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GEOMECHANICAL SUBSTANTIATION OF THE PARAMETERS OF A CONTINUOUS CHAMBER SYSTEM OF MINING WITH CAVING OF THE ROOF ROCK

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INTRODUCTION

The study which led to the results presented in this article was conducted within the framework of the program "Devising Efficient Alternative Systems for Mining Ore Deposits at Substantial Depths." The problem of controlling rock pressure during the mining of ores at significant depths is usually solved by filling the excavated space. Excavating deposits with a filling allows a high degree of extraction of the mineral from deep under ground, permits minimal deformation of the pillars and leaves the surface undisturbed, and helps reduce mine wastes. However, in the mining of minerals of moderate or low value, working the deposit or sections of it with the use of a filling is not always economically justifiable. Instead, high-productivity, low-cost stoping methods must be used.

An example of a deposit requiring this approach is the Nikolaevsk bed located in Vostochnii Primor'e. The mine-engineering situation at the deposit is characterized by a severe stress-strain state in the rock mass being worked and inadequate stability of large exposures of the enclosing rock — which precludes the use of a mining system with open stopes. A continuous chamber system employing a chamber—pillar scheme and controlled caving of the roof is being tried out on a experimental section of the Nikolaevsk mine. The goal of the experiments at this stage is to evaluate the stress-strain state of the rock mass in structural elements of the proposed development system.

MINING OPERATIONS AT THE NIKOLAEVSK MINE

The Nikolaevsk mine is extracting complex ores containing lead, zinc, silver, bismuth, and cadmium. The Nikolaevsk deposit is a series of large horizontal beds with highly complex shapes and distinct meandering contacts. The main ore body, "Vostok-1," extends from a depth of 700 m to 1100 m, its width ranges from 4-6 to 60-70 m, and its width in the central part ranges up to 600 m. Another ore body that is about 50 m thick and is estimated to extend to a depth of more than 2000 m was detected below "Vostok-1" by means of boreholes.

The deposit is characterized by a highly nonuniform distribution of the metals in the ore bodies, which contain coarse inclusions of country rock (limestones). The mineralization coefficient for the individual sections (blocks, rooms) ranges from 0.55 to 1.0.

The deposit was originally worked with a system of sublevel drifts and the use of the room-and-pillar method. A congealing filling was used for the primary chambers and a dry (or hydraulic) filling for the secondary chambers. Ten years' experience with this technology showed that it has several problems [1]: the pillars between the chambers gradually crumble after the chambers have been mined and take on an "hourglass" shape, with their width in the midsection decreasing from 20 to 12-15 m; the two-stage stoping requires a corresponding tightening of standards on the reserves that are prepared and ready for excavation; caving of the filling during excavation of the secondary chambers ultimately reduces the yield of metals in the form of concentrate by 2-5%.

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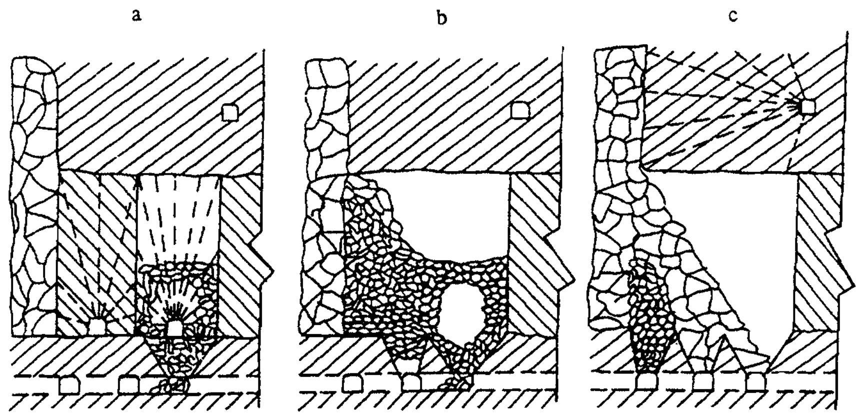


Fig. 1. Successive stages in a continuous chamber system of mining with controlled caving of the roof rock: excavation of chamber reserves (a), removal of ore from the exploded pillar (b), and caving of the rock overhang (c).

Studies conducted by the Mining Institute (of the Siberian branch of the Russian Academy of Sciences) proved the efficacy of using a layered mining system with a congealing filler for much of the deposit. By leaving the rock inclusions in place, this technology makes it possible to reduce ore depletion by a factor of 1.2-2.0 and increase the efficiency of preparatory-cutting operations by a factor of 1.7-2.5.

Also, economic calculations show the efficiency of using more productive mining methods when working sections with low-grade ores, especially on the flanks of deposits. Such methods are best used when the ore-content coefficient is 0.9 or higher.

MINING SYSTEM

Figure 1 illustrates the proposed mining system, involving controlled caving of the overlying rock. The deposit is broken up into blocks that include a chamber and a pillar. The reserves in the chamber are excavated first, with the chipped-out ore taken to the surface through openings in the seam. The haulage stages are located at the ends of the pillars. The pillars are blown up after excavation of the chamber and the ore is removed under a protective rock overhang. The roof rock is caved to a specified height after the ore has been removed from the block. This ends the block stoping cycle, and work proceeds to the next block.

A prerequisite to using this system is that ore be cut and removed only in one block on each flank (stoping front).

GEOMECHANICAL SUBSTANTIATION OF THE PARAMETERS OF THE TECHNOLOGY

Formulation of the Problem and Initial Data. An ore body of thickness m is located at the depth H = 900 m (a bed in the test section, Fig. 2). The working region includes the goaf with roof rock caved to the height h, a room of width b_c, and a pillar of width b_p located between the room and the goaf. The span of the working is L (due to symmetry, we will examine half of the complete region of the solution). Calculations were performed by the finite-element method using an elastic model with plane strain for the complementary stress-strain state of the rock mass. There were 1050 nodes in the theoretical region. The system of linear equations in nodal displacements was solved by the iterative method of successive upper relaxation. Table 1 shows the geometric parameters of the calculated variants.

In solving the problem, we assumed that the components of the initial stress state of the rock $\sigma_{ij}{}^0$ (before the beginning of mining) is a superposition of the stresses due to the force of gravity and stresses that are independent of the depth of the horizontal tectonic stresses $T_x{}^0$, $T_z{}^0$:

$$\sigma_y^0 = \gamma y, \quad \sigma_x^0 = T_x^0 + \lambda\sigma_y^0,$$

$$\sigma_z^0 = T_z^0 + \lambda\sigma_y^0, \quad \tau_{xy}^0 = \tau_{xz}^0 = \tau_{yz}^0 = 0.$$

TABLE 1

Variant	L	b_p	b_c	h	m
	meters				
1	200	20	20	0	40
2	200	20	20	20	40
3	200	20	20	40	40
4	200	20	20	60	40
5	200	16	24	40	40
6	200	12	28	40	40
7	200	0	40	40	40
8	160	20	20	40	40
9	160	16	24	40	40
10	160	12	28	40	40
11	160	0	40	40	40
12	120	20	20	40	40
13	120	16	24	40	40
14	120	12	28	40	40
15	120	0	40	40	40
16	200	20	30	40	40
17	200	0	50	40	40
18	200	20	20	40	20
19	200	20	20	40	10

TABLE 2

Rock	c_0, MPa	φ, deg
Limestone	7,1	50
Skarn	12,2	58

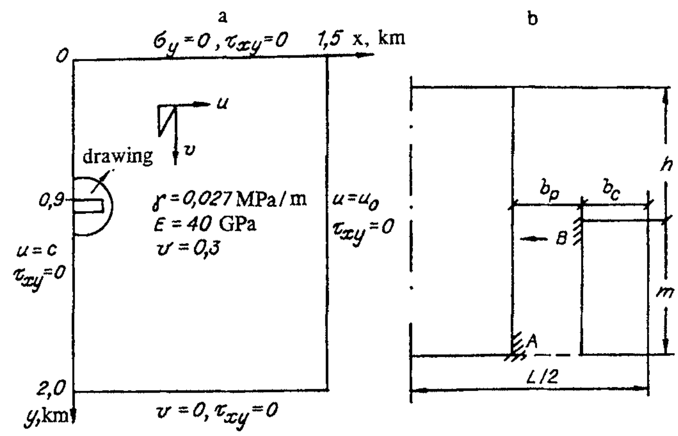


Fig. 2. Diagram of the theoretical region (a) and the main parameters of the system (b).

Here, $\lambda = \nu/(1-\nu)$ is the coefficient of lateral thrust; ν is a coefficient expressing the transverse strains in the rock. The z axis is oriented perpendicular to the xy plane and the compressive stresses have a positive sign.

The level of the tectonic stresses T_x^0 is determined by suitable selection of the horizontal component of the displacement u_0 on the right boundary of the theoretical region (see Fig. 2). This allows us to assign an arbitrary ratio of the components of the initial stress field at a certain fixed depth y = H:

$$\sigma_x^0 : \sigma_y^0 : \sigma_z^0 = \lambda_x : 1 : \lambda_z. \tag{1}$$

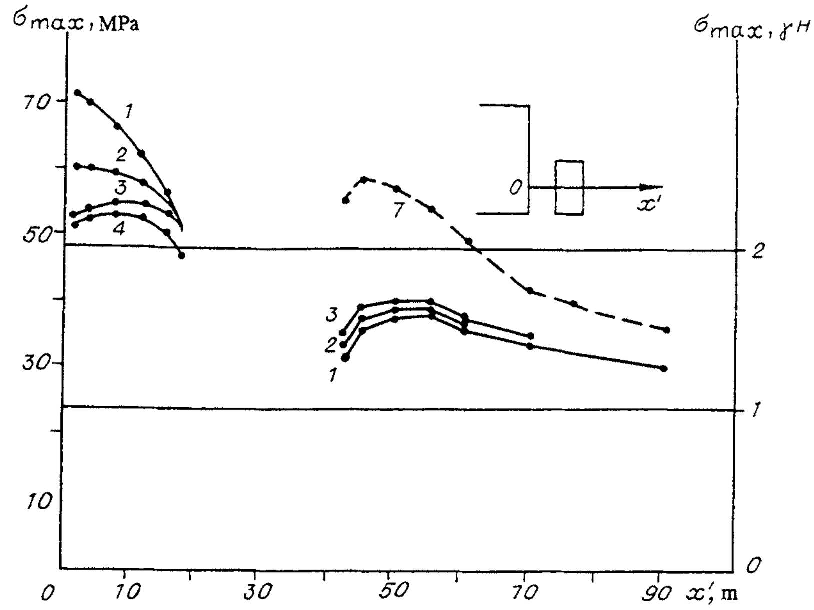


Fig. 3. Curves of bearing pressure along the line Ox′ for different computational variants.

Since the problem is being solved in a two-dimensional formulation (plane-strain condition $\varepsilon_z = 0$), the following relation is valid:

$$\sigma_z = \nu(\sigma_x + \sigma_y) + [\lambda_z - \nu(1 + \lambda_x)]\gamma H,$$

where σ_x, σ_y, and σ_z are the values of the corresponding components of the stress tensor that are obtained from the solution of the two-dimensional problem and satisfy condition (1).

To allow comparison of the results, the main part of the calculations was performed for an initial hydrostatic stress state at the depth H = 900 m ($\lambda_x = \lambda_z = 1$). For variants 3 and 7 (see Table 1), we also examined situations with a high level of tectonic stresses $\lambda_x = 1.5$, $\lambda_z = 3.0$ or $\lambda_x = 3.0$, $\lambda_z = 1.5$ [2].

The actual cracked rock mass is characterized by nonuniformity of its strength and strain properties, and only qualitative estimates of its state can be obtained in a modeling based on the method of elastic superposition. Table 2 shows generalized data on the strength properties of two types of rock in the Nikolaevsk deposit. The data is based on information obtained from tests of samples [3, 4].

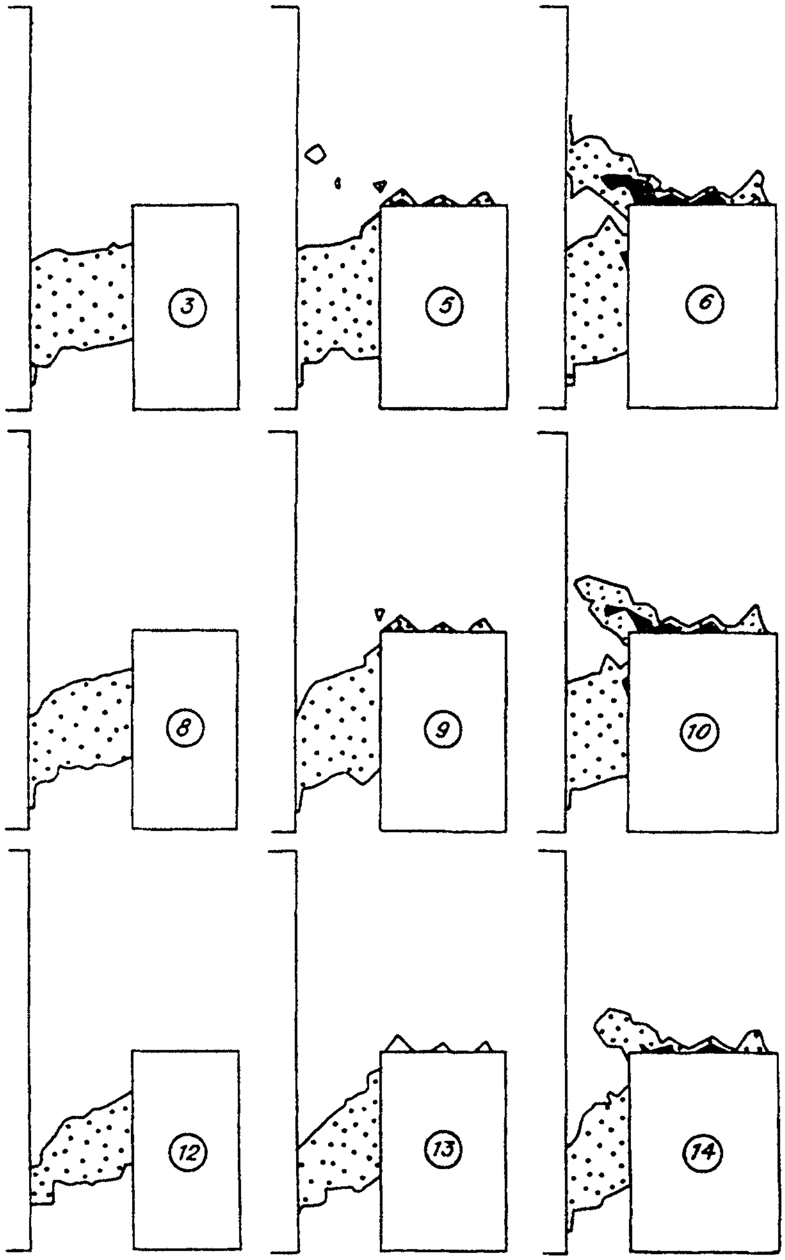
Here, c_0 and φ are the parameters of the rectilinear envelope of the limiting Mohr circles (cohesion and angle of internal friction). We determine the safety factor for the rock in the ore body as follows:

$$k = \frac{2c_0 \cos\varphi + (\sigma_{max} + \sigma_{min}) \sin\varphi}{\sigma_{max} - \sigma_{min}}, \tag{2}$$

where σ_{max} and σ_{min} are the largest and smallest principal stresses in the rock found from the elastic solution; k = 1 becomes infinite with a hydrostatic stress state equal to 1, when the Mohr circle is tangent to the envelope, and with a hydrostatic stress state less than 1, for the supercritical state.

A comparative analysis of the state of the rock mass for different values of the parameters is performed in the following manner. We calculate values of the safety factor k at the nodes of the finite-element grid by means of Eq. (2) and establish zones of supercritical deformation ($k < 1$) for the two types of rock shown in Table 2.

Analysis and Discussion of the Results. Possible collapse of the pillar and caving of the roof rock of the room impose the main limitations on the choice of the parameters of the mining system. As noted, the main system parameters that determine the stress-strain state of the rock mass are the thickness of the deposit, the height of the caving zone, the span of the working, and the width of the pillar and chamber. The size of the caving zone h is one of the parameters that make it possible to control the stress state of the mass. The pressure of the caved roof rock on the soil in the goaf is no greater than $\gamma h \approx 1$ MPa, which is considerably less than the initial vertical stresses acting in the ore body ($\gamma H = 24.3$ MPa). The presence of caved rock in a



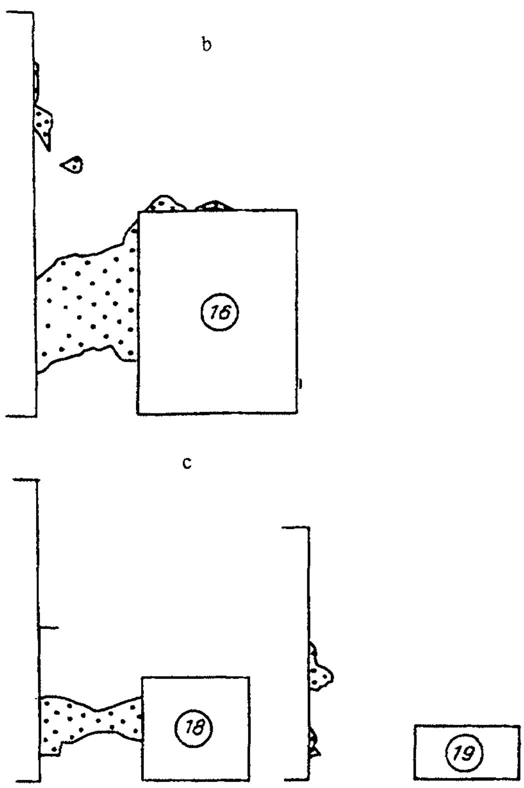


Fig. 4. Regions of the critical state of rocks according to the Coulomb—Mohr criterion for the strength specifications of skarn (dark areas) and limestone (points).

the goaf increases the load-carrying capacity of the pillar as a result of lateral thrust, so estimates obtained without allowance for the caved rock have a certain safety factor in terms of the stability of the pillar.

Figure 3 shows the distribution of the maximum principal stresses (subvertical stresses) in the central horizontal section of the ore body; the numbers of the curves correspond to the numbers of the theoretical variants in Table 1. An increase in the height of the caving zone is accompanied by a reduction in the load on the rock mass. Cutting and chipping of ore in the primary chamber is followed by an increase in the load on the pillar and on the rock on the other side of the chamber. A highly nonuniform stress state is formed in the pillar, with regions of high stresses in its side near the roof and in its lower section on the goaf side. Meanwhile, a region of tensile stresses is formed in the roof of the room. The value $h \approx m = 40$ m is probably close to optimum, since a further increase in h to 60 m only slightly (by less than 10%) reduces pressure on the pillar.

With a fixed size of caving zone $h = 40$ m, a comparative analysis of the stress state of the rock mass showed that the load in the central part of the pillar increases with an increase in the span of the working. The load increases from 35-40 MPa at $L = 120$ m to 50-55 MPa (2.1-2.3γH) at $L = 200$ m. A decrease in the width of the pillar from 20 to 12 m is accompanied by an increase in the load on its middle section by 10-15 MPa and an increase in the dimensions of the tension region in the roof. Figure 4 shows the regions of supercritical strain according to certified strength data for the two types of rock (see Table 2). The numbers in the figure correspond to the numbers of the theoretical variant in Table 1. There is a region of tensile stresses between the roofs of the room and the goaf for each of the spans (120, 160, and 200 m) with a pillar width of 12 m, which is indicative of the low stability of the room and the inexpediency of using 12-m-wide pillars in the mining system even in the case of short spans (for the given thickness $m = 40$ m).

An increase in the width of the room b_c from 20 to 30 m with fixed dimensions for the pillar $b_p = 20$ m and span $L = 200$ m (see variants 3 and 16 in Fig. 4) is accompanied by a slight increase in stresses in the pillar (by up to 5%) and an increase in the level of tension in the roof of the room.

A reduction in the thickness of the deposit (see variants 18 and 19 in Fig. 4) with fixed values of the remaining parameters ($L = 200$ m, $b_c = b_p = 20$ m, $h = 40$ m) leads to an increase in the load on the pillar and a reduction in the

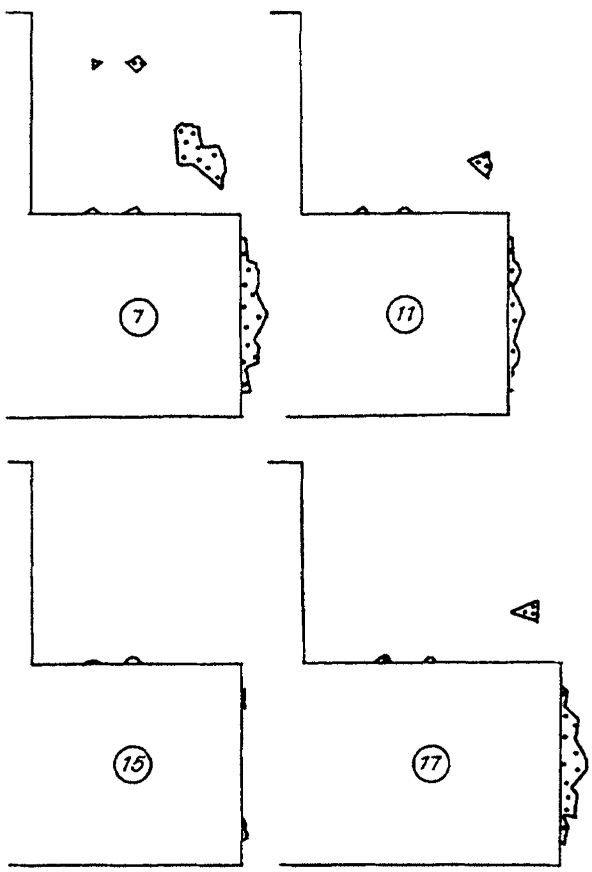


Fig. 5. Regions of the critical state of rocks after blasting of the pillar.

tensile stresses in the roof of the room. The stability of the pillar increases in this case, despite its additional loading as the thickness of the deposit decreases.

Maximum bearing pressure ahead of the stoping front is located 10-15 m from the wall of the room and reaches 40 MPa (1.6γH).

In order to reduce the loss of broken ore and improve the stability of the pillars and the working as a whole, it is best to anchor them in the rocks of the floor of the seam 7-10 m from the bottom contact of the ore body. The rock mass on the opposite side of the room undergoes minimal additional loading in this case, which improves conditions for making boreholes and haulage stages in the next block. These openings should generally be dug in the sides of the pillar of this block during this period.

The next stage is caving and complete removal of the pillar ($b_p = 0$), resulting in the formation of an overhang (a rock projection into the goaf). Mining of the pillar leads to stress redistribution in the outer part of the rock mass, increases maximum bearing pressure in the central horizontal section of the ore body to 58 MPa (2.4γH), and shifts the maximum 5-10 m toward the side of the excavation (curve 7 in Fig. 3). Figure 5 shows the possible regions of supercritical deformation of the rock for variants 7, 11, 15, and 17. An analysis of the configurations of the tensioned zones and the magnitude of the tensile stresses in the robbed area shows that robbing the overhang leads to cutting of the rock projection that tapers away from the rock mass. The size of the tensile region increases with an increase in the span of the working.

The presence of tensile stresses in the rock overhang formed after excavation of the pillar shows that the stability of the roof proper will be determined by the extent to which the rock has been disturbed. In drawing the ore of the exploded pillar, the roof may undergo stratification and cave spontaneously. The latter will be accompanied by unloading and swelling of the rock of the floor of the haulage stages. At this point, the section of the rock mass that contains haulage stages and boreholes in the next block will be subjected to an additional load that might also lead to a deterioration in their condition.

Figure 6 illustrates the effect of high horizontal tectonic stresses on the stress-strain state of structural elements of the given mining system. Despite the lower stability of the roof of the room, the variant in which the stoping front is oriented in

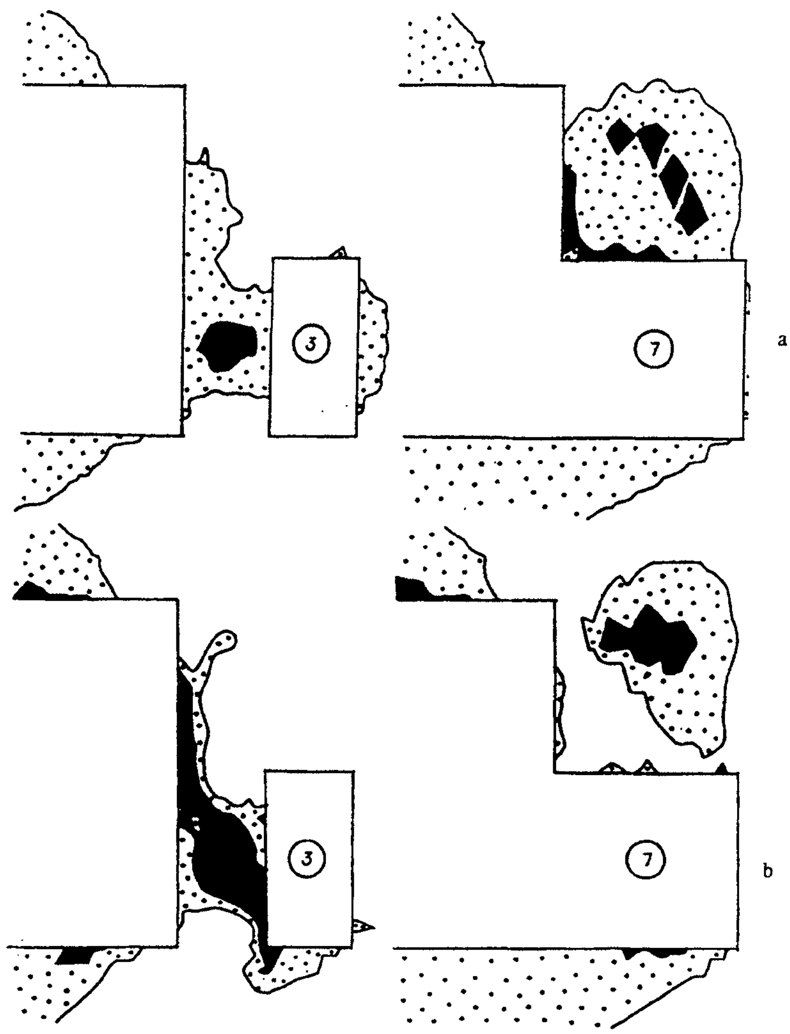


Fig. 6. Regions of the critical state of rocks with allowance for the tectonic stresses $\lambda_x = 1.5$ and $\lambda_z = 3.0$ (a); $\lambda_x = 3.0$ and $\lambda_z = 1.5$ (b).

the direction of action of the minimum horizontal tectonic stresses is preferable due to the greater stability of the pillars in this case.

CONCLUSIONS

The completed geomechanical analysis permits the following conclusions.

1. The proposed mining system, with controlled caving of roof rock, allows mining to be done safely to depths of 900 m. Here, the span of the working can be up to 200 m. The following parameters (for a deposit 40 m thick) are recommended for production tests: height of caving zone h = 40 mm; width of pillar and primary chamber $b_p = 20$ m and $b_c = 20$ m. A decrease in the thickness of the deposit increases the stability of the pillar and makes it possible to reduce the width b_p while keeping the deformation of the pillar below the critical value.

2. The initial stress state of the rock mass (before mining operations begin) affects the stability of the structural elements of the mine. For the Nikolaevsk deposit, where one horizontal component of the initial stress field is 1.5 times as great as the vertical component corresponding to the weight of the overlying formations and the other horizontal component is three times greater, it is recommended that stoping be done so that the direction of the operations front coincides with the direction of action of the minimum horizontal stress.

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