, $n = 1$, $C_1 = 0$, $C_2 = -1$.

In Eq. 18, the plus sign is chosen before the expression. In this case, the expressions Eq. 18 and Eq. 19 coincide; consequently, for both equations the Cauchy-Riemann conditions and Laplace's equations should be valid, which were obtained by the present solution

$$\theta_x = -\mathrm{A}\Phi_y,\qquad \theta_y = \mathrm{A}\Phi_x,\qquad \theta_{xx} + \theta_{yy} = 0,\qquad \mathrm{A}\Phi_{xx} + \mathrm{A}\Phi_{yy} = 0.$$

Verify the fulfillment of these generalizations, indeed $\theta_x = 0,$ $\mathrm{A}\Phi_y = 0,\ \theta_y = \alpha,\ \mathrm{A}\Phi_x = \alpha$. Then the Cauchy-Riemann conditions for the known solution also take place $0 = -0,,\ \alpha = \alpha,$ i.e. the functions obtained in [19] are a particular solution with respect to Eq. 18.

This follows from the fact that the functions αx and αy are the simplest solution of Laplace's equation, which allows an entire class of harmonic functions in various combinations. It is assumed here that the functions are not necessarily linear and may depend simultaneously on several coordinates. For example, a more complex function $A\Phi$ is a second-order function

$$A\Phi = \alpha xy \,. \tag{20}$$

It satisfies the Laplace's equation and, through the Cauchy-Riemann conditions, determines the second argument function θ, in a form

$$\theta = \frac{1}{2} \cdot \alpha \cdot \left(x^2 - y^2\right), \tag{21}$$

which is also a solution of Laplace's equation. There can be quite a lot of harmonic functions satisfying the Cauchy-Riemann conditions.

They affect the final result in different ways, and can be characterized by different boundary conditions of practical problems.

Unlike the result obtained by the method of separation of a variable, argument functions that also appear in solution Eq. 19 are nonlinear, and simultaneously depend on two coordinates, what extends the solution possibilities.

It can be shown that the expressions Eq. 17, Eq. 18, taking into account Eq. 20 and Eq. 21, give a qualitatively and quantitatively close result, presented in [19], Fig. 1.

The action of punch which attached to border of semiplane is shown without including friction.

Central point of coordinates is located on contact surface.

Strained near-contact zone, within which a connection between the force P and constant C_σ is formed in expressions Eq.17, Eq. 18. entered in consideration.

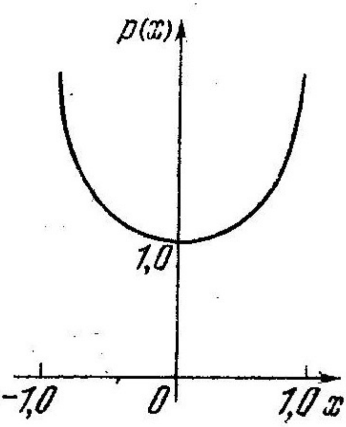

Fig.1. Diagram of distribution of normal stresses along the horizontal axis

Expression Eq. 17 can be considered in the working version along contact surface, as follows

$$\sigma_y = p(x) = 2C_\sigma \cdot exp\left(\frac{1}{2} \cdot \frac{\pi}{2by_0} \cdot x^2\right),$$

$$\tau_{xy} = 0.$$

Value σ_0 is chosen according to boundary conditions Eq. 8.

Discussion of research results of strain-stress state using an argument functions method and a method of functions of a complex variable

Considered cases of decisions (Fig.1.) is the test, which confirmed legitimacy of the got new result. However, it can be examined in wider plan: including friction on contact surface, using different character of distribution of forces of external influence, charts of loading of elastic half-space, boundary conditions and charts of the stress states, difficult geometry of tools.

As a result, it can be noted that introducing into consideration argument functions, that make it possible to close the problem and to reveal generalizing characteristics for them in a form of the Cauchy-

Riemann and Laplace's equations, introduces sufficient certainty in obtaining the final result.

Conclusions

1. A method of the argument of functions, before used in the theory of plasticity and in dynamic tasks got development.

2. Generalizing differential equations are got as conditions of Cauchy-Riemann and Laplace, allowing to define harmonic functions for existent and perspective tasks.

3. With the use of new approaches the plane task of theory of elasticity is decided in an analytical form.

4. The got result in the special cases is comparable with the results of theoretical and experimental researches of other authors.

5. There is a sufficient prospect with the use of offered approach to decide the row of the intricate applied problems of plane task of theory of elasticity.

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MINIMISING THE NEGATIVE IMPACT OF RELOADING POINTS FOR SHOVELS ON THE DEVELOPMENT OF MINING OPERATIONS IN THE DEEP OPEN PIT MINES

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Abstract

The article analyzes the main problems of combined transport and designs of reloading points in the deep open pit mines. The use of face shovels at the reloading points complicates the truck traffic pattern, increases the lifting height and the range of rock mass transportation by dump trucks. In many iron ore open pit mines there is a tendency to transfer of reloading points to the upper pit levels.

The purpose of the article is to develop and substantiate a design of the reloading point for the shovel with better operating performance than the reloading points based on the face shovels.

The main idea of the work is to use the design features of extraction and loading equipment in order to substantiate the rational design of the reloading point. The use of a backhoe hydraulic shovel as an extraction and loading equipment at the reloading point of new design is foreseen. The conceptual difference from the existing reloading points is that the rock is not piled, but loaded to the receiving trench.

The proposed reloading point allows avoiding a temporary non-working pillar. The use of backhoe hydraulic shovels at the reloading points will increase the use of road and rail transport in reconstruction of traffic patterns in the deep open pit mines.

Introduction

An outlook for improving the efficiency of deep open pits is the further improvement of combined transport. The process of reloading the rock mass is an important part of mining technology. Its significance grows with further pit deepening [1,2]. The process features of reloading points for shovels in the road and rail transport adversely affect the development of mining operations. Firstly, pillars [3] are formed in the working area of the open pit under the reloading points. They adversely affect the dynamics of mining operations and may cause a decline in iron ore production. Secondly, the mining conditions are created in the open pit mine, under which the shortest rock haul by trucks is possible when crossing the rail tracks and roads [1]. But conversely, when crossing the roads and rail tracks, the loss of truck tonnage takes place for another reason - due to waiting time at the railway crossings. Thus, the routine practice in the open pit mines is to detour around the dead-end tracks at the reloading points. In order to prevent crossing the transport communications the trucks increase the rock haul distance. In view of this, in many iron ore open pit mines there is a tendency to transfer of reloading points to the upper pit levels.

Analysis of trends in the development of reloading points for the road and rail transport

Many domestic and foreign open pit mines use combined transport systems. Moreover, each type of transport should operate under the most convenient and favorable conditions; this is a prerequisite to achieve the maximum technical and economic efficiency of the transport process. In [4-7], the regularities of changes in the technical and economic parameters of rock haulage under various mining conditions were investigated and the features of mining operations when using the combined transport were studied.

A separate approach in research is to optimize the parameters of reloading points and study the influence of their designs on the features of interaction of the adjacent links of combined transport [4]. A negative feature of combined transport is the need to build the reloading points, combining two adjacent links of transport. The reloading points complicate the production process and require additional investment.

The reloading points for the road and rail transport can be made in the form of reloading docks, stockpiles [4] or bins of different operating principles and designs [8]. The reloading points in the form of docks, bins and other structures have serious drawbacks: a high level of damage risk and rapid wear of dumpcars and bins resulted from high dynamic loads when unloading the rock; the rigid relationship between road and rail transport operation.

The reloading points using front shovels prevail in road and rail transport in the open pit mines.

Despite the optimization of operating and capital costs for rock transportation, the use of combined transport in the open pit mines is complicated by pillars located under transport communications and reloading points, as well as by increase in the distance of truck haul in order to avoid crossing the rail tracks (detour of dead-end tracks).

In the deep iron ore open pit mines, the combined transport will maintain its efficiency only if it settles the issue of transferring the reloading points following a progress in mining depth. Without solving this problem, the haul distance will continue to grow. Currently, the method of reloading the rock by shovels from trucks to dumpcars has become the most common when using the combined road and rail transport. The equipment of the reloading point consists of one or several shovels and dozers. The length of the stockpile, as a rule, is divided into two parts: the dump truck unloading area (filling) and the shovel operation area (loading). The length of these parts depends on the number of trucks to be simultaneously unloaded, and should be at least 40 m.

The widespread use of the method to reload the rock by shovels is explained by the reasons as follows [5]: the possibility of handling the large volumes of rock mass, the independence of transport facilities to deliver and ship the rock, the simplicity of the reloading point design, and the high speed construction.

However, there are certain disadvantages of reloading by shovels, that reduce the efficiency of combined road and rail transport and limit, thus, the depth of its entering the open pit: a large size in plan, low throughput, the need for forming the temporary non-working pillars.

The feasible depth of using the rail transport in the open pit mine is 150-250 m. In this context, the number of reloading points within the open pit field can vary from 1-2 to 6-8.

The reloading points are often placed on the working pit wall. The lifetime of such a reloading point in one place ranges from one to three years [3]. The semi-stationary nature of in-pit reloading points makes it difficult to fulfill the requirement of proportional mining operations at the adjacent levels with a certain speed in a given direction.

Taking into account the unfavorable conditions above, there is a need to reduce the size of reloading points located in the deep pit area, that, in turn, will increase their number and reduce the transfer step.

Setting objectives of the article (task assignment)

The purpose of the article is to develop and substantiate a design of the reloading point for shovels with better operating performance than the reloading points based on the face shovels.

The main idea of the work is to use the design features of extraction and loading equipment in order to substantiate the rational design of the reloading point. The use of a backhoe hydraulic shovel as an extraction and loading equipment at the reloading point of new design is foreseen. The following tasks were performed during the work: analysis of trends in the development of combined transport and designs of reloading points; development of diagrams of reloading points using backhoe hydraulic shovels; theoretical grounds of parameters and the broad technical and economic assessment of the reloading points.

Technological features of backhoe hydraulic shovels

The backhoe hydraulic shovels have become widespread in the foreign mining companies, and are being more and more used in domestic open pit mines. The Hitachi hydraulic shovels are used in the iron ore open pit mines of the Kryvyi Rih basin. Mainly, they are used in complex mining invironment, when eliminating the effects of landslides and traffic jams resulted from the rock mass collapse, when developing the watered areas of deposits and constructing the water sumps [9,10].

In [10-13], it is proposed to use the backhoe hydraulic shovels as the main equipment of the reloading point for shovels. The design features of hydraulic shovels enable us to construct such a reloading point, which has a less negative impact on the dynamics of mining operations. The conceptual difference from the existing reloading points is that the rock is not piled, but loaded to the receiving trenches below the level of the shovel location. The dimensions of the receiving trench are determined by the technical characteristics of the reloading shovel, its position with respect to the trench and the stockpile capacity required.

Technology of construction and operation of the reloading point

The rock delivered by trucks to the reloading point is unloaded downhill along the length of the receiving trench in the filling area

[11,12] (Fig. 1*a*). The dumping front moves in the direction from the unloading site to the rail tracks. The rock is loaded simultaneously with filling the part of stockpile. For this purpose, the receiving trench is pre-divided into two sections: filling and loading. The rock is loaded from the stockpile to the transport facilities by shovel with downward digging from the end face across the entire width of the receiving trench. Thus, in the process of loading the dump cars, the shovel is moved on the "cushion" formed of the rock unloaded to the receiving trench in the direction from the end to the centre [11].

The width of the receiving trench is determined as follows [12]. The maximum dumping radius of the shovel $R_Р$ that ensures uniform filling the dump car, determines the axis of its travel along the receiving trench, from which the maximum digging radius at the level of standing $R_Ч$ is plotted. The depth of the reloading trench is determined by operating parameters of the backhoe hydraulic shovel and it does not exceed 7-8 m for most shovels.

The rail track is located along the longitudinal slope above the bench between its toe and the upper edge of the receiving trench. The lane of rail transport and the upper edge of the receiving trench are separated by a safety pillar with the width $B_Ц$.

The maximum width of the receiving trench is determined by the performance specification of the reloading shovel (Fig. 2*a*) [12], м

$$B_{тр} = R_ч + R_р - B_ц - \frac{1}{2} b_т , \quad (1)$$

The minimum width of the site at the pit level to locate the reloading point is calculated by, м

$$B_{пп} = b_т + B_ц + B_{тр} + B_м + b_в + b_{пр} , \quad (2)$$

where $b_{пр}$ is a width of the sliding triangle from the free surface, m; $b_в$ is a width of the safety embankment, m.

a b c

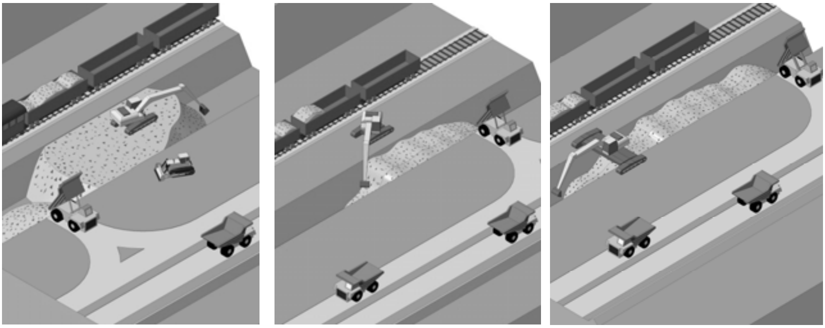
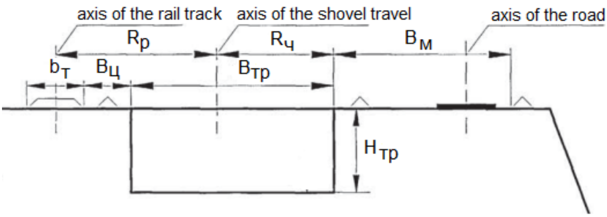
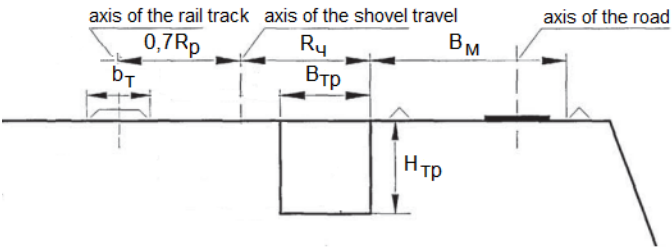


Fig. 1. Concepts of in-pit reloading points for the road and rail transport using the backhoe hydraulic shovels as the reloading equipment

a



b



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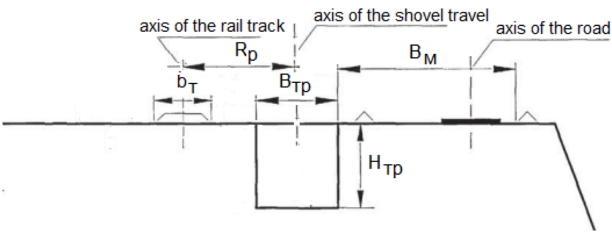

Fig. 2. Calculation models of reloading points

Analysis of the method for the formation and operation of the reloading point using a backhoe hydraulic shovel, which is located on the rock dumped to the receiving trench [12], allowed the following features of this method to be identified. Comparing this method with a reloading site equipped with a front shovel, it should be noted that the general need for ensuring continuous operation of the reloading point is to divide it longwise into two sections (dump truck unloading area and dump car loading area). This leads to an extension of the length of the reloading point and increase in the volume of the rock stored. This feature is a disadvantage of this method; we cannot load the rock until the cross section of the trench is filled with rock.

When filling the trench with the rock, the unloading area for trucks quickly moves (since the trench is not deep). This feature causes a large volume of dozer planning operations, about twice as much as at the reloading point with a front shovel. The reloading point, where the shovel does not travel over the rock dumped within the trench outline, has no this drawback.

The reloading shovel moves over the site located between the upper edge of the receiving trench and the axis of the rail track (Fig. 1*b*). This position of the shovel makes it possible to remove the rock from the receiving trench with no need for full filling.

The surface of the reloading site divided into the dumping area for trucks and the loading area for dumpcars is located not perpendicular to the pit wall (axis of the rail track), but parallel, coinciding with the longitudinal axis of the receiving trench.

This design of the reloading site makes it very compact, mobile and capable to load the rock with minimum filling.

The mining operations using the developed technology are made as follows (Fig. 2) [13]. A backhoe hydraulic shovel (1) excavates a receiving trench (2). This trench (2) is conditionally divided by width into two sections: the unloading wall (3) and the loading wall (4). The receiving trench (2) is filled with rock by trucks (5) on the unloading wall (3). In the general case, in order to prevent from crossing the haul roads, the receiving trench wall (2) located closer to the lower pit benches serves as the unloading wall (3); the receiving trench wall located closer to the higher pit benches serves as the loading wall (4).

The rail track (7) is located along the loading wall (4) of the receiving trench (2).

The rock from the receiving trench (2) is reloaded by a hydraulic shovel (1), located on the loading wall (4) of the receiving trench (2) onto dump cars (8) located at the level of the hydraulic shovel.

The reloading point (Fig. 3) operates until the higher bench (8) has been mined out. Upon advancing the mining operations on the higher bench, a backhoe hydraulic shovel (1) excavates a receiving trench (2) on the free site.

After commissioning the relocated reloading point, the lower bench (9) is mined out. In order to increase the throughput of the reloading point, two or more backhoe hydraulic shovels are placed on the loading wall of the trench at a safe distance from each other.

The use of the proposed reloading point design provides an increase in the capacity of mining equipment and reduces the negative impact of open pit transport on the dynamics of mining operations.

The width of the receiving trench is determined as follows (Fig. 2*b*). To ensure uniform filling the dump car, we locate the axis of the shovel travel from the axis of the rail track at a distance of 0.65-0.7 from the maximum unloading radius of the shovel $R_{\text{Р}}$, from which the maximum digging radius at the level of standing $R_{\text{Ч}}$ is plotted.

The maximum digging radius at the level of standing $R_{\text{Ч}}$ determines an opposite wall of the trench, where the dump trucks are unloaded.

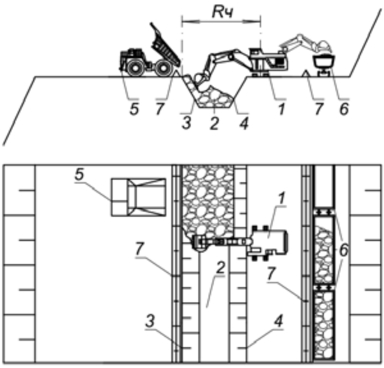

Fig. 2. Method of formation and operation of the reloading point [13]

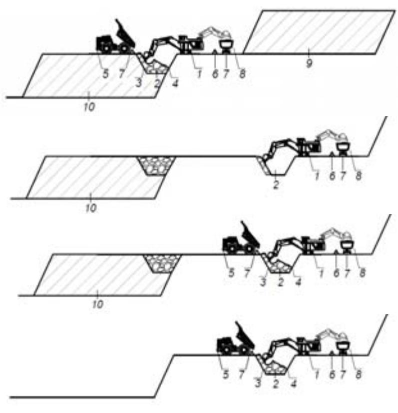

Fig. 3. Diagram of mining operations when shifting the reloading point

The width of the trench is determined by considering the shovel parameters and its safe position on the loading wall of the trench, *m*

$$B_{mp} = R_{ч} - 0{,}5B_{x} - c\,, \tag{3}$$

where B_X is a width of the crawler track undercarriage, m; *c* is a distance from the outer edge of the crawler track to the edge of the trench, *m*.

The minimum width of the site at the open pit level to locate the reloading point is calculated by (Fig. 2*b*), *m*

$$B_{nn} = \frac{1}{2}b_{m} + 0{,}7R_{p} + R_{ч} + B_{м} + b_{в} + b_{np}\,, \tag{4}$$

Tables 1 and 2 give a comparative analysis of the advantages and disadvantages of reloading points with different types of shovels.

Alternatively, the reloading point with the receiving trench located between the crawler tracks of the shovel can be considered (Fig. 1*c*).

To ensure high receiving capacity of the trench, it is necessary to manufacture the shovel with an increased width of the crawler track undercarriage (10-12 m).

The reloading point of this design has a smaller width and retains the advantages of previously considered concepts, *m*.

$$B_{mp} = B_{x} - 2\left(b_{г} + c\right) \tag{5}$$

The minimum width of the site at the level of the reloading point is calculated by the formula (Fig. 2c), *m*

$$B_{nn} = \frac{1}{2}b_{m} + R_{p} + 0{,}5B_{mp} + B_{м} + b_{в} + b_{np}\,, \tag{6}$$

The key parameters of the reloading point affecting the performance of the open pit mine are the width, length, capacity and productivity of the reloading shovel.

Table 1

Comparative analysis of the designs of reloading points for shovels (advantages)

Concept with the use of a front shovel	Concept with position of the hydraulic shovel on the rock in the receiving trench	Concept with travel of the hydraulic shovel over the pillar along the receiving trench
1. Simple design. 2. The standard extraction and loading equipment is used.	1. No conflict with the development of mining operations. 2. Simple design. 3. Reducing the truck haul distance: no more extra lifting the rock, no need for detour of dead-end tracks. 4. An ability of one reloading point to serve several pit faces when using a powerful hydraulic shovel.	1. No conflict with the development of mining operations. A non-stationary, sliding reloading point. 2. Simple design. 3. The small width, length and area of the receiving trench. 4. Reducing the truck haul distance: no more extra lifting the rock, no need for detour of dead-end tracks. 5. An ability of one reloading point to serve several pit faces when using a powerful hydraulic shovel. 6. Less need for dozers. 7. The possibility to load the dumpcars with no need for complete filling the trench with rock.

Table 2

Comparative analysis of the designs of reloading points for shovels (disadvantages)

Concept with the use of a front shovel	Concept with position of the hydraulic shovel on the rock in the receiving trench	Concept with travel of the hydraulic shovel over the pillar along the receiving trench
1. The reloading point is located on two adjacent sites (the upper one is an area for moving and unloading the trucks, the lower one is an area to locate a rock stockpile, a shovel and a rail track). 2. The reloading point occupies a large area. The stockpile is divided by length into two sections, which are alternately used for receiving and loading the rock. 3. Large volume of dozer operations. 4. Increase in the haul distance by road due to the need to detour around the dead-end tracks and lift the rock to the height of the reloading point. 5. Creation of obstacles in the development of mining operations. 6. The use of identical shovels at the faces and at the reloading points limits their throughput and increases their number.	1. The reloading point occupies a large area. The stockpile is divided by length into two sections, which are alternately used for receiving and loading the rock. 2. The shallow depth and the wide width of the receiving trench leads to its rapid filling and large volume of dozer operations. 3. Before loading the dump-cars, it is necessary to completely fill the relevant part of the trench with rock. 4. To construct the receiving trench the smoothwall blasting is used.	1. To construct the receiving trench the smoothwall blasting is used. 2. The small capacity of the receiving trench requires coordinated operation of road and rail transport. The volume of the receiving trench provides a rock stock for 2-6 shifts.

Study of dependence of key parameters of the reloading point on the parameters of the shovel

Let us investigate the dependence of the receiving trench capacity on the shovel position as to the receiving trench and on the operating parameters of the shovel. Fig. 4 shows the results of the calculation of the receiving trench capacity of 7-8 m deep and 100 m long using the Hitachi EX 2500-6 shovel ($R_ч$ = 16.5 m, R_P = 12 m), the range of the shovel operating parameters is 0.75-1 of the maximum values, the rock fragmentation index is 1.35.

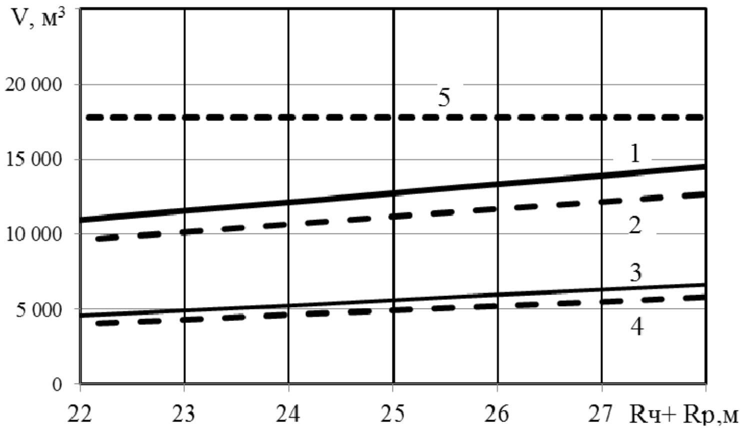

Fig. 4. Dependencies of the stockpile capacity of the “receiving trench” type on the performance of the reloading shovel, the stockpile length is 100 m (1,2 - concept with the shovel moving over the rock within the receiving trench outline; 3,4 - concept with the shovel moving over the pillar along the receiving trench, 5 - the capacity of the rock stockpile using a front shovel; 1,3 - trench depth is 8 m; 2,4 - trench depth is 7 m).

The capacity of the near-wall reloading stockpile of 100m long and 12m high is 17.8 thousand m^3 related to the pillar. Let us compare the capacity of the reloading stockpile of the receiving trench type with the near-wall stockpile, taking the latter as 1. An increase in the degree of utilization of the operating parameters of the backhoe hydraulic shovel from 0.75 to 1 increases the comparative capacity of the reloading stockpile from 0.52 to 0.60 for the trench of 7 m deep and from 0.73 to 0.83 for the trench of 8 m deep with position of the shovel on the rock within the trench outline, and from 0.22 to 0.34 for the trench of 7 m deep, and from 0.28 to 0.38 for the trench of 8 m deep with the position of the shovel outside the trench outline.

The concept with position of the shovel on the rock within the trench outline is characterized by a less specific capacity (0.83), but close to the capacity of the near-wall stockpile. However, this concept still has drawbacks inherent in near-wall stockpiles: the need to divide the reloading point into two sections of equal length - the truck unloading section and the dumpcar loading section, as well as a large amount of dozer operations. In fact, for two-section reloading points, the calculated capacity refers to the entire length of the reloading point, being constant throughout all stages of the normal operation of the stockpile (simultaneous dumping and loading the rock from the reloading point). If the designs of the compared reloading points have only one stockpile, this will mean that the reloading point can only be in one of two states at a time (filling or loading the rock from the stockpile), that makes simultaneous and independent operation of adjacent links of the combined transport impossible.

The concept with position of the shovel outside the receiving trench outline is characterized by a far lesser specific capacity of the stockpile (0.34-0.38) with full use of digging radius. The areas for truck unloading and dumpcar loading are located on opposite walls of the trench and do not interfere with the independent and simultaneous operation of adjacent links of the combined transport. The special order of truck dumping must be observed only in the location of the shovel. At the same time, maneuvering and unloading the trucks in the area of the shovel operation should be carried out according to the shovel operator's instructions (similar to the regulations for loading the trucks at the faces). The specific capacity of the reloading point of 5 000 m^3/100 r.m. ensures, during two shifts, the independent operation of adjacent links of combined transport with the throughput of the reloading point of 1.5 million m^3/year. With a limited working area of the open pit and with high synchronization of road and rail transport operation, the length of the receiving trench may be reduced to 50 m (3 $R_{чmax}$). When the capacity of the railway train is 400 m^3 (a train of 10 dumpcars with a payload of 105 tons), the loading operation is provided by a trench section of 8-10 meters long.

Integrated technical and economic assessment

Tabl. 3 presents the comparison of the key parameters of reloading points for the shovels. The EKG-10 shovel with a bucket capacity of $10m^3$ and Hitachi EX 2500-6 with a bucket capacity of $15m^3$ are considered as the reloading equipment. The following parameters are taken to be constant in the calculations: length of the reloading points L=200 m, depth of the receiving trench h=8 m.

The capacities of the reloading points ensures the independent operation of road and rail transport. But the reloading point with the backhoe hydraulic shovel has a smaller width and does not interfere with mining operations on the upper benches. When locating the in-pit reloading point at a depth of 150 m, an extra volume of the rock amounts to 1,9-3,6 million m^3, depending on the design of the reloading point.

An assessment of the economic feasibility of using one or another conception of rock mass reloading is determined by comparing capital and operating costs, taking into account mining and capital costs for the construction of the site, capital costs for extraction and loading equipment for the reloading point and operating costs for rock reloading operations.

When evaluating a long period of time, the rock haulage traffic directed to the reloading point tends to the center of gravity of the reloading point. In the case of the "receiving trench" reloading point, the traffic flows will tend to the middle of the trench in length, in the case of the near-wall reloading point - to the middle of the near-wall stockpile.

Let us find a formula that allows us to determine the difference in the haul distance and in the transport operations between the reloading points using front shovels and backhoe hydraulic shovels.

Table 3

Comparison of in-pit reloading points for shovels

Type of the stockpile, brand of the shovel	Near-wall stockpile, EKG-10 shovel	Receiving trench (an EX 2500-6 shovel is located on the dump)	Receiving trench (an EX 2500-6 shovel is located on the trench wall)
Width of the reloading point, m	80-90	65-70	65-70
Length of the stockpile / dead-end track, m	200/200	200/200	100/200
Height (depth) / width of the stockpile, m	12/20	7-8/25	7-8/10.5
Capacity of the stockpile, thou m^3	18-20	14-16	6-7
The volume of additional overburden (when the depth of the reloading point location is 150 m), thou m^3	3600	2850	1875
Costs for construction of a site for the reloading point, UAH mln.	90	71.25	46.9

Let us consider the working pit wall, on which a reloading point equipped with a front shovel is located.
In the working area, there are faces, from where the rock is delivered by truck to the reloading point (Fig. 5).

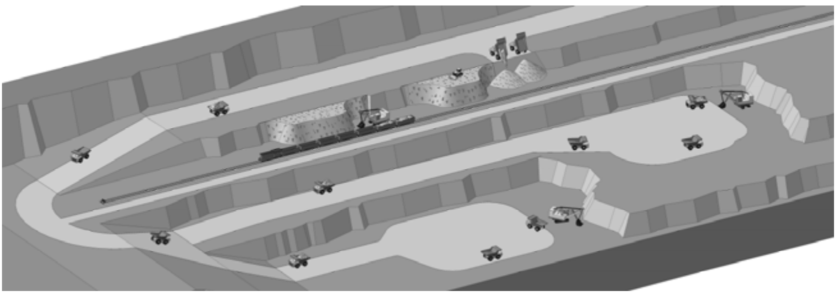

Fig. 5. Concept of mining operations using the road and rail transport

The maximum reduction in the rock haul distance (in plan) during the transition from the near-wall stockpile to the "receiving trench" stockpile can be determined by the formula as follows, m

$$\Delta S = 0{,}5L_{nn} + L_m + L_m + 0{,}5L_{nn} = 2L_m + L_{nn} ,$$

where $L_{пп}$ is a length of the reloading point, m;

L_m is a length of the dead-end track, m.

The reduction in transport operations is determined by, tkm

$$\Delta P = \Delta S \times Q_{nn} , ,$$

where $Q_{пп}$ is the throughput of the reloading point, t/year.

For input data (Table 3), the reduction in haul distance is

$$\Delta S = 2L_m + L_{nn} = 2 \times 200 + 150 = 550 \text{ m.}$$

The reduction in haulage with throughput of the reloading point of 1.5 million m^3 (4.2 million tons) is determined by:

$$\Delta P = \Delta S \times Q_{nn} = 0{,}55 \times 4{,}2 = 2{,}31 \text{ million tkm}$$

Taking into account the average cost of 1tkm in the open pit mines of the Kryvyi Rih basin, equal to USD 0.2/tkm (UAH 5.3/tkm), the savings in transport operations will amount to 2.31 × 5.3 = UAH 12.2 mln.

When constructing a new reloading point with its location at a depth of 150m, the volume of mining and capital operations at the "receiving trench" reloading point with position of the shovel within the trench outline is by 750 thousand m^3 less than at the near-wall stockpile, and in the case of the "receiving trench" stockpile with position of the shovel outside the trench outline is by 1 725 thousand m^3 less than at the near-wall stockpile. Savings in operating costs for stripping operations (excluding the cost for pit transport) will amount to UAH 18.75 million and UAH 43.12 million, respectively. The performed calculations allow us to recommend an implementation of the developed flowsheet for in-pit reloading the rock mass using a hydraulic shovel [13].

Conclusions

The article proposes the solution of currently important scientific and technical task to improve the reloading points for shovels using the combined road and rail transport. The proposed reloading point allows avoiding a temporary idle pit wall.

The results of study may be used in the designs of in-pit reloading points in iron ore open pit mines of the Kryvyi Rih basin. The use of backhoe hydraulic shovels at the reloading points will increase the use of road and rail transport in reconstruction of transport patterns in the iron ore open pit mines of the Kryvyi Rih basin. In future, the negtive impact of reloading points on dump trucks using front shovels will be studied more insight.

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ESTIMATION OF EFFECTIVENESS OF DEVELOPMENT OF HEAT POTENTIAL OF FLOODED MINE FIELD

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Abstract. The global scientific and practical experience shows examples of cost-effective usage of low-potential heat of water from flooded mines for heating buildings of various purposes. In this respect, this chapter addresses the current issue of quantitative estimation of thermal and capacitive resources of "Novohrodivska 2" mine, as well as establishes the possibilities and effectiveness of its usage for heating buildings. It was established that "Novohrodivska 2" capacitive resource is represented by a network of flooded mine workings and an developed massif stratum on an area of 18 km^2 in the range of elevation marks from -370,3 to +120 m. Created geofiltration model of the mine field, based on the finite-difference solution of non-stationary planned filtration equations in "Modflow" software, allowed establishing the dynamics of groundwater pressure reduction in zones of development of coal deposits and restore their level after mine drainage shutdown. The predicted position of groundwater level within the mine field at the present moment of time and the planned launch date of the geothermal module was established. The results obtained made it possible to estimate the horizon-by-horizon variation of temperature of mine water and their natural thermal potential of 1300 TJ, within the boundaries of the flooded massif. The performed thermodynamic calculations for the non-operational massif showed that the total heat loss in a process of pumping out, injection and

storage of mine water does not exceed 15 %. The usage of mine water with a temperature of 26-28 °C as a low-potential energy source in heat pumps, in comparison with other alternatives, gives the greatest heat conversion coefficients (4.5-7.5), which allows saving a significant amount of heat pump power.

Introduction

One of the cardinal directions of restructuring the coal industry of Ukraine and restoring the natural regime in coal mining regions is closure of non-operational and unprofitable mines. Due to this, many small mining towns began to have an acute shortage of thermal energy in conditions of constantly growing prices for gasoline and diesel fuel. An example of the present situation is “Novohrodivska 2” mine in Krasnoarmiysky coal-mining region that underwent liquidation by the order of the Ministry of Coal Industry of Ukraine № 237 of May 24, 2001, with maintaining the level of flooding by drainage regime. The town of Novohrodivka with a population of 15 thousand people is located in immediate vicinity of it (10 km), some industrial and civilian facilities of which remained without heating and hot water supply during the cold season. At the same time, the world scientific and practical experience (Germany, France, England) shows examples of profitable usage of low-potential heat from flooded mines to heat single- and two-storey buildings of various purposes, in comparison with other types of energy carriers. In this regard, the issue of quantitative estimation of thermal and capacitive resources of “Novohrodivska 2” mine, as well as determination of a possibility and effectiveness of its usage for heating buildings, becomes urgent.

Mining-geological and hydrogeological conditions of “Novohrodivska 2” mine in Donbass during liquidation.

The examined mine, located in a southern part of Krasnoarmiysky coal-mining region of Donbass, started operation in 1951 and has since been developing seams k_8 and l_1. The upper limit of coal seam development has a mark of +120 m, above which rocks are mostly filled with water [156] to the mark of the local drainage base of groundwater in the valley of r. Solona (+155 m). Closed Korotchenko mine is from the south of “Novohrodivska 2” mine, and operating mines “Novohrodivska 1” and “Rossiya” are from the north and east.

The mine field is geologically and structurally located within the southwestern wing of the Kalmius-Toretska depression and is confined to the footwall of a large regional tectonic disturbance - Selidovsky thrust fault. Mid-Carboniferous sediments (C_2^6 and C_2^5), overlapped by Paleogene-Neogene sands and Quaternary loamy soils are present in a structure of the area. Series C_2^5 contains a large amount of sandstones and a small amount of coal [1] in the lower part. "Novohrodivska 2" mine developed the coal seam k_8 out of the seams of series suitable for industrial development to the mark -370,3 m, with average water inflows into mine workings of 100-120 m^3/hr and frequent water influxes from overlying sandstones and limestones.

Balance reserves of coal in k_8 seam in the territory of "Novohrodivska 2" mine in 1995 were estimated to be 988 thousand tons, and industrial - 841 thousand tons. Thus, as a result of losses caused by mining conditions of deposit development, more than 140 thousand tons of coal were left in the ground. In addition, according to the data of PJSC "Donbassgeology", about 120 thousand tons of off-balance reserves of coal are contained within the mine field in the series C_2^5, concentrated in substandard and thin seams k_7^5 and $k_8^н$.

Balance reserves of coal l_1 are estimated at 17355 thousand tons, and industrial reserves - 12644 thousand tons, which corresponds to losses of 4711 thousand tons. Off-balance reserves of series C_2^6, mainly concentrated in seams l_4 and l_5 are estimated at 3215 thousand tons. Accounting the losses and off-balance reserves allows concluding that more than 8 million tons of coal are concentrated at the present time within the boundaries of "Novohrodivska 2" mine during liquidation, the properties and composition of which are given in Table 1.

Table 1

Properties and grade composition of coal in the territory of "Novohrodivska 2" mine

Seam index	W^a, %	$V^г$, %	$Q_б^г$, MJ/kg	Coal rank
k_8	1,3-3,2	34-40	32,81-35,01	G
l_1	5,0-10,2	40-44	31,12-32,51	D
k_7^5	1,5-2,5	38-46	33,10-33,53	G
l_4	1,1-2,4	33-39	33,70-33,86	G

Data analysis of Table 1 shows that the moisture content of coal of mine area by seam samples varies in a wide range - from 1.1 to 10,2%, averaging 3-5%. The devolatilazation is also quite diverse, with an average value of 39 %, it varies from 33 to 46% [3]. The specific heat of combustion of coal varies slightly and on average is 32 MJ/kg. The sulfur content ranges from 2,5 to 3,5 % (Group 3 of sulfur content). It should be noted that according to their physico-chemical characteristics, all the coals of series C_2^6 and C_2^5 are suitable for development by the method of underground burning.

The hydrogeological conditions of "Novohrodivska 2" mine field are closely connected to its geological structure [4]. Mine water of "Novohrodivska 2", as well as water of adjacent mines, was characterized by sulphatic magnesium-calcium-sodium composition and mineralization of 3,1-3,4 g/dm^3 during the operation period. In this case, the flooding of a significant volume of workings of k_8 and l_1 seams (around 4 million m^3) practically did not affect their chemical composition. At the present time, the mine water has mineralization of 3,3-3,7 g/dm^3 and contain the following basic microcomponents (mg/dm^3): lithium - 0,039-0,05; bromine - 0,01-0,022; lead - 0,017-0,05; manganese - 0,55-1,82. It should be noted that the content of almost all components does not exceed the MPC. After discharge to the surface and settling in the Maslovsky pond-clarifier, located in the upper reaches of the Solony stream, the mine water practically does not change its composition. However, at a distance of 100 m downstream, after the municipal wastewater from the Novohrodivka treatment plants enter the stream, water salinity and hardness in it decrease to 2,2-2,7 g/dm^3 and 15,0-21,7 mmol/dm^3 respectively.

Development of a numerical model and solution of geofiltration problems in a coal massif disturbed by mining. Creation a conceptual model and schematization of hydrodynamic processes in a mine field. The model is based on the data of geological and hydrogeological structure of "Novohrodivska 2" mine field. Model created in the licensed software «MODLOW v. 4.5» (Schlumberger Water Services, Canada), displays two industrial layers, a separating layer between them, as well as the roof of l_1 seam and the bottom of k_8 seam. As a result, it contains five layers with angles of inclination corresponding to their mining and geological conditions, and has an area of 20 km^2 (4000×5000 m) (Fig. 1).

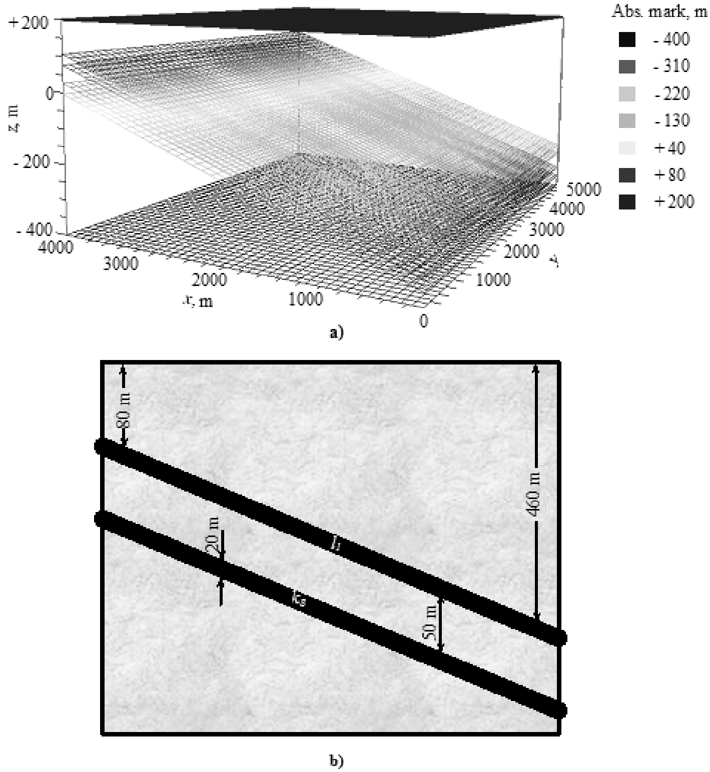

Figure 1 - Three-dimensional representation of geometry of simulated area (*a*) and a schematic section of "Novohrodivska 2" mine (*b*)

Thicknesses of productive strata in the model were assumed in accordance with the dependency of permeability of the underworked rock massif on a multiplicity of its underworking (on average equal to 10-40 thicknesses of a coal seam).

To specify the outer boundaries of the simulated area, the recommendations given in the papers [5] were used, according to which the tectonic disturbance (Novohrodivsky fault in the north of the mine field) is a screen in a path of groundwater movement. This determines the necessity of setting the hydrodynamically impermeable boundary in the fault area. In the southwest and southeast, where coal seams have a direct hydraulic connection with watered Paleogene-Neogene deposits, it is necessary to set the boundary condition of the third kind, reflecting the interconnection of groundwater flow rate of the Paleogene-Neogene horizon into the productive stratum with the difference of hydrodynamic pressure in them.

At the same time, the resistance determining the interconnection between the flow rates and the difference in pressure at the bassets of coal seams is determined by the total value of permeability of seams and Paleogene-Neogene deposits, recalculated in accordance with the dimensions of calculated blocks. In places of groundwater cross-flows between ”Korotchenko” - “Novohrodivska 2” mines and “Novohrodivska 2” “Novohrodivska 1” mines set boundary conditions of the second kind with flow rates corresponding to their specific values (Table 2).

Table 2

Distribution of values of specific cross-flows in zones of barrier pillars between mines

Abs. mark of interval of depth, m	Specific cross-flow (parameter km(B/L)	
	”Korotchenko” – “Novohrodivska 2”	“Novohrodivska 2”– “Novohrodivska 1”
–300…–250	0,36	0,16
–250…–200	4,20	0,12
–200…–150	0,40	0,21
–150…–100	0,36	0,02
–100…–50	0,36	0,10
–50… ± 0	0,26	0,15
± 0…+50	0,58	0,24

**Km* - water transmissibility, m^2/day; *B* and *L* - the width of a front and the length of a filtration path respectively, m

The internal boundaries of “Novohrodivska 2” mine model are stoping and development workings, displayed by the boundary conditions of the first kind with a hydrodynamic pressure equal to the absolute mark of bottom of coal seams. The position of these boundaries was determined by constructing a plan of mine workings in AutoCAD software environment and transferring the contours of excavation areas to simulated layers (Fig. 2). When simulating the mine operation after shutting down water drainage, internal boundary conditions were not set.

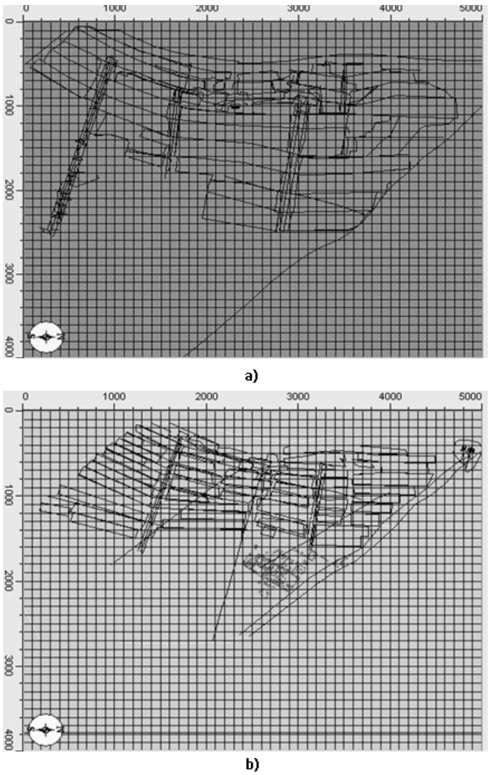

Figure 2 - Model contours of stoping areas based on the mine workings plan for the seam k_8 (*a*) and l_1 (*b*)

The volume of cavities within the mine field was represented as a sum of crack-porous and mined space. Its change in the model was set layer-by-layer on the basis of mining plans in each developed horizon with a thickness Δz [5]. According to the accepted design scheme, the mine shaft was a perfect borehole into which a time-dependent water inflow moves from aquifers. When schematizing the filtration properties of a massif of rocks, according to generalizations carried out in [6], the value of filtration coefficient of Mid-Carboniferous sandstones for the interval of 0-200 m was assumed to be equal to 0,2 m/day, and for the interval of 200-500 m - 0,08 m/day.

Based on the existing theoretical concepts [6-7] on the permeability of mined rock massif, the value of porosity and filtration coefficient in a range of simulated mine workings was set to an average increase of 7-10 times compared with zones outside of mining operations. The model discretization step in space was 100 x 100 m (2000 blocks in total), which made it possible to consider the configuration of workings within the mined seams at the mine scale with sufficient accuracy, while the time step did not exceed 20 days. Infiltration of precipitation in the upper layer of the model was set at 25 % of their average annual amount in the region. Zones of increased infiltration (settling ponds of mine waters, erosional relief dissections) were simulated by setting the intensity of infiltration feeding that is uneven over the area.

Capacitive properties of the strata of rocks lying above the upper boundary of mining operations (+120 m), were determined by the elastic capacity of the aquifer of weathering crust of Carboniferous (10^{-3}) and gravitational water loss of Paleogene-Neogene sands (0.1). The average value of effective porosity in a range of marks +120÷+155 m (basis of groundwater discharge in the valley of the r. Solona) was assumed equal to 0.2.

Solution of an inverse problem to prove adequacy of a developed model

Completed and planned stages of usage of resources of "Novohrodivska 2" mine are shown in Fig. 2 in the time section. The first stage corresponds to the period of mine operation and coal mining, and the second stage - to its liquidation with the operating drainage and maintaining the groundwater level at -157 m in the north wing. The next stage corresponds to the completion of drainage regime and flooding of mine workings to the mark +113,3 m (as of April 15, 2012). During this period, employees of JSC "Dniprogiproshakht" conducted observations of the mine's flooding rate, which provided data for solving the reverse geofiltration problem within the mine field. The start of operation of a hydrogeothermal module, with the aim of developing the thermal and capacitive resources of the mine for heat supply of Novohrodivka, is expected at the fourth stage.

The developed methodology is applicable to calculation of groundwater level changes within the mine boundaries throughout all four stages of its operation. However, for an adequate prediction of water inflows in the model, it is necessary to perform an epignostic modeling, the purpose of which is to correct a hydrodynamic role of external boundaries of aquifers and their filtration properties [8]. At the same time, the values of hydrodynamic parameters of aquifers should be characteristic of hydrogeological conditions of "Novohrodivska 2" mine. The basis for their variation was the results of pilot filtration works and measurements obtained by the Production Geological Enterprise "Artemivsk hydrogeological party" during the mine operation [2].

The inverse (epignostic) problem was solved under the conditions of non-stationary filtration mode, the main criterion of correctness of the solution of which was the similarity of actual and simulated values of the mine shaft flooding. The results of solution show that the model was able to almost completely reflect the dynamics of water level raising in the system of mine workings during the third stage of mine operation. Fig. 5.11 shows the model distribution of groundwater level in the mine field before and after the drainage shutdown (as of April 15, 2012, at least 2000 days), as well as its intermediate positions throughout the flooding period [9]. In this case, the absolute error between the actual and model level data is within 3 – 11 m, and the relative error does not exceed 10 % (Table 3).

Analysis of distribution of groundwater level shows that before the drainage is turned off, the cone of depression follows the contours of mining zones. The largest subsidence is confined to zones where the volumes of developed space are the largest. As flooding occurs, the noted patterns smoothen, while the rise of the groundwater level occurs unevenly, with a slowdown in intervals of the greatest concentration of mine workings and falling behind the water level in the shaft.

Table 3

The error of determining the dynamics of raising water level during flooding of "Novohrodivska 2" mine according to the results of simulation

Time since the beginning of flooding, days	Abs. mark of flooding water table, m		Absolute error, m	Relative error, %
	actual	model		
300	-120,50	-110,45	10,05	8,34
900	-1,15	-1,05	0,10	8,69
1200	25,40	22,90	2,50	9,84
1500	95,10	92,12	2,98	3,13
1800	110,80	99,85	10,95	9,88
2000	109,72	113,3	3,58	3,16

Thus, the created geofiltration model of "Novohrodivska 2" mine adequately reflects the dynamics of flooding of workings considering the specifics of mining-geological and hydrogeological conditions of the mine field. By using the suggested approach, it is possible to calculate the change of the level regime, the velocity and direction of filtration of groundwater in various mine horizons when using it to create a hydrothermal module. On the basis of the created and calibrated numerical model of geofiltration, the prediction of position of mine water level at the present time (June 15, 2015) and the beginning of the fourth stage of mine operation (launch of the hydrothermal module) was carried out. The results of solving the problem on April 15, 2012 (see. Fig. 3*b*) were assumed as basic conditions. The boundary conditions remained the same as in the inverse problem.

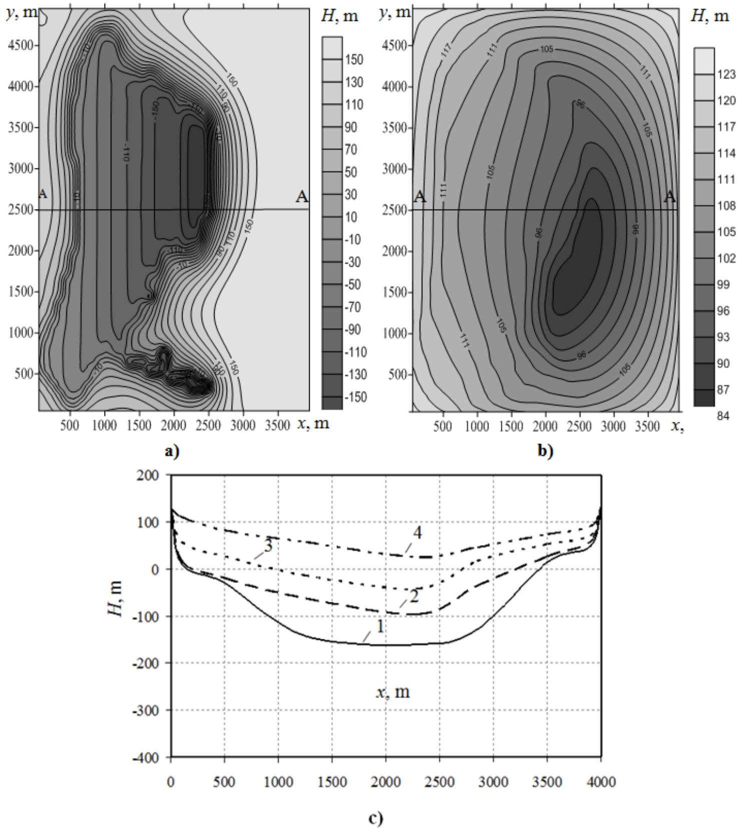

Figure 3 - Groundwater level (*H*, m) in the mine field before drainage shutdown (*a*), after 2000 days after shutdown (*b*) and at intermediate times along the profile A - A (*c*): 1-4 - after 100; 700; 1000 and 1500 days after drainage shutdown respectively

Fig. 4 shows the predicted position of groundwater level (plan view of hydroisogipsos) within the "Novohrodivska 2" mine field, obtained from the simulation results and characterized by the following features of dynamics of changes of the level surface. When the drainage is shut down, the tendencies to a gradual increase in groundwater level over time and a decrease in the hydraulic slope towards the center of the mine field remain that were established in the epignostic problem. The overall level difference is 30 m. The influence of it raising to the marks of +120...+130 m does not affect the flooding of day surface, since the closing of the upper aquifer of Paleogene-Neogene sands and the level of flooding of the developed space occurs above these marks.

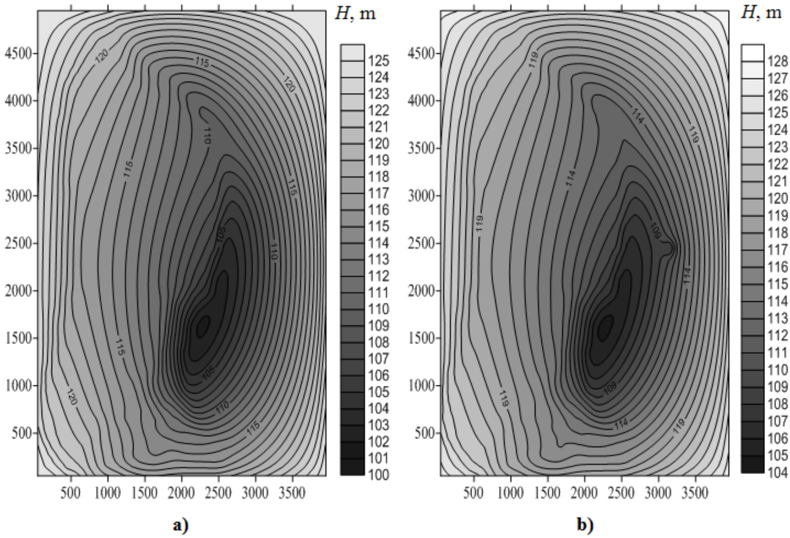

Figure 4 - Predicted position of groundwater level within a mine field as of June 15, 2015 (*a*) and a start of operation of hydrogeothermal module (*b*)

In general, the model adequately reflects the hydrodynamic situation and makes it possible to establish the main features of filtration within the mine field in existing conditions and with various design decisions aimed at the pumping out and injecting groundwater. The obtained model distributions of levels can serve as a basis for developing geotechnological schemes for a complex usage of thermal energy contained within flooded mine workings for heat and cold supply of buildings of various purposes through a system of operation wells, which is discussed in the next chapter.

Quantitative estimation of a geothermal resource of a flooded mine and profitability of its development with a help of heat pumps

To develop geotechnological solutions aimed at using the heat contained in flooded workings and disturbed rock massif, it is necessary to determine the existing thermal potential of "Novohrodivska 2" mine.

In this case, in the first approximation, it can be assumed that hydrodynamic parameters of seams do not depend on heat transfer processes [9], and the water temperature and temperature of a rock skeleton coincide at every point. Assume that mine water movement within the mine field occurs along the collapsed massif and flooded workings, heat exchange in the computed plane is absent [10], the H axis is directed down (Fig. 5). The heat flow q caused by heat of earth interior enters the flooded mined space from the bottom (from the depths). A neutral stratum of rocks, the temperature of which is constant and equal to the average annual temperature in the region (about +10°C), lies above, 6-7 m below the day surface. Under these conditions, a differential equation of heat conduction about H axis considering convection is

$$\frac{\partial^2 T}{\partial H^2} - \frac{V}{a} \cdot \frac{\partial T}{\partial H} = 0, \tag{1}$$

under the following boundary conditions

$$T = T_1 \text{ when } H=H_1;$$
$$q = -\lambda \partial T / \partial H \text{ when } H=H_2.$$

The general solution of equation (1) with given boundary conditions is [9]

$$T = T_1 + \frac{q}{\lambda B}\left[expB\left(H - H_2\right) - expB\left(H_1 - H_2\right)\right];\ B = \frac{V}{a}. \tag{2}$$

In this case, the thermal potential of mine water contained in the flooded workings is determined from the expression

$$Q=C \cdot \rho \cdot t \cdot V_v. \tag{3}$$

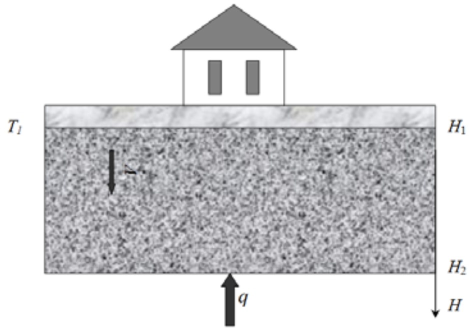

Figure 5 - Scheme for calculating temperature of groundwater within mine field

In formulas (1) – (3): T_1, H_1 - temperature and distance to the neutral stratum; H - depth of location; a, λ - thermal diffusivity and thermal conductivity of water-saturated rocks; V - vertical filtration velocity; Q - quantity of heat; C, ρ, t, V_v - specific heat capacity, density, temperature and volume of mine water, respectively.

Fig. 6 shows the temperature and existing thermal potential of mine water contained in flooded workings [10] calculated by formulas (2)-(3). It should be noted that the results of calculations are in good agreement with the actual data obtained by the Production Geological Enterprise "Artemivsk Hydrogeological Party" during temperature measurements at various horizons, and the total amount of thermal energy accumulated by mine water averages 1300 TJ [10].

Usage of heat energy from "Novohrodivska 2" mine during liquidation is associated with the periodic injection and pumping out of mine water from flooded mine workings. At the same time, water cooled as a result of heating the building to 7 °C will flow to the horizon of ±0÷+100 m with an average temperature of 12 °C, and water heated as a result of conditioning the buildings to 30 °C will be directed to the horizon with marks of -300÷-400 m and temperature 26 °C. There is a two-month period of inactivity between the periods of injection and subsequent pumping out of mine water: April-May - for cooling buildings and September-October - for heating. Due to the temperature difference between flooded horizons and the injected water, they will change their temperature both during the period of operation and during the period of inactivity of the geothermal module. In addition, loss of temperature during the water movement is inevitable in the process of injection of mine water due to lack of thermal insulation of wells. The given thermo-physical characteristics of usage of mine water for heat and cold supply of buildings indicate the need to fulfill predictions for changes of their temperature.

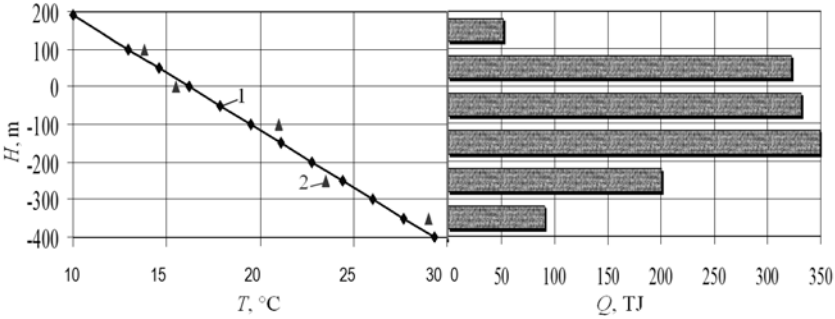

Figure 6 - Changes in temperature and amount of heat of water concealed in a flooded massif of "Novohrodivska 2" mine: 1 – 2, calculated and actual data respectively

Reducing the temperature of hot water in the well when it is pumped into flooded mine workings can be calculated according to the following calculation scheme [10]

$$T(z,t) = T_n + \frac{G}{\beta}(\beta\beta - 1) + (T_0 - T_n + \frac{G}{\beta})\exp(-\beta z); \qquad (4)$$

$$\beta = \frac{2\pi}{\frac{QCw}{\lambda_a} \cdot \ln\frac{2Z(t)}{d}}, \quad Z(t) = 2\sqrt{a_a}t,$$

where $T(z,t)$ - the corresponding temperature at a given depth z after t days since the start of water injection; T_n, T_0 is the temperature of the earth neutral stratum reduced to the well mouth and the water pumped; G - geothermal gradient; β - an indicator characterizing heat exchange with the environment; Q, C_w - flow rate and volumetric heat capacity of the injected water; λ_a,a_a - average coefficient of heat conductivity and thermal diffusivity of rocks surrounding the pipe; d - the outer diameter of the pipe.

The results of calculations by the formula (4) are shown in Fig. 7 [10]. From the graphs obtained, it is concluded that the temperature of injected water at the well bottom rises first and after some time (approximately by the middle of the injection period) stabilizes.

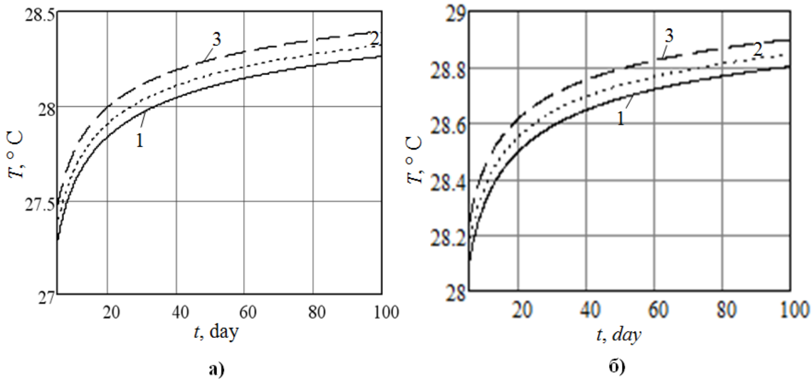

Figure 7 - Water temperature change at the well bottom on duration of injection with a flow rate of 450 (*a)* and 650 (*b*) m^3/day: 1,2,3 - at injection depth of 600, 550 and 500 m respectively

The average loss of water temperature during the flow along the well are approximately 1,5 °C (no more than 5%). In addition, an insignificant (no more than 0,5 °C) influence of the pumping rate on water temperature was found. Thermophysical properties of rocks characteristic for "Novohrodivska 2" mine were assumed in the calculations: G=0,026 °C/m; C_w=4187 kJ/m^3; λ_a=245 kJ/m·day·°C; a_a=0,05 m^2/day. Technological parameters of injection: T_0= 30°C; d= 0,2 m.

The change of temperature of injected water after entering the flooded mine workings and the rock massif can be approximately determined using the analytical solution of Lapshin N.N. on the basis of Lauwerier solution and formula [9]. These solutions do not consider conductive transfer in the aquifer, which is acceptable at high flow rates and injection rates; however, when developing a real geotechnological scheme that assumes simultaneous injection and pumping out of water of different temperatures through a system of several wells, it is necessary to use a numerical heat transfer model.

$$T(r,t) = T_0 + (T_w - T_0)\,\mathrm{erfc}\,\frac{\left(\sqrt{\lambda_1 C_1} + \sqrt{\lambda_2 C_2}\right)\pi\left(r^2 - r_s^2\right)}{Q\,C_в\sqrt{t-\eta}}; \qquad (5)$$

$$\eta = \frac{\pi\left(r^2 - r_s^2\right)m}{Q\,\overline{C}}; \quad \overline{C} = \frac{C_w}{C_{sk}},$$

where $T\ (r,t)$ is the temperature of mine water at a distance r from the well t days after the start of injection; T_w is the initial temperature of groundwater; $\lambda_1 C_1$ and $\lambda_1 C_1$ - thermal conductivity and volumetric heat capacity of the rocks of roof and bottom, respectively; r_s - well radius; r_s is the volumetric heat capacity of rocks containing the injected water.

Using the expression (5), it is possible to make a prediction of changes of temperature of water entering the flooded mine horizon. Fig. 11, a shows the change of this temperature at different distances from the bottom of injection well. Thermo-physical properties of rocks and technological parameters of injection are assumed as follows: λ_1=221 kJ/m·day·°C; λ_2=150 kJ/m·day·°C; z=550 m; C_1=1840 kJ/m^3; C_2=1656 kJ/m^3; Q= 50 m^3/day.

To estimate the change of water temperature during the inter-heating period, the flow rate of the injection well was conventionally assumed to be 0,1 m^3/day in the formula (5), what made it possible to consider only heat loss into the surrounding massif. The results of calculations showed that during this period of 60 days, the temperature of mine water does not fall below 28°C. The following calculation was made for the period of heating season (November-March) during the pumping out of mine water from a depth of 550 m. It was established that at this time the water temperature decrease was insignificant at first, and then it quite sharply reduced from 28 to 25,5°C.

Similarly, calculations were performed for the case of injection, inactivity and subsequent pumping out of cooled mine water, as a result of heating up the buildings to 7 °C, pumped into the flooded mine horizon at a depth of 100-200 m (Fig. 8*b*). The results of calculations show that during the period of pumping out the temperature of water used for cooling buildings varies slightly (from 8 to 9,5°C).

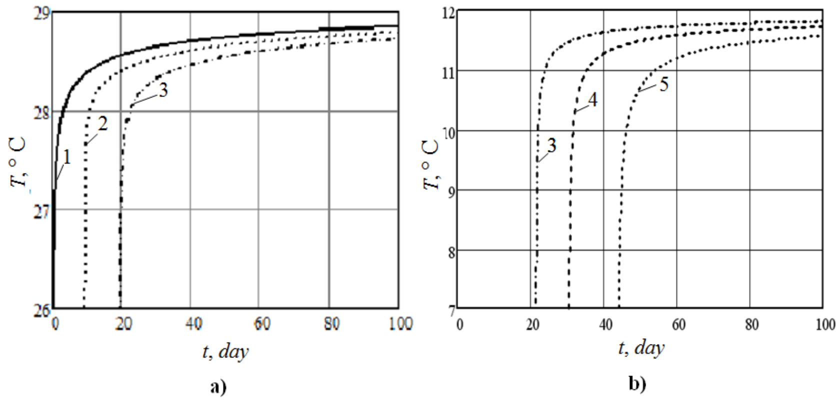

Figure 8 - Temperature change of warm (*a*) and cold (*b*) water, injected into ”Novohrodivska 2” mine: 1-5 - at a distance from the well bottom of 1; 15; 20; 25 and 30 m respectively

At the same time, the conversion coefficients of heat K_h and cold K_c, representing the ratio of the heat output of the pumps to the electricity consumed by them and determined from the following expressions, are taken as the main indicator of pump efficiency

$$K_h = h \cdot \frac{T_1}{T_1 - T_2};\ K_c = h \cdot \frac{T_2}{T_1 - T_2}, \qquad (6)$$

where h is the coefficient of thermodynamic perfection; T_1, T_2 - temperature of condensation (of heat consumer) and evaporation of the refrigerant (low-potential energy source), *K*.

To determine K_h of a heat pump using the formula (6), which uses mine water pumped out of the -300÷-400 m horizon as a low-potential source of thermal energy, it is necessary to set its temperature change (from 28 to 25,5 °C) determined from the expression (5). Also, the coefficient of thermodynamic perfection (assumed to be 0,6) and the temperature of heat consumer (the temperature of hot water entering the heating system, from 50 to 70 °C, depending on the outside air). Analysis of the obtained results shows a slight decrease in K_h (not more than 0.1), caused by a small fluctuation (1,3°C) of the temperature of mine water pumped out during the heating period. Similarly, K_c was determined, the value of which slightly increases (not more than 0,2) by the end of the summer period due to decrease in temperature difference between the heat consumer and the low-potential energy source.

Scientific and practical interest is the performance of a comparative analysis of usage of mine water in heat pumps with other types of low-potential sources of thermal energy (heat of outside air, groundwater and natural water flows). To do this, graphs (Fig. 9) were constructed in Mathcad software to change the K_h and K_c depending on the source and temperature of the heat consumer.

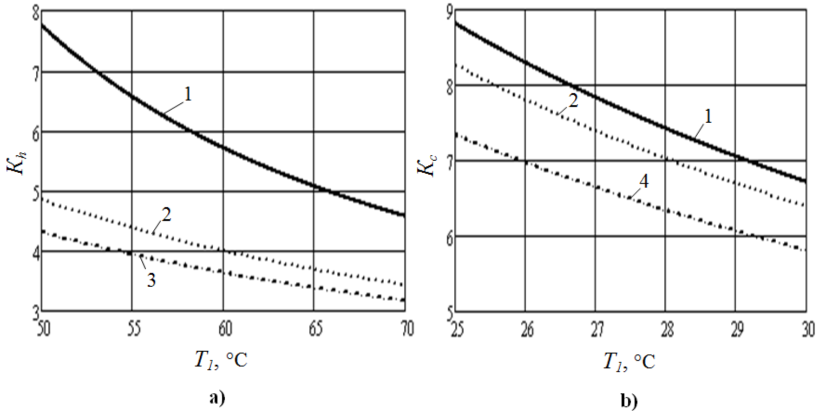

Figure 9 - Comparison of conversion coefficient of heat (*a*) and cold (*b*) of a heat pump when using: mine water (1), soil base (2), reservoirs (3) and groundwater (4) as a low-potential energy source

Graphs (Fig. 10) were also constructed to show heat energy savings during the heating and summer periods when using mine water in heat pumps. The following parameters were assumed in the calculations: during the heating period - the temperature of the soil base and water bodies is 10 and 5 °C respectively, the heat flow for heating buildings in Novohrodivka (15 thousand people) is 600 GJ/day; in the summer period - heat flow for air conditioning of buildings is 168 GJ/day.

Analysis of graphs in Fig. 9-10 shows that with the usage of mine water in a heat pump, the highest heat and cold conversion coefficients are achieved. Their usage is especially efficient in the heating period, causing K_h to be 1.5 and 2 times more than when using heat of the soil base and open water bodies as a source of low-potential energy. When air-conditioning the buildings, pump K_c operating on mine water also exceeds other variants - the heat of the soil base and the groundwater by 10 and 25% respectively.

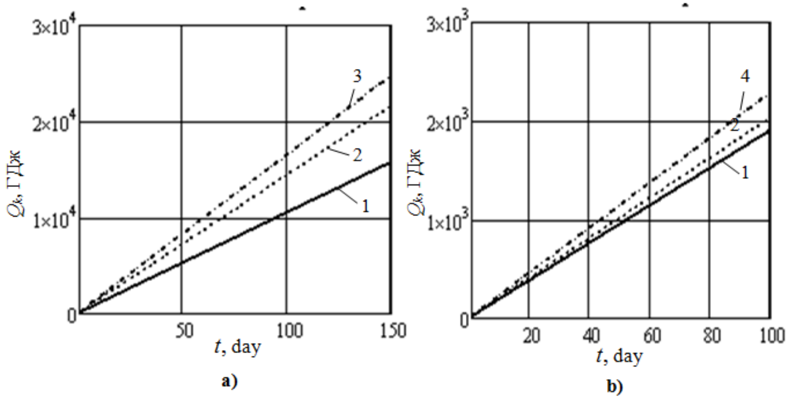

Figure 10 - Change of power Q_k used by a heat pump in heat equivalent for heating (*a*) and air conditioning (*b*) of buildings in Novohrodivka. See designations in Fig. 12

The amount of energy saved by a heat pump during the heating period when using mine water, in comparison with alternative low-potential energy, is on average 5,000 GJ. During the conditioning of buildings, this value is equal to 300 GJ. The results obtained indicate high efficiency and profitability of using mine water as an energy source in heat pumps.

Conclusions

Created geofiltration model of the mine field, based on the finite-difference solution of non-stationary planned filtration equations in "Modflow" software, displays the uneven character of permeability and water inflow in a mined massif depending on its crack-porous structure and volumes of the developed space. Performed on a basis of epigenostic modeling, the approbation of the model allowed specifying the hydrodynamic influence of external boundaries of a calculated area and filtration properties of rocks, as well as establishing the dynamics of lowering the pressure of groundwater in zones of coal mining and restoration of their level after mine drainage shutdown. At the same time, the absolute error between the actual and model data of distribution of mine water levels is within 3-11 m, and the relative error does not exceed 10 %.

With the help of the model, the predicted position of groundwater level within the mine field at the present moment of time and the planned launch date of the geothermal module was established. The results obtained made it possible to estimate the horizon-by-horizon variation of temperature of mine water and their natural thermal potential of 1300 TJ, within the boundaries of the flooded massif.

The performed thermodynamic calculations for the non-operational massif, based on the results of numerical simulation of geofiltration and analytical solutions, showed that the total heat loss in a process of pumping out, injection and storage of mine water does not exceed 15 %. The usage of mine water with a temperature of 26-28 °C as a low-potential energy source in heat pumps, in comparison with other alternatives (soil bases, surface water, groundwater), gives the greatest heat conversion coefficients (4.5-7.5), which allows saving a significant amount of heat pump power.

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INFLUENCE OF THE GAS INCLUSIONS IN QUARTZ GRAINS OF SANDSTONES ON OUTBURST OF ROCKS

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Abstract. The article presents the results of the study of volumes of gas inclusions in quartz grains of oligomictic sandstones of Donets coal basin. It describes the main genetic types of inclusions, their conditions of formation, shape, size, as well as features of the transformation under the conditions of catagenesis of studied rocks. The presence and distinctive features of the primary and secondary inclusions were established by studying of their homogenization temperatures. Thanks to these studies, it was found that the homogenization temperature of the primary inclusions is 1.5-2 times higher than for secondary ones. A characteristic feature of secondary inclusions in quartz grains of carbonic sandstones of the Donetsk Basin is that they decorate the plastic microdeformations of these grains, thereby forming numerous Boehm stripes. The gas inclusions of Boehm strips are indicators of paleotemperature, and the Boehm strips themselves carry information about paleopressure. To calculate the volume of gas inclusions, a new method was proposed. This method is easy to use and does not require significant financial expenses. For its implementation, standard petrographic thin sections are used, which are examined using an optical microscope with a total magnification of 1200 times. Using the proposed method, the volumes of gas inclusions in quartz grains of sandstones of different substages of catagenesis were established. The largest volumes of gas inclusions are established for the middle substage of catagenesis. Considering that rock and gas outbursts occur only at the middle substage of catagenesis, this confirms the theory about the additional effect of gas inclusions on the progress of gas-dynamic phenomena in mine opening.

Introduction. At the present time emissions of coal, sandstone, salt and porphyrite are described [1]. Common to all of these rocks is their belonging to sedimentary rocks and the presence of significant amount of gases under high pressure. Most often the main gases are methane or carbon dioxide with different amounts of impurities. Sandstone outburst is one of the main problems that occur during the development of coal deposits. It leads to deteriorating safety of mine works and increasing the cost of coal. The study of this problem in the coal-bearing sediments of Donbas, scientists are engaged in fact from the middle of the XX century. Particular attention was paid to coal-bearing sandstones. This is due to the fact that sandstones are the main rocks in which preparatory workings are laid, since they are the most powerful and resistant to rock pressure compared to other rocks (mudstones, siltstone and limestone). However, sandstones are also the main reservoirs, which are characterized by accumulations of gas of both industrial value and numerous microdeposits. Significant gas accumulations and low permeability of rocks, which does not allow sufficiently degassing the coal massif and extract gas as associated energy raw materials, create favorable conditions for the occurring of gas-dynamic phenomena in coal mines.

In the 70s-80s of the last century, scientists who were researching gas inclusions in minerals began to pay attention to gas-dynamic phenomena in sandstones, salt, and other rocks. According to E. Roedder, the cause of gas-dynamic phenomena in salt mines is the natural decrepitation (explosion, opening) of gas inclusions in thousands of tons of salt, which led to rock outburst in the mine workings [2-6].

The presence of gas inclusions in minerals is possible in rocks of different genesis. For example, petrographic studies of nepheline from the Khibiny apatite deposit (Kola Peninsula) showed a large number of small fluid inclusions. The author [7] points out that the shape of the inclusions is predominantly rounded, sometimes there are inclusions with well-marked faceting elements. The size of the inclusions does not exceed 30 microns. Within individual nepheline crystals, inclusions are distributed unevenly and form clusters of irregular shape. The composition of these inclusions is predominantly methane (up to 70-90% methane), but along with the combustible gases of the hydrocarbon composition, bituminous substances of the oil series have also been found that fill the pores and the smallest cracks in the rocks.

In addition to alkaline rocks of the Khibiny massif, gases and hydrocarbon bitumens are established in quartz of granites, granite-gneisses and in gneisses of the Berezovsky gas-bearing region and other deposits.

As a result of a microscopic study of the salts of two Caspian deposits, Tyulyus and Ayrshagyl, it was found that the salts of both deposits in a large volume are saturated with gas-liquid inclusions, which characterizes the presence of closed microporosity [8]. The shape of the inclusions is cubic or rectangular. The size of the inclusions reaches 50 microns. In fig. 1 shows gas and gas-liquid inclusions in thin sections of salt samples taken from the core of oil wells in the Volga region.

a *b*

Fig 1. Gas-liquid inclusions in the salts of Ayrshagyl deposit *a* and Tyulyus deposit *b*, 250^X magnification

The study of inclusions in samples of halite in the Dnieper-Donets Basin allowed S. Shekhunova [9] to establish liquid, gas-liquid and three-phase inclusions in size from <1 to 200 microns, the correct cubic shape. The total gas saturation of the solutions of inclusions ranges from 1-10 to 1000 cm^3/l. The chemical composition of gases is nitrogen (up to 87%), carbon dioxide (up to 10%), methane and its homologs (~ 3%), oxygen (up to 1%). This indicates a significant effect of the atmospheric gases of the Earth in filling the salt inclusions of the specified region, formed both as a result of the growth of salt crystals and during their tectonic deformations at the micro level.

The study of the carbon sandstones of the Donets Basin at the micro level allowed us to reveal numerous gas inclusions in the quartz grains of the rocks. The composition of secondary inclusions was established using the overall gas analysis method. The overall gas research consists in the destruction of a certain amount of sample in vacuum using ball mills of various designs and obtaining the average composition of the gas components. Using this study, it was found that secondary inclusions in quartz grains of sandstones are mainly represented by methane, carbon dioxide and nitrogen [10].

The presence of gas inclusions in the minerals of rocks of different genesis with increasing temperature and pressure leads to the appearance of internal stress in the mineral, which provokes the natural decrepitation of gas inclusions. It is decrepitation of gas inclusions that can act as an additional energy source of rock emissions. Gas in closed pores, under high pressure, weakens the strength of sandstone cement and, together with gas in open pores and cracks, leads to the destruction, crushing and removal of rocks during emission.

According to V. Yakshin [11], the volume of gas inclusions in gangue quartz reaches 10 cm^3 or more per 100 g of this mineral. The gas pressure in the inclusions reaches 500 MPa. The huge amount of gas inclusions in the quartz grains of the sandstones of Donets Basin, and the possible pressure under which they are located, allows us to consider them as an additional factor affecting on the progress of gas-dynamic phenomena in coal mines.

This brief analysis of publications suggests that fluids in general, and gases of different composition, in particular, are contained in minerals and rocks of all three types: magmatic, metamorphic, sedimentary. During mining works, gas inclusions in the rocks can lead to all sorts of gas-dynamic phenomena. Since this direction has extensive research and a lot of publications, we will focus on a narrower direction - emissions of sandstones containing coal seams. This problem, concerning the safety of the work of the miners, the integrity of the equipment, the additional costs and, ultimately, the cost of the final product for the Donbass coal deposits, began in the middle of the last century. During this time, several thousand emissions occurred in the Donbass, which leads to significant material, social and environmental problems.

The aim of this work is to study the regularities of formation and subsequent transformation gas inclusions in clastic quartz of gas-bearing sandstones of the Donets Basin on different stages of catagenesis, their typification, to determine conditions of formation and transformation of this inclusions, establish their volumes on the example of different coal-bearing areas of basin, as well as the substantiation of practical significance of the obtained results in the context of studying the rock outburst and changes of reservoir properties of sandstones.

Materials and methods. During the investigation, sandstone thin sections were studied. Samples of sandstone were taken within the distribution of different grades of coal. This made it possible to analyze sandstones closed porosity of different stages of transformation. It is known that the transformation of sandstones of Donets Basin occurs under the influence of catagenesis and tectonic load. The substage of catagenesis is conditionally determined by the adjacent coal rank. Sandstone sections were examined using a POLAM R-111 optical microscope at magnifications from 100 to 1200 in reflected, transmitted, and oblique illuminations [12]. Data processing was carried out using personal computer.

Main part. Gas inclusions in the quartz grains of Donets Basin sandstones are represented by two genetic types - primary and secondary inclusions.

Primary inclusions were formed at the time of crystal formation in the parent rocks. They are well identified by optical research, since they are mainly two-phase and consist of liquid and gas phases (Fig. 2). In general, in the quartz grains of the sandstones of the Donets Basin, their amount is 1-2% of the total amount of all inclusions. The characteristic feature of primary inclusions is their shape. With an increasing of the stage transformation of rocks and tectonic load, the primary inclusions acquire the structural or morphological form of the host mineral. The size of primary inclusions usually exceeds the size of secondary inclusions, on average 1,5-2 times. The size of the primary inclusions in the quartz grains of the sandstones of Donets Basin on average 2-3 μm.

Fig.2 Primary gas-liquid inclusions in quartz of Donets Basin sandstones, 250^X magnification

Secondary inclusions are formed in early catagenesis, initially in the form of plastic and brittle deformations into which fluid from intergranular pores penetrates. Further, defects filled with fluid are healed, forming fluid inclusion chains. They were first described by Augustus Boehm in 1883, in the Alpine deposits. Later, these defects were called “Boehm stripes” [13].

Thus, the secondary inclusions in the quartz grains of the sandstones of Donets Basin are presented in the form of Boehm stripes (Fig. 3). Boehm stripes are presented both in catagenesis, in terrigenous sediments, and in metamorphic and igneous rocks. But with the transformation of rocks from one stage to another, for example, from sedimentary rocks to metamorphic or igneous, the inclusions are completely transformed or disappear, since the above processes are characterized by a complete structural reorganization of the rocks. In the study of sandstone sections, Boehm stripes are well identified by oblique illumination method.

Fig. 3 Quartz grain with Boehm stripes, carboniferous sandstone of Donets Basin, oblique illumination method, 100^X magnification

Boehm stripes in the quartz grains of Donbas sandstones formed during the catagenetic transformation of sandstones directly in the conditions of coal basin [14]. The prevailing development of Boehm strips in sandstone quartz grains is explained by increased gas content formed during the process of coal carbonization and dispersed organic matter in sedimentary rocks. Sandstone, as the main reservoir, accumulates gas in open pores and cavities. During the transition from the early substage of catagenesis to the middle one plastic microstrains begin to form. Fluid from the intergranular space migrates to microdeformations in the clastic grains of sandstones and heals these microdeformations, thereby forming numerous chains of gas inclusions. Later on, at the middle substage of catagenesis, these microdeformations are healed and gas inclusions are altered. They acquire an isometric form, clearly identified in the optical study. The largest number of Boehm strips is found in quartz grains of sandstones from the middle substage of catagenesis.

In the late substage of catagenesis, the sizes of gas inclusions decrease (Fig.4). This is explained by the fact that, with an increasing of catagenetic transformation and tectonic loading, the inclusions are divided to the smallest size and form pores of the size of hundredths and thousandths of microns. Subsequently, with an increasing of pressure and temperature, the gas from the inclusions migrates into zones with lower pressure, for example, into fractured zones or into intergranular space.

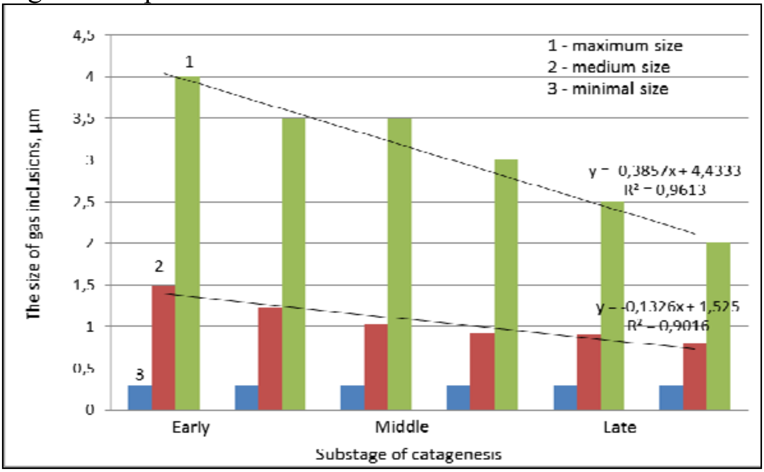

Fig. 4 Influence of the catagenesis on changing of gas inclusions size in quartz grains of sandstones

Thus, with the help of the conducted research, the ontogenesis of secondary gas inclusions has been established. This process includes the emergence of secondary inclusions in the early substage of catagenesis, maximum development in the middle substage, and annihilation in the late substage of catagenesis.

Investigation of gas and gas-liquid inclusions in clastic grains of rocks also has one more important practical interest. A significant pressure in the inclusions and their quantity create an additional volume that must be taken into account at the time of studying reservoir properties of rocks, in particular porosity. Determining the volume of gas inclusions in the clastic grains of rocks will help to establish an additional amount of closed micropores gas of the Donets Basin gas-bearing sandstones.

In general, closed porosity of rocks is determined as the difference between absolute and open porosity. It is believed that closed porosity has only a scientific interest, so in practice it is ignored.

The disadvantage of determination the closed porosity by the calculation method, as the difference between absolute and open porosity, is that we only get the values of the volume of pores that is contained in the cement of the rock. The amount of gas that is represented as inclusions in the clastic grains of rocks is not actually taken into account, since in determining the total porosity by the grinding method, the investigated rock is not always crush to the size necessary for the opening of these inclusions (to microns and nanometers).

Considering that the inclusions in the clastic grains of rocks are also part of the closed porosity, it was proposed to divide the closed porosity into the cement closed porosity (the volume of closed porosity contained in the cement rock material) and grain closed porosity (the volume of gas inclusions in clastic rock grains).

To establish the volume of closed grain porosity, a method was proposed. The method is easy to use, economically advantageous, because it does not require additional expenses. To implement it, standard petrographic thin rock sections are used, which are made in laboratories of geological organizations to determine various indicators in minerals and rocks. The thin sections are examined on a POLAM R-111 type microscope. The study is conducted at a magnification of 1000-1200 times, using an object micrometer.

The essence of the method is that the volume of gas inclusion is determined by the ratio of the area of this inclusion to the area of the investigated part of in the clastic grain of the rock. Therefore, choosing an informative part of clastic grain in thin sandstone section, it is necessary to tie it to the correct geometric shape. In most cases, it can be square or rectangular. Using the well-known formulas for determining the area of a square or rectangle, it is necessary to determine the area of the investigated part of thin section. When the area of the investigated part is known, it is necessary to calculate the area of all inclusions that are in this part of clastic grain. The volume of gas inclusion with a diameter of less than 0,4 mm without a significant error can be taken equal to the volume of the sphere [3]. But since the sphere is a three-dimensional figure and a thin section is a two-dimensional subspace, we take each inclusion as a circle and calculate the area of inclusion through the radius of the circle. In the end, we calculate the total area of all inclusions in the part of the studied grain. In percent by the formula (1), we determine the volume of inclusions in the clastic grain of the rock

$$V_{incl} = (S_{incl} \div S_{inf.p}) \times 100\% \quad (1)$$

where $V_{\text{incl.}}$ is total volume of gas inclusions;

$S_{\text{incl.}}$ is total area of inclusions;

$S_{\text{inv.p.}}$ is the area of the investigated part of thin section.

Examining the required number of investigated parts in the each section of the rock, using statistical data processing, we obtain the average value of the total volume of gas inclusions in the rock interlayer [15].

Using this method, it was found that the largest volumes of gas inclusions are characteristic of the middle substage of catagenesis and on average reach 4 % (Fig. 5). The early substage of catagenesis is characterized by insignificant volumes, which constitute no more than 2 %. This is due to the fact that the microdeformations in quartz and their gradual filling with gas take place in the early substage of catagenesis. The middle substage is characterized by the largest number of microdeformations. Gas inclusions that fill microdeformations acquire clear geometric shapes, so-called negative crystals, and relatively stable sizes. Sandstones of the middle substage of catagenesis are the most informative in studying inclusions in clastic grains. At the late substage of catagenesis, the volumes of gas inclusions decrease and reach 2 %. This is due to the fact that in the late substage of catagenesis, under the action of high pressure, the structure of quartz begins to change. Monolithic detrital grain turns into an aggregate of small blocks that are cleaned under the action of the so-called dry remelting or structuring process, deformation and micro-inclusions go to the newly formed boundaries.

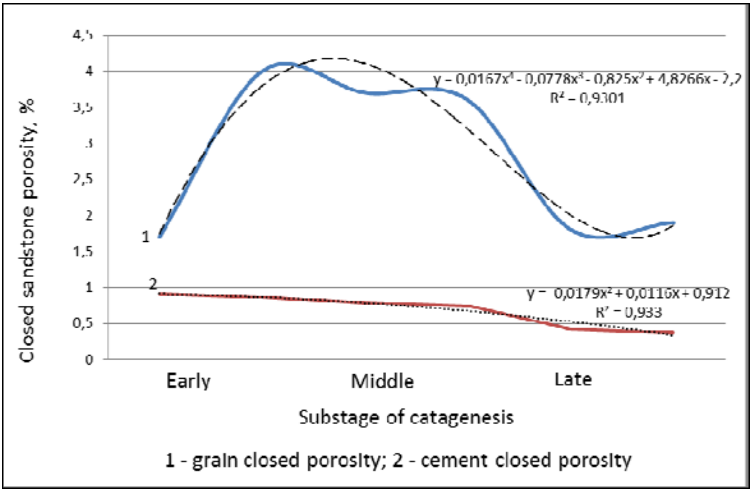

Fig. 5 Influence of the catagenesis on changing of sandstones closed porosity

The described process of ontogenesis of fluid inclusions of essentially gas composition for the conditions of the Donetsk coal basin is probably a general pattern of transformation of sandstones of the catagenesis stage. But such an assertion requires additional studies of these rocks from a certain stage of lithogenesis of different sedimentary basins, both coal, oil and gas, terrigenous, etc. The problem is that the Donbass underwent a significant uplift during postcarboniferous time and the underlying deposits, with a thickness of several kilometers were washed away. Now on the territory of the so-called Big Donbass coal seams are deposited at shallow depths or even come to the surface, including anthracite.

The reasons for the uplift have not been fully elucidated and have been the subject of scholarly discussion over the years. The most common causes of uplift are salt diapirism, mantle plume or joint uplift under the influence of a combination of factors [16]. The problem of the formation of significant gas accumulations of hydrocarbon composition in the closed micropores of quartz sandstones of Donbass can cause a corresponding tectonic impact on these rocks. But such a pattern can also be general. This requires additional studies and samples of sandstones from another sedimentary basin, where tectonic stresses did not manifest as in the Donbas.

It is important to note that the volumes of gas that we obtain using this method are not absolute. Conducting a study of thin sections with a magnification of 1000-1200 times the minimum size of inclusions, which we can observe, is 0.3 microns. Conducting research at larger magnifications would allow us to investigate smaller inclusions and, accordingly, establish additional gas volumes. This method is not aimed at establishing absolute values, but at establishing regularities of changing the volume of inclusions under the conditions in which the studied rock is located.

From the graph in fig. 5, it can be seen that the values of closed grain porosity are several times higher than those of closed cement porosity, which averages 1%.

The largest volumes of gas inclusions are established for the middle substage of catagenesis and reach 4 %. Considering that rock and gas outburst occur only at the middle substage of catagenesis, this confirms the theory put forward earlier about the additional effect of gas inclusions on the gas-dynamic phenomena in mining workings.

The volume of gas inclusions must also be taken into account during the determination of the reservoir properties of rocks. Determining the absolute porosity in laboratory conditions, it is necessary to grind the rock to the size that would allow revealing the smallest inclusions in the rock-forming grains. Part of the gas contained in the rock-forming grains will substantially supplement the data of absolute porosity of the rocks.

For example, for sandstones of the middle substage of catagenesis (Stakhanov mine, Krasnoarmeysky district, Donets Basin), an average of 3,5 % of gas in closed micropores of quartz grains was established. The absolute porosity of the studied sandstone is 7 %, but considering the volume of gas in quartz grains, it can reach 10-10,5%.

Previously, inclusions in detrital grains of terrigenous deposits did not take into account during determination of closed and absolute porosity, and when determining reservoir properties of rocks, they were crushed to approximately the sizes of average diameters of rock-forming sandstone grains 0,2÷1 mm. These data were obtained by us in the study of sandstone samples after their grinding and investigation of reservoir properties in several third-party laboratories (Poltava, Dnepr, Donetsk). To obtain more reliable data, we need special mills or new methods that allow crushing the detrital grains of sandstones to a micron size, or perhaps even smaller. Similar studies are planned to be performed at the Laboratory of the Study of Structural Changes in Rocks of IGTM NAS of Ukraine in the coming years.

Conclusion. The study of gas and gas-liquid inclusions in the clastic grains of rocks has an important scientific and practical interest. These inclusions involve genetic information about minerals, as well as being a source of data on secondary transformations of rocks. The results of the study of gas inclusions in the quartz grains of the sandstones of Donets Basin allowed establishing the basic regularities of their transformation at different substages of the catagenesis. The ontogenesis of secondary gas inclusions has been established. This process includes the emergence of secondary inclusions in the early substage of catagenesis, maximum development in the middle substage, and annihilation in the late substage of catagenesis. Cement closed porosity depends in large part on the mineral composition of the micas, the mineralogical form of carbonates, and in catagenesis it is transformed simultaneously with the change in the structure and form of these minerals. The developed method of calculating the volume of gas inclusions, allowed establishing that the volume of grain closed porosity on average is 1,5-2 times higher than the volume of cement closed porosity. These volumes must be considered as additional during the determination the reservoir properties of rocks and forecasting gas-dynamic phenomena in mining workings.

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PRODUCTION OF NANOSTRUCTURED IRON COMPOUNDS FROM METALLURGICAL WASTES

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Abstract

The problem of utilization of waste from the metallurgical industry is closely related to the development of adequate methods for the processing of inorganic substances, which accumulate in huge quantities in the dumps after production processes. A thorough study of the mineral and elemental composition of the dust concentrates of one of the metallurgical plants made it possible to subsequently apply fundamentally new technological methods for their processing. The criterion for the selection of technology for processing dust concentrate was its chemical composition. The main elements are carbon (C) and oxygen (O_2), the contents of which were 32.78 and 37.57 mass%, respectively. The content of metals in the form of metal phases or their oxides was: Fe -16,1; Zn - 2,2; Ca - 3,3; Pb - 0,7; Mn - 0,4; Co - 0,02; Cu, As, and Cd - 0,01 wt%. The content of sulfur (S) and phosphorus (P) was 1,0 and 0,1 wt%, respectively. The results of this study are devoted to the production technology of nanostructured iron powders, obtained from secondary raw materials, which can be used on an industrial scale - as catalysts in inorganic and organic synthesis, production of modified automotive and aviation fuels, as well as in the production of electric welding electrodes and in other technological processes.

Introduction

The dust concentrate is formed mainly in the blast metallurgical process, and is the material that is formed in the process of cooling and filtration of exhaust gases. In this study, a multicomponent mixture of metallic and mineral phases present in dust concentrate, was used. The main solid-phase components are amorphous carbon and ferrosilicon, on which the newly formed ultradisperse particles of iron, zinc, manganese and their oxides are adsorbed. Considering the fact that each year about 300 thousand tons of such a dust concentrate are produced from each blast furnace, it is logical to assume the practical significance of our research. According to X-ray fluorescence (Table 1) and X-ray phase (Table 2) analyzes, the main metals of the concentrate are iron (16,1 mass%), calcium (3,3 mass%) and zinc (2,2 mass%).

Table 1

Chemical composition of the dust concentrate (mass%)

Fe	Si	Ca	Zn	S	Pb	Mn	P	Co	Cu	As	Cd
16,1	5,8	3,3	2,2	1,0	0,7	0,4	0,1	0,02	0,01	0,01	0,01

The chemical composition and mineral-phase composition of the dust concentrate are presented in Table 1 (X-ray fluorescence analysis data) and Table 2 (X-ray phase analysis), respectively.

Table 2

Mineral composition of blast furnace dust concentrate
(x-ray phase analysis data)

Mineral phase	Main chemical element	Mineral content (%)	
		in a native concentrate	after reductive roasting
Amorphous carbon	C	32,78	0
A mixture of iron-containing mineral phases - hematite (αFe_2O_3), magnetite ($FeFe_2O_4$), goethite ($\alpha FeO(OH)$) and reduced iron $\overset{\circ}{Fe}$)	Fe	23,01	47,56
SiO_2	Si	7,39	31,0
$CaSO_4$	S	4,245	0
CaO	Ca	2,868	2,15
ZnO, $\overset{\circ}{Zn}$	Zn	2,738	16,26
PbO_2	Pb	0,808	2,11
MnO_2	Mn	0,633	0,92
P_2O_5	P	0,229	0
As_2O_5	As	0,030	0
CoO	Co	0,025	0
CuO	Cu	0,012	0

CdO	Cd	0,011	0
Oxygen in other oxides and crystallohydrates	O	25,221*	0

note: * - an indicator characteristic of dust concentrate that has been stored for a long time in dumps (several years) in conditions of full contact with moisture

The main mineral phases are amorphous carbon (32,78%), minerals containing iron (III, II) oxides - hematite (αFe_2O_3), magnetite ($FeFe_2O_4$), goethite ($\alpha FeO(OH)$) and reduced iron ($\overset{\circ}{Fe}$) (total 23,01%), silicon oxide (7,39%) and calcium sulfate (4,25%). The noteworthy phases from which additional zinc extraction is possible - zinc oxide (ZnO) and reduced fine zinc ($\overset{\circ}{Zn}$), totaling 2,738%. Certain concerns are caused by the relatively high content of lead oxides (0.808%), which implies the use of additional measures to prevent the release of this element into the environment during processing. Because lead belongs to the first class of hazard compounds, the maximum daily average inorganic lead concentration in air is 0,0003 mg/m^3. In addition, the dust concentrate contains toxic compounds such as manganese (IV) oxide (0,633%) and arsenic oxide (0,03%). The maximum permissible concentrations of manganese and arsenic in the atmosphere of metallurgical enterprises are 0,001 mg/m^3 and 0,003 mg/m^3, respectively. The dust concentrate also contains an insignificant amount of phosphorus (V) in the form of P_2O_5 (0,229%), the extraction of which in the form of an enriched fraction is very problematic, or practically unprofitable. Elements such as cobalt, copper, and cadmium are present as oxides - CoO (0,025%), CuO (0,012%), and CdO (0,011%), whose preparative extraction is also unprofitable.

Given the elemental composition of the dust concentrate, our attention was focused on the development of an adequate technology for the utilization of iron.

In addition, Table 2 (column 4) shows the change in the content of the elements in the dust concentrate after a recovery burn. Changing the content of individual phases requires a special analysis and suggests the feasibility of using such technological treatment as a previous regenerative spoil before the removal of target metals - iron and, possibly, zinc.

Research results

Electron microscopic studies (with an X-ray spectral analysis of the constituent components) of the carbon phase indicate that on its surface there are ultradispersed (nanosized) metal particles - iron and zinc, which do not aggregate with each other. Moreover, these particles are represented in a wide dimensional range - from tens of nanometers to the first micrometers (Fig. 1). Depicted in fig. 1 the structure of the carbon matrix, which holds the metal particles, confirms the RFA* data that it is amorphous carbon: on the photo - spongy structure.

*- X-ray phase analysis

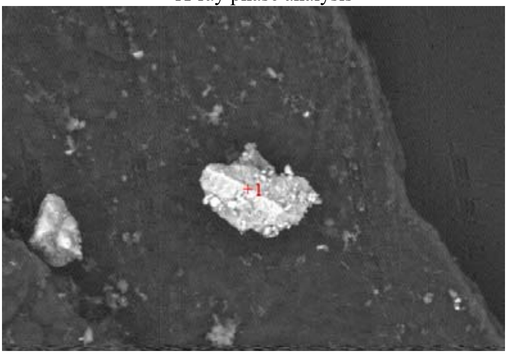

Fig. 1. REM*-image of the carbon phase with adsorbed on its surface of the "*de novo*" metal particle size 28,1 μm x 15,0 μm (magnification 1570 x)

In table 3, a chemical composition of metal parts is shown (in Fig. 1) at the point, in the same position as “+1”. The main element is iron (98.92%). The impurity elements - zinc, silicon, chromium, and aluminum - are 0,4%, 0,39%, 0,19%, and 0,1%, respectively.

Table 3

The chemical composition of the metal particle formed "*de novo*" on the surface of the carbon particle present in the dust concentrate (%)

Fe	Zn	Si	Cr	Al
98,92	0,40	0,39	0,1	0,10

*- X-ray emission microscopy

Modern metallurgical processes are based not only on the processing of natural ores, but also on the processing of a significant by weight (up to 30%) of the amount of secondary iron-containing raw materials. A considerable amount of the latter consists of secondary materials that contain, in addition to iron and its compounds, also non-ferrous metals, in the vast majority zinc, and much less - tin, nickel, chromium and others. Taking into account the fact that the temperature regime in modern metallurgical processes reaches 1773 K (1500°C) and above, the gases produced contain initially vapor or liquid phase - aerosols of the corresponding non-ferrous metals, and as they are cooled, they are condensed forms. Therefore, in the composition of the metal particles formed by "*de novo*" on a carbon matrix, there are impurities of the corresponding elements: Zn (0,4%), Si (0,39%), Cr (0,19%) and Al (0,1%). For example, the temperature of condensation of zinc vapor to the liquid state at normal pressure is about 906°C, and to the metallic state - about 420°C. Investigation of the mechanism of the formation of "*de novo*" metal phases in dust emissions in the blast furnace process - the subject of the next publication.

Taking into account the fact that the dust concentrate contains 32,78% amorphous carbon, which can be used as an effective pyrometallurgical iron oxide reducing agent (Fe_2O_3, Fe_3O_4) [1], we have tested the reducing carbon dioxide for the purpose of preliminary enrichment metal phase ($\overset{0}{Fe}$) and ensuring maximum removal of iron from waste (off-balance metal source).

The process that occurs in the working of a blast furnace, i.e. high temperature (1773 K), is accompanied by the formation of liquid iron ($\overset{\circ}{Fe}$) and liquid slag - calcium silicate ($CaSiO_3$) and gasified carbon oxide (IV) and carbon oxide (II). In this case, there is a cascade of oxidation-reduction reactions involving carbon and present in the dust concentrate of mineral phases

$$\underset{magnetite}{\overset{+2}{Fe}O\cdot\overset{+3}{Fe_2}O_3}+4\overset{+4}{Si}O_2+10\overset{0}{C}+CaCO_3\xrightarrow{T,2073K,+20e^-}, \quad (1)$$

$$\underset{ferrosilicon}{3\overset{0}{Fe}\overset{0}{Si}} + Ca\overset{+4}{Si}O_3 + 10\overset{+2}{C}O\uparrow + CO_2\uparrow$$

$$\overset{0}{C}+\overset{0}{O_2}\uparrow\xrightarrow{T,1773K,-4e^-}\overset{+4}{C}\overset{-2}{O_2}\uparrow, \quad (2)$$

$$\overset{+4}{C}\overset{-2}{O_2}\uparrow+\overset{0}{C}\xrightarrow{T,1770K,+2e^-}2\overset{+2}{C}\overset{-2}{O}\uparrow, \quad (3)$$

$$\underset{hematite}{3\overset{+3}{Fe_2}O_3}+\overset{+2}{C}O\uparrow\xrightarrow{T,873K,+2e^-}\underset{magnetite}{2\overset{+2}{Fe}O\cdot\overset{+3}{Fe_2}O_3}+\overset{+4}{C}O_2\uparrow, \quad (4)$$

$$\overset{+2}{Fe}O\cdot\overset{+3}{Fe_2}O_3+\overset{+2}{C}O\uparrow \xrightarrow{T,873K,+2e^-} 3FeO+\overset{+4}{C}O_2\uparrow, \quad (5)$$

magnetite

$$\overset{+2}{Fe}O+\overset{+2}{C}O\uparrow \xrightarrow{T,873K,+2e^-} \overset{0}{Fe}+\overset{+4}{C}O_2\uparrow, \quad (6)$$

$$\overset{+3}{Fe_2}O_3+3\overset{0}{C} \xrightarrow{T,1773K,+6e^-} 2\overset{0}{Fe}+3\overset{+2}{C}O\uparrow, \quad (7)$$

$$\overset{+2}{Zn}O_{(solid)}+\overset{0}{C} \xrightarrow{T,\ 823K,\ +2e^-} \overset{0}{Zn}_{(liquid)}+\overset{+2}{C}O\uparrow. \quad (8)$$

In addition, when regenerative firing, amorphous carbon (C) interact with silica (SiO_2), forming elemental silicon (Si) and silicon oxide (II) (SiO)

$$\overset{+4}{Si}O_2+2\overset{0}{C} \xrightarrow{T,1773K,+4e^-} \overset{0}{Si}+2\overset{+2}{C}O\uparrow, \quad (9)$$

$$\overset{+4}{Si}O_2+\overset{0}{C} \xrightarrow{T,1773K,+2e^-} \overset{+2}{Si}O+\overset{+2}{C}O\uparrow. \quad (10)$$

The reaction products (9) and (10), in turn, actively interact with the reduced iron ($\overset{\circ}{Fe}$) - the product of reactions (6) and (7), forming a new product - ferrosilicon (FeSi):

$$\overset{0}{Fe}+2\overset{+2}{Si}O \xrightarrow{T,2273K,-2e^-} \overset{0}{Fe}\overset{0}{Si}+\overset{+4}{Si}O_2. \quad (11)$$

The final reaction of the regenerative burning process of iron oxide (III) in the dust concentrate is as follows:

$$\overset{+3}{Fe_2}O_{3(solid)}+3\overset{+2}{C}O\uparrow \xrightarrow{T,\ 823K,\ +6e^-} 2\overset{0}{Fe}_{(liquid)}+3\overset{+4}{C}O_2\uparrow. \quad (12)$$

In addition, it should be added that the carbon ($\overset{0}{C}$) itself, as a reducing agent, is very little effective, since both iron oxide and solids are ineffective, and contact between them is ineffective. Therefore, carbon is only effective in combination with oxygen ($\overset{0}{O_2}$) in the process of combustion (with oxygen substitution). In this case, the carbon (IV,II) oxides are formed, first - $\overset{+4}{C}O_2$(IV), then - $\overset{+2}{C}O$ (II). Specifically gasified carbon monoxide (II) at high temperature (823 - 873*K*) provides rapid reduction $\overset{+3}{Fe_2}O_3$ to $\overset{0}{Fe}$ (reaction 4, 5, 6). From the point of view of the expediency of the use of the previous thermochemical reduction of iron oxides, it is attractive that the newly formed iron ($\overset{\circ}{Fe}$) - reaction product (4), (5) and (6) reacts very actively with organic acids - lactic, oxalic and citric, both in aqueous and non-aqueous media [2]. And depending on the commercial demand for processed products, it is possible, respectively, to obtain salts of appropriate organic acids - lactates, oxalates, or citrates of the iron. For example, the reaction of reduced iron with oxalate acid occurs with the displacement of hydrogen from the acid molecule and the formation of water-insoluble iron (II) oxalate:

$$\overset{0}{Fe} + 2\overset{+1}{H_2}C_2O_4 \xrightarrow{pH\leq 1,\ T,358K,-2e^-} \overset{+2}{Fe}(HC_2O_4)_2\downarrow + \overset{0}{H_2}\uparrow. \quad (13)$$

In the case when oxalic acid reacts with the presence of such mineral phases as iron (III) oxide ($Fe_2O_3\cdot 2H_2O$), hematite (αFe_2O_3) or magnetite ($FeFe_2O_4$), the reaction products are, respectively, the water-soluble iron (III) salt or the water-insoluble iron (II) salt according to the reactions (14) and (15):

$$\overset{+3}{Fe_2}O_3\cdot 2H_2O_{(\text{solid})} + 6H_2C_2O_{4(liquid)} = 2\overset{+3}{Fe}(HC_2O_4)_{3(liquid)} + 5H_2O, \quad (14)$$

$$\overset{+2}{Fe}O_{(solid)} + 2H_2C_2O_{4(liquid)} = \overset{+2}{Fe}(HC_2O_4)_{2(solid)} + H_2O. \quad (15)$$

The products of reactions (14) and (15) have a fundamental difference: iron (III) oxalate in acidic medium is water-soluble (the solution is green), iron (II) oxalate is water-insoluble salt and soluble only in strong mineral acids (HCl, HNO_3, H_2SO_4, $HClO_4$ etc.).

The appearance of iron (III) ions in the extract is confirmed by an analytical test of potassium hexacyanoferrate (yellow “bloody” salt) $K_4[\overset{+2}{Fe}(CN)_6]$. In the course of the reaction, a complex salt is formed - "Berlin blue", which forms a sediment of characteristic blue color:

$$4\overset{+3}{Fe}(HC_2O_4)_3 + 3K_4[\overset{+2}{Fe}(CN)_6] \rightarrow \overset{+3}{Fe_4}[\overset{+2}{Fe}(CN)_6]_3\downarrow + 12KHC_2O_4. \quad (16)$$

Preparation of nanosized powdered iron from products of pyrometallurgical reduction of dust concentrate

Nanosized iron particles have special properties that are different from the properties of iron, which is in the standard macrocrystallic state. It is known that the melting point, electrical conductivity, the activation energy of electron transitions, the catalytic activity depend on the size of the nanoparticles of iron. One of the main reasons for abnormal changes in the properties of a metal is the quantum-dimensional effects that occur in particles of less than 20 nm in size. Accordingly, technological techniques that allow the preparation of nanosized metal phases with a diameter of 1-20 nm are of special interest. Nanoparticles of such sizes can be used as components of magnetic carriers, X-ray contrast agents for magnetic resonance imaging, for the production of ferromagnetic liquids, and also as active components of catalysts.

In order to obtain a commercially useful product - finely divided (nanosized) iron, which is used as a catalytic additive to propellant fuel, iron oxalate (III) obtained at the previous stage of leaching with oxalic acid was used. The reaction of obtaining nanodispersed iron (0) from iron (III) ions can not occur in an aqueous medium, since hydroxyl groups of water (OH^-) will interact with iron (III) ions to form a hydroxide - $Fe(OH)_3$. Therefore, the recovery of iron (III) ions into iron (0) can occur only in a non-aqueous medium, and only if the solvent has an appropriate dielectric constant (ε)*, which contributes to the course of the corresponding oxidation-reduction reaction. In addition, to ensure the proper conditions for the reaction of the recovery of iron ions (III) to iron (0), it is necessary to transfer iron oxalate (III) to iron chloride ($FeCl_3$)

$$Fe(HC_2O_4)_3 + 3HCl \rightarrow FeCl_3 + 3H_2C_2O_4. \quad (17)$$

The oxidation-reducing reaction occurring in a medium of dimethyl sulfoxide (*DMSO*) with the course of processes for the restoration of ions of *Fe* (III) to *Fe* (0) is described by the equation

$$2\overset{+3}{Fe}\overset{-1}{Cl_3} + 2\overset{0}{Mg} \xrightarrow{T,298K;\ DMSO;\ +6e^-} 2\overset{+2}{Mg}\overset{-1}{Cl_2} + 2\overset{0}{Fe}\downarrow + \overset{0}{Cl_2}\uparrow. \quad (18)$$

In this case, the reaction takes place at room temperature (*298 K*) with the use of finely dispersed magnesium spatula ($\overset{0}{Mg}$) as a reducing agent. The choice of magnesium as a reducing agent is due to a significant advantage of the value of its standard oxidation-reduction potential ($E^\circ_{Mg^0/Mg^{2+}} = -2{,}363\ V$) in relation to a similar indicator in iron ($E^\circ_{Fe^{3+}/Fe^0} = -0{,}037\ V$). Such a potential difference contributes to the smooth flow of the oxidation-reduction reaction with the formation of a new metal phase ($\overset{\circ}{\mathrm{Fe}}$) on the surface of magnesium ($\overset{0}{Mg}$). In turn, the formation of the gas phase ($\overset{0}{Cl_2}$), which is self-removing beyond the note: * - the dielectric constant (*ε*) DMSO 46,6; the dielectric constant (*ε*)H_2O 78,6 limits of the reaction medium, allows the reaction to proceed in the irreversible direction - that is, to the right, to the end. After 5-10 minutes after the reaction began, there was an appearance of magnetic properties in the magnesium particles in which the formation of an iron siege ($\overset{\circ}{Fe}$) was observed. The latter can be removed from the reaction medium by magnetic separation. In order to optimize the kinetic indices and ensure the maximum possible yield of useful products to the reaction mixture, an excess of reducing agent is added.

Conclusions

The mechanism of formation of highly dispersed metal phases of iron, non-ferrous metals and amorphous carbon in the processes of conversion and condensation of components of blast furnace gases at high cooling rates was first times investigated and proved. For the first time, in order to effectively utilize the dust concentrate - one of the main waste products of the metallurgical industries, a technology for its processing is proposed, which includes regenerative thermal treatment (regenerative burning) at relatively low temperatures (773-823 K), followed by leaching of the residue formed after such treatment, with oxalic acid. After successive transformations of the salt of iron (III) oxalate in the salt of iron (III) chloride, the latter is dissolved in a non-aqueous solvent - dimethyl sulfoxide, in which an oxidation-reduction reaction is performed, the final product of which is iron (0). As a reducing agent, fine-grained magnesium ($\overset{\circ}{Mg}$) is used. At the same time, the yield of the target metal (in the first place, iron) is doubled. Nanodispersed iron (0) can be used as an active catalytic addition to the fuel of jet (rocket) engines, or as a raw material for the manufacture of magnetically sensitive elements.

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