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Control of Rock Mechanics in Underground Ore Mining

V I Golik[1,2, a], A B Efremenkov[3, b]

[1]North-Caucasian State University of Technology, 44 Nikolaev street, Vladikavkaz, 362021, Russia[1]

[2]Vladikavkaz Scientific Center of the Russian Academy of Sciences and the Government Republic of North Ossetia-Alania, 22, Markusa street, Vladikavkaz, 362027, Russia

[3]Yurga Institute of Technology, TPU Affiliate, 26, Leningradskaya street, Yurga, 652055, Russia

E-mail: [a] v.i.golik@mail.ru, [b] abe73@rambler.ru

Abstract. Performance indicators in underground mining of thick iron fields can be insufficient since geo-mechanic specifics of ore-hosting fields might be considered inadequately, as a consequence, critical deformations and even earth's surface destruction are possible, lowering the indicators of full subsurface use, this way. The reason for it is the available approach to estimating the performance of mining according to ore excavation costs, without assessing losses of valuable components and damage to the environment. The experimental approach to the problem is based on a combination of methods to justify technical capability and performance of mining technology improvement with regard to geo-mechanical factors. The main idea of decisions to be taken is turning geo-materials into the condition of triaxial compression via developing the support constructions of blocked up structural rock block. The study was carried out according to an integrated approach based on the analysis of concepts, field observations, and simulation with the photo-elastic materials in conditions of North Caucasus deposits. A database containing information on the deposit can be developed with the help of industrial experiments and performance indicators of the field can be also improved using the ability of ore-hosting fields to develop support constructions, keeping the geo-mechanical stability of the system at lower cost, avoiding ore contamination at the processing stage. The proposed model is a specific one because an adjustment coefficient of natural and anthropogenic stresses is used and can be adopted for local conditions. The relation of natural to anthropogenic factors can make more precise the standards of developed, prepared and ready to excavation ore reserves relying on computational methods. It is possible to minimize critical stresses and corresponding deformations due to dividing the ore field into sectors safe from the standpoint of geo-mechanics, and using less cost-demanding ways of keeping rock massif stable.

1. Introduction

Ore-hosting deposits, in most cases are rocks and half-rocks with higher acoustic impedance, localized in the developed residuum. Thickness of the rock area with their strength falling around small workings is about first meters, and it far more on the chamber contacts. Inside these areas there is a border zone of weakness.

Excavation of mineral resources leads to the development of technology-related cavities, which are the reasons for arising stresses and deformations in ore-hosting areas, and can cause even earth's crust destruction over them. In underground mining of thick ore fields the performance indexes are insufficient because of inadequate consideration of geo-mechanic specifics of ore-hosting fields, as a consequence, critical deformations and even earth's surface destruction are possible, lowering the indicators of full subsurface use this way [1]. The reason for it is the available approach to estimating the performance of mining according to ore excavation costs, without assessing losses and damage to the environment when nature destructing technologies are of top importance [2].

The integrity of the earth's surface can be provided by means of residual supporting capacity of fractured rocks. At present, the concepts on strengthening mechanism of rocks can't provide full understanding the conditions how to be involved in operation of the ore-hosting field, limiting, this way, the sphere of applying nature-saving underground mining technologies. Therefore, establishment of correlation between the value of mountain pressure and residual supporting capacity of rocks, as well as development of methods to control rock masses when ore extracting and cavities filling, which are aimed at obtaining sufficient indexes of using the Earth's interior are scientifically and practically important issues.

The approaches to solution of geo-mechanical problems are distinguished by combination of methods for justifying technological capacity and efficiency of improving the technology on the base of geo-mechanical factors. The main idea of decisions to be taken is ability of geo-materials to develop the support constructions of blocked up structural rock block. To provide correctness of outcomes the study was carried out according to an integrated approach involving the analysis of concepts, field observations, and simulation with photo-elastic materials in conditions of particular deposits.

2. Materials and Methods

The theory by Protodyakov M.M. (1933) is the most appropriate one for control of ore-hosting field state, in terms of this theory a working is subject to rock weights only, which are within the natural self-supporting arch at the height far smaller than the mining depth.

This theory was further developed in papers of Russian and foreign geo-mechanics. V.D. Slesarev (1948) found out a parameter of stability, i.e. resistance to failure of rocks forming a truss. A.A. Borisov (1964) joined the theory of arch with stability of a rock layer in the door head. S.V. Vetrov (1975) determined a stable position of working as an equality of blocked up rock strength, forming a hinged arch with a weight not exceeding that of natural self-supporting arch [3].

Maximal allowable outcropping spaces in ore-hosting field:

$$L_F \le L_P; h \le H; k \ge 1$$

$$L_P = f(R_1, d_{1.2}, k_2, в, y) \tag{1}$$

where L_F - maximal actual outcropping space of a cavity roof, m; L_P - limiting arch span, m; h – height of mining affected zone, m; H – depth of mining measured from cavity edges to the bedrock level, m; k – coefficient of safety margin, units; R_1 – rock strength (man-made massif), MPa; $d_{1,2}$ – horizontal and vertical sizes of structural blocks, m; k_2 – safety factor, units; $в$ – rock thickness providing no reliable roof constructions, m; γ - rock density, t/m^3.

The stability of ore-hosting field is ensured in conditions of sufficient mechanical strength of the bottom row of blocked up structural blocks loaded by rock weight within the natural self-supporting arch (Fig. 1).

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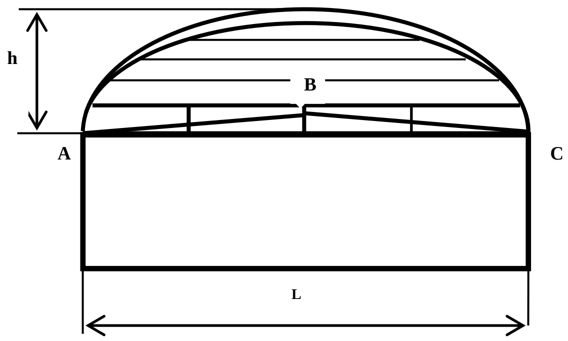

Fig. 1. Operational scheme of three-hinged arch of blocked up rocks: A, B, C – hinges: L – limiting space of level outcropping in the door head; h – height of the natural self-supporting arch

The roof keeps the level form if stresses in the supporting layer do not exceed the ultimate strength of rocks in the contact zone of two rock blocks (Fig. 2).

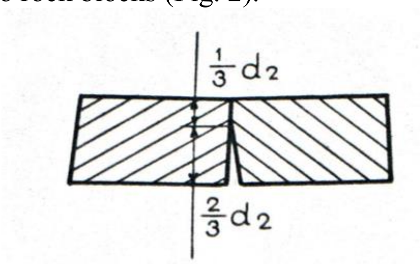

Fig. 2. The condition of a mine working level roof: d_2 – vertical size of the sructuraal rock block

Preservation of the Earth's surface from destruction over the developed mining workings of deposits is provided via controlling the level of natural and technology-related stresses by combining technological processes in time and space. New nature and resource saving technologies and technical means were developed on this base and positive outcomes of their application in underground mining were registered in Russia, Kazakhstan and Ukraine.

Nature and resource saving requirements at mining enterprises consist mainly in the use of low quality mineral resources and wastes in production. Their insufficient activity is improved through recycling. Therefore, optimization of the mining technology on the base of geo-mechanical factors, conversion of geo-materials into the triaxial compression state is an important economic and social problem.

Efficiency of a mining technology is determined via comparing some mining variants, taking into account costs at all processing stages and protection of inhabitants living in the mining-affected zone according to the criterion of Earth's surface safety.

3. Experiment

A promising direction in mining is chamber mining with filling of the mined-out space with hardening stowing mixtures of improved composition and strength.

If the stages in preparing recourses are combined rationally in time and space, a mining company works uninterruptedly and efficiently. The condition of recourses prepared for excavation over those ready for excavation depends on conditions and routine of development, cutting, acquisition of ore recourses in the exploitation block, way of keeping and depreciation of the mined out space.

The blocks are prepared gradually for excavating, that slows down the rates of mining because production capacity of a mining company depends on the volume of recourses ready for excavation.

The division of the ore field into safe zones according to geo-mechanical principles includes into the volume of ready for excavation recourses zones of the field protected from critical deformations by natural constructions of blocked up structural blocks. A group of blocks within such a zone makes it possible to prepare recourses for excavation independently, improving intensity, this way.

Safety of ready for excavation recourses is determined by the possibility of building a construction of blocked up rocks, which keep a flat door head:

$$2a = 2d_1\left(\frac{10R_2}{k_1 yH} - 1\right) \tag{2}$$

where d_1- horizontal dimension of a structural rock block, m; R_2 – ultimate strength of rocks, kg/cm^2; H – depth of mining, m; γ - volume weight of rocks, t/m^3; k_1 – safety margin.

At d_1 = 3 m, R_2 = 1400 kg/cm^2; H = 500 m, γ = 2.7 t/m^3 and k_1 = 1, the ultimate width of arch is 54 m.

The equivalent width (L_f) of an ore body with strike *A=500* and dip *B=640 m* is (dimensions are in the plan):

$$L_f = \frac{AB}{\sqrt{A^2 + B^2}} = \frac{500x640}{\sqrt{500^2 + 640^2}} = 394m. \tag{3}$$

The equivalent width exceeds the maximum permissible one; therefore, the ore field is divided into zones 54×54 m via advancing of rooms and filling them with hardening mixtures, the strength of which is calculated according to surcharging with rock mass to the surface level. For example, there are 4 blocks on the area 54×54 m with rated recourses ready for excavation via chamber mining with hardening filling for 12 months. The norm of ready for excavation resources should be 6-12 months.

The most difficult way of mining is mining of thick fields, opening onto the earth surface, a typical example is Tyrnyaus tungsten and molybdenum deposit in the zone of metamorphically altered rocks of upper and middle Paleozoic era and Lower Jurassic, cut with dikes, stocks and sheeted bodies different in age and composition. The ore zone is located among igneous rocks: crystalline schists, pegmatites and granite-gneisses. More than 60% of ore reserves are in the Main Skarn.

The ore body is a saddle-shaped basin with wings falling north and south at an angle of 60-70^0 and eastwards immersing hinge at an angle of 55-60^0 under the thickness of biotite hornblendes. The thickness of the deposit in the inflection is 100-120 m, on the sides it is up to 3-5 m (the average one is 50 m).

The strength by M.M. Protodjakonov and density, respectively, for skarn ores are 16-20 and 3 t/m^3, these of rocks of the hanging side (hornfels) - 12-14 and 2.6 t/m^3, the rocks of the recumbent side (marbles) - 8-12 and 2.5 t/m^3. Ore doesn't tend to oxidation, caking and spontaneous combustion. The degree of ore fragmentation is 1.5.

The main reserves are excavated in induced block caving with blasting holes. The reserves of the main skarn are mined in induced block caving with ore breaking by blasting holes in a clamped medium in one-stage order.

The overlapping rocks break down over the mined blocks on the main skarn and a single crater is formed. There are moving rocks in the pitcrater when breaking arch pillars and blocks.

At a shallow depth of mining the crater has steep edges. As the depth of mining increases, breakage of the hanging side in the thick part of the main skarn becomes smooth. The edges of the crater on the hanging side become flat, and the transverse dimensions of terraces and cracks zone increase. At a depth of 350 m below the surface at some distance from the main pitcrater another crater is formed. Craters above the blocks in the thick part of the main skarn are developed at a speed of 20 to 50 mm/s, having the shape of tubular hollows without characteristic displacement lines.

The complex of studies on geo-mechanical phenomena included instrumental observations and simulation of the process using methods of photo-mechanics and mathematical theory of elasticity. The quantitative estimation of the displacement parameters was made as a result of reference-type station data processing. The method of observation included recording crater dimensions on the earth's surface with registering the dynamics of their development and construction of hazardous zones [4].

The shift in the North-West Skarn was slow. The largest registered subsidence was 149 mm. Horizontal deformations ranged from 8-9 to 3.0 × 10-3. After 1981 the process of rock shifting slowed down. In 1989 the ceiling of block 4 was broken horizontally 2537 m, and the movement of rocks intensified. The largest subsidence was 102 mm. Horizontal deformations in the reference interval are 35-36, in the zone of tectonic disturbance E = 8.3.10-3. The break angle was 76^0.

In 1991 the process of rock shifting became even more active and resulted in opening of rupture cracks on the earth's surface. The largest subsidence (600 mm) was recorded by bench 7; its subsidence rate was 15 mm/s. Over one year 13 craters were formed due to mining with frontal discharge. The standing time of rocks is 10 to 128 days. The volume of ore output 10,000 to 100,000 thousand tons.

The craters appear together with subsidence of the earth's surface at a speed of about 20 mm / s. and occurrence of cracks in the form of closed contours. A clear correlation between time of ore excavation and depth of mining operations was not observed. The craters were formed at a depth of mining from 80 to 550 m. The minimum time of crater formation is 5-10 days, the maximum - 3 years.

The criterion of field state is the ratio of output volume V_p and craters V_f variants are possible:

$$\frac{V_p}{V_f} > 1, \frac{V_p}{V_f} \approx 1, \frac{V_p}{V_f} < 1 \tag{4}$$

In the first case loosened rocks are carried from the release ellipsoid area, cavities are accumulated, and field is subject to intense destruction. As ore is released, a mass of overlying loose rocks starts moving, the earth's surface moves smoothly and slowly, and a shallow crater appears (1.5 m).

In the second case the crater depth increases; the earth's surface breaks rapidly, the period of crater formation is reduced. The peculiarity of rocks breaking process over the mined space is its limited area. As the rock mass is mined out, the rocks and the earth's surface descend. The time of caving rarely exceeds 2-3 weeks. The average depth of craters does not exceed 20 m.

In the recumbent side field of the Main Skarn there were no shifts registered. The average displacement speed of the rocks was 5.7 mm / month in 1991 on its northwestern side, whereas a critical speed was 40 mm/month. The maximum height in the northwestern side of Mukulansky open cast was more than 700 m with a total inclination angle of 26-27^0. The height of the ledges is 50-55 m with a slide angle of 50^0 and about 70 m with a slide angle of 40^0.

The breaking zone is a loose medium, characterized by lacking adhesion and weakening surfaces of tectonic cracks type. The calculated angle of edge inclination in it can be equal to the natural slope angle, being conditionally equal to the angle of internal rocks friction of 37-38^0 with an unlimited height. The rock displacement process being a consequence of natural structure leads to complete destruction of connection between the structural blocks of rocks and subsidence of large sections with slight disruption of the earth's surface. The zone of rupture influence is determined according to the magnitude of the earth's surface critical deformation - horizontal expansion, and also to the slope and curvature of discontinuous structures arising in underground mining.

The blind deposit was mined out via underground chamber-integral system with dry filling. Chambers 95 m high with dimensions of 30x30 m were separated in plan by inter-chamber pillars of the same dimensions. Thus, chambers 6-7 are located under the bottom of the northern side at a depth of 120 m [5].

The hanging side of the blind deposit, where opencast benches are located, is composed of biotite hornfels (structural blocks are 0.5-1.5 s). Structural blocks are formed, as a rule, as three systems of cracks with a strike for the first system of 50-60^0, an angle of incidence of 55-60^0; for the second system - 100-110^0 and 60-70^0, and for the third system - 170-180^0, 25-30^0.

In chamber 7 the roof was broken and the crater arose on the surface of the opencast. When ore mining the crater was expanded, reaching a width of 50 m along the strike and 30 m across the strike. The rock sedimentation rate was 2.7 m/s. After crater formation the surface over the chambers sagged by 5.5 m, and on the mountains 2435 m the settling rate gradually decreased to 2 mm/s.

In case of subsidence crater formation in underground mining workings the safety factor is reduced by up to 30%. The obtained analytical dependencies are verified on models using the methods of photo-mechanics and mathematical theory of elasticity [6].

The models were made of optically active polyurethane with the value of the stripe $\sigma_H^{1,0} = 75.6$ kg/cm^2 for the conditions: $H = 350\,m$ - depth of the workings from the day surface; γ = 3.0 t/m^3 – density of the overlapping rocks. To obtain the normal stress on the free edge of the working number of the band order is multiplied by the band value ($\sigma_H^{1,0}$).

Strength condition of the edge:

$$\sigma_1 - \sigma_2 \geq \sin\delta.(\sigma_1 + \sigma_2) + \sigma_3.(1 - \sin\delta), \tag{5}$$

where $\sigma_1(\sigma_2)$ – value of stresses in the edge point; δ =30^0 – inner friction angle; σ_3 = 140-160 MPa.

The models were stressed with lateral extension coefficients 0.5; 1.0; 1.5. The values of the lateral extension are specified according to the features of Tyrnyauz deposit located in the activity zone of the Elbrus mountain system, it redistributes lateral pressure when height vs. width of the chamber > 1.

The problem of estimating the stability of an inter-chamber pillar with decreasing width from 30 to 20 m was solved by modeling. Variants of the chamber section shape were examined to improve or reduce stresses along the edges and in the locks without changing roof dimensions and shape and keeping the same chamber dimensions. The roof was changed from wedge-shaped into elliptical.

The aim was to obtain stress extremes in the underground excavation of chambers and when approaching workings of the opencast. If a chamber is filled dry with the size of pieces up to 200 mm, it has an about 25 m high (30% of the chamber height) cavity after filling shrinkage, the prospect of a dry filling was estimated when combining underground and open methods of deposit mining.

The results of simulation are as follows.

Projection on a vertical plane:

1. For a lateral distance factor λ =0.5, the maximum stresses in lock zones of roof and chambers walls are: 75.6x7.5 = 567 kgs/cm^2. At the top of the roof arch the stress is 75.6x2.0 = 151.2 kgs/cm^2. In the inter-chamber pillar the maximum compressive stresses decrease to 75.6x6.5 = 491.4 kgs/cm^2.

2. For a lateral distance factor λ =1.0: the maximum stresses in lock zones of roof and chambers walls are: 75.6x6.5=491.4 kgs/cm^2; being the same at the top of the roof arch; in the inter-chamber pillar the maximum stresses decrease to 75.6x5.5=415.8 kgs/cm^2.

3. For a lateral distance factor $\lambda = 1.5$: the maximum stresses in lock zones of roof and chambers walls are 75.6x6.5 = 491 kgs/cm^2, at the top of the roof arch compressive stresses increase up to 75.6x 8.5 = 642.6 kgs/cm^2.

Vertical section of the chamber:

The zones of maximum stresses are roof arch and bottom trench.

For coefficient of lateral extension - 0.5; 1.0; 1.5 the stress is 415.98; 1020.6; 1400 kgs / cm^2, respectively.

With regard to simulation results it was concluded that configuration and orientation of cavities affect the stability of combined excavations in the zone. A high concentration of stresses is revealed in the zone of the bottom trench entry provided that it has a wedge shape.

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The spherical arch provides approximately twofold reduction in stress. The 30 m wide inter-chamber pillar provides stability, when decreasing width of the inter-chamber pillar to 20 m its stability drops. The maximum concentration of stresses is registered in the zone of the trench entry.

The problem of inter-chamber pillar mining optimization was solved with the help of modeling. Modeling conditions:

- lateral extension is 0.5H; 1.0H; 1.5H; 2.0H;
- inclination angle of the main force vector to the vertical axis $\alpha = 0^0; 30^0; 40^0; 50^0$ respectively for each lateral extension;
- Variants without filling both chambers and with filling of one chamber. The filling module is E = 0.1 MPa, that of the field is E = 1.4 MPa, 14 times higher than filling module.

Stress in any point of the model:

$$\sigma_m = \sigma^{1,0} n^0 \tag{6}$$

where $\sigma^{1,0}$ =0.1 kgs/cm^2 for one stripe; n^0 - stripe number in the model point.

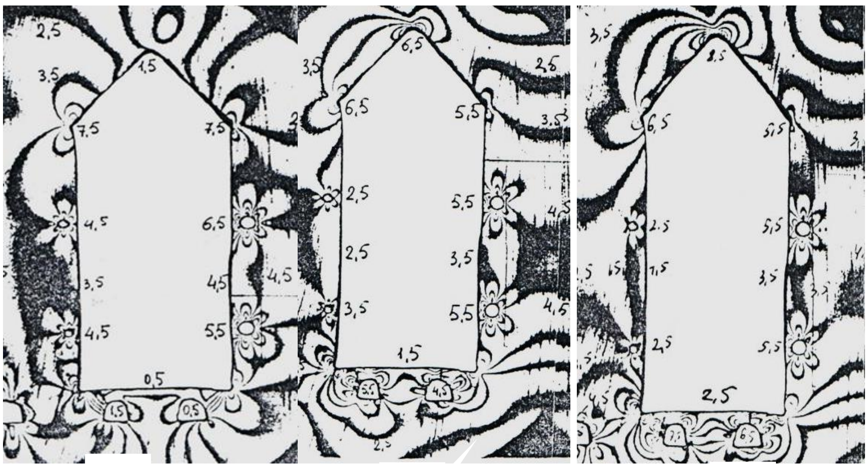
Natural stress:

$$\sigma_H = \gamma H \frac{\sigma_m}{\sum}, \tag{7}$$

where $\gamma = 3.0$ t/m^3 – density of ore and rocks; H = 30 kgs/cm^2 -depth of occurrence from the day surface.

Analyzing the simulation results it can be concluded (Fig. 3):

- chamber roofs and bottoms are most stressed;
- arch-formed roof reduces the stress by 25%;
- filling of one chamber reduces the stress in the roof top by 2 times;
- in the inter-chamber pillar increased stresses appear.



1 2 3

Fig. 3. Stresses in the elements of the open chamber vs. horizontal stress: 1-0.5; 2- 1.0; 3-1.5.

4. Results and discussion

Open-underground mining of deposits is difficult to control in terms of geo-mechanical field stability. Dry cavity filling has no effect on the situation [7-8].

Under equal conditions the reaction of field to technological impact depends on magnitude and sign of stresses in the field. Field displacement is a consequence of internal deformation and destruction of elementary mineral particles, complicated by medium anisotropy [9-11].

The internal stress field of structural separate parts of the deposit depends on the external stress field:

$$\sigma_{b.} = [\ \sigma_{Ms}\, f\, \Sigma(\ 1...n\ )\ T]\ k_{gn} \tag{8}$$

where σ_b is the stress field of rocks; σ_{ms} – massif stress field; T – tensors of elastic inhomogeneity moduli: random, constant, etc.; k_{gn} – coefficient of geological disturbance.

In volumetric compression conditions in massifs, tectonic structures close, strength and modulus of deformation increase, as a result, the volumetric stress state transformations appear; stretching and shear on the cavity edge to compression at the depth of the mass. In the short-term destruction process in the out-of-the-way stage rheological processes take place in the out-of the limit area.

Depending on rocks deformation of in the massif, there are several destruction areas simultaneously [12-13]. Near the rupture there is a zone of broken rocks with a minimal supporting capacity. It is followed by weakened rocks of transcendent deformation, the strength increases as far from the rupture. Finally, there is a zone of untouched rocks in the stage of pre-limit deformation. Maximal preservation of the massif is possible via minimizing the first two zones and increasing pre-limit deformation, where strength is determined in terms of time:

$$\sigma ci\ (t) = kt \times [\sigma cl + (\sigma ci - \sigma cl\ )]\ e - at \tag{9}$$

where $\sigma_{ci.}$ – limit of instantaneous strength at one-axial compression, MPa; σ_{cl} – limit of the long-termed strength at one-axial compression, MPa; K_t – tectonic faulting coefficient; a – approximation parameter; t - time.

Condition of massif stabilty:

$$\Sigma\ (\ \sigma_3 + K\ \sigma_2\ \sigma_1\ ) \le \Sigma\ \sigma_o\ K \le \Sigma\ \sigma_H\ K \tag{10}$$

$$\Sigma\ \sigma_H\ \ K = f\ (\ \sigma_r\ ,\ h_c,\ P_{min}\ ,\ P_{max}\ ) \tag{11}$$

$$\Sigma\ \sigma_o\ K\ K_{pH} = f\ (\ \sigma_H\ ,\ H\ ,\ B\ ) \tag{12}$$

where $\sigma_{3,2,1}$ – main stresses in the massif; σ_H , σ_o – stresses in the zones of destructed and weakened rocks, respectively, MPa; h_c – height of the rupture affecting zone, m; P_{max} and P_{min} – maximal and minimal technological impact; σ_r – residual strength of rocks at one-axial compression, MPa; H and B – height and width of technologically affecting zone, m; K - factor of tectonic faulting impact; K_{pH} – factor of stress loading out.

The control of massif state is in meeting the following conditions:

$$[\sigma_{HM}] < \sigma^R < [\sigma^R]_C \tag{13}$$

where $[\sigma^R]_C$ - critical stresses in the destructed rocks.

Zone height of critical stresses propagation:

$$H = f[\sigma]_C^{\ R} > h_T = f[\sigma]_T \tag{14}$$

where hT – zone height of technology-related impact, m;

$$h_T = \int_{min}^{max} fV \int_{min}^{max} fS \int_{min}^{max} fm \tag{15}$$

where H' – zone height of deformation to critical stresses, m; S_p - area of weakening, m^2; α_H - dip angle, degree; m_n – normal thickness, m.

Geo-mechanical processes of rocks destruction and technological indexes are expressed in the condition [14-15]:

$$\sigma \cdot K_3 = \int_{l\min}^{l\max} fx\left(d_{x1}, d_{x2} \ldots . d_{xn}\right) \rightarrow P, R = \int_{l\min}^{l\max} f \cdot x\left(d_{hZ} + dh_p\right) \qquad (16)$$

where σ - stresses in the mining-affected zone, MPa; K_3 – coefficient of stress correction; l_{max}, l_{min} – outcropping spaces of rocks, m; $x_1 \ldots x_n$ – technological, physical and mechanical and other characteristics; P – ore losses, share units; R – ore contamination, share units; $h_з$ – height if the filled massif, m; h_P – height of mining effect, m.

The proposed model is special because of coefficient of stress correction K_3. In filling cavities with hardening mixtures the level of stresses is reduced due to characteristics of filling hardening material (Fig. 4).

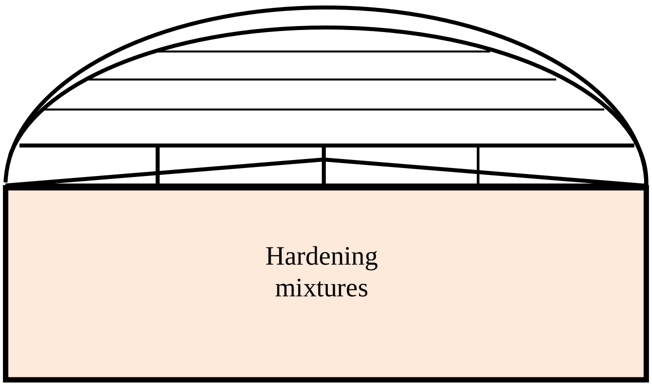


Fig. 4. The layout of roof support with hardening mixtures to support the ore-hosting layer

Work performance depends on rock mass vs. side support of the artificial massif walls. In conditions of volumetric compression, filling strength increases 1.5-3 times. The state of massifs, losses and contamination are determined by the volume of cavities, the volume of mined out ore and strength of rocks.

As for S.V. Vetrov massif stability is provided according to the condition:

$$L_P > 2\alpha = d_1\left(\frac{R_c}{KH\gamma} - 1\right), \qquad (17)$$

where a - is a half-space of the blocked up rocks, m; d_1 - horizontal size of the structural block of rocks, m; R_c - rocks resistance of the structural block to compression, t/m^2; γ - volume weight of rocks, kg/m^3; H_Y- depth of the blocked rock arch shoe, m; K - safety factor.

Filling of cavities with hardening mixtures stabilizes the massif state; the effect is possible due to reducing losses and contamination. The possibility of stress control is provided by limiting deformations in filling cavities with hardening mixtures:

$$\sigma_1 \leq \sigma_2 \leq \sigma_3 = \sigma_H \cdot K_1 K_2 K_3 K_4 \tag{18}$$

where σ_1 – stresses in the zone of undamaged rocks, MPa; σ_2 - stresses in the clean-up operation zone, MPa; σ_3 - stresses in the filled massif, MPa; σ_H - rated resistance to compression of filling, MPa; K_1 - coefficient of filled massif inhomogeneity; K_2 - coefficient of increasing strength upon time; K_3 - coefficient of increasing filling strength in the massif; K_4 - of work conditions.

Stresses after filling the cavities:

$$\sigma_{..} = n_1\sigma_{n..3} + n_2\sigma_{c..3} + n_3\sigma_{m..3} + n_4\sigma_{\mathrm{H}..3} + n_5\sigma^R = \sum_1^{\ell} n_\ell \sigma_m^y \tag{19}$$

where $\sigma_{n.3}, \sigma_{c.3}, \sigma_{m.3}, \sigma_{H..3}$ - support values of high-, medium- and low-strength hardening mixtures; ℓ - number of hardening elements; $n_1, \ldots, n_5$ - mass number of material in the amount of mixture; σ_m^y - strength of mixtures.

Model of controlling the massif state via stress uploading:

$$\sigma_1 \leq \sigma_2 \leq \sigma_3 = \sigma_H \cdot K_1 K_2 K_3 K_4 \tag{20}$$

$$PY_T = C_T - Z_T - Y_{T_{\alpha T}} - Y_{T_{RS}} - Y_{T_{RP}} \tag{21}$$

$$\mathrm{B}_{\mathrm{a}} = \mathrm{B}_{\mathrm{c}}\mathrm{E}_{\mathrm{B}}, \tag{22}$$

where σ_1 – stresses in the zone of undamaged rocks, MPa; σ_2 - stresses in the clean-up operation zone, MPa; σ_3 - stresses in the filled massif, MPa; σ_H - rated resistance to compression of filling, MPa; K_1 - coefficient of filled massif inhomogeneity; K_2 - coefficient of increasing strength upon time; K_3 - coefficient of increasing filling strength in the massif; K_4 - of work conditions; PY_T - profit from ore excavation and processing, ruble/t; C_T value of the mined out ore, ruble/t; Z_T full costs of ore mining and processing, ruble/t; $Y_{T_{RS}}$ – loss caused by 1t of contaminating mass along the block edge, ruble/t; $Y_{T_{RP}}$ – loss caused by processing of 1 t of contaminating mass inside the block, ruble/t.; B_{a} – number of alternative binding agents, weight units; B_{c} – number of standard binding agents, weight units; E_{B} – coefficient of binder equivalence.

The possibility of control stresses makes it possible to use low-strength compositions of hardening mixtures based on recycled ore dressing wastes after activation, e.g. mechanic and chemical technology for filling the mined out zones of the massif [16-17].

5. Conclusions

Performance indicators in underground mining of multi-structural fields can be improved via using possibilities of ore-hosting fields to develop supporting constructions, which keep the balance of the system “surface – massifs – environment”.

The relation of natural to anthropogenic factors can make more precise the standards of developed, prepared and ready to excavation ore reserves relying on computational methods. Economic efficiency can be improved via better control of mining operations.

Development of a database containing information on the deposit and optimization of performance indexes with the help of geo-mechanical recourses of ore-hosting fields provide a balance of geo-mechanical system with low economic expenses and minimal losses and contamination of ores at the stage of mining processing. It is possible to minimize critical stresses and corresponding deformations due to dividing the ore field into sectors safe from the standpoint of geo-mechanics, and using less cost-demanding ways of keeping rock massif stable, as well as to specify norms of mined out, prepared and ready for excavation ores.

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