in aggregate rock strength with a decrease in the elastic moduli and the nature of its deformation, can be used to develop a method for assessing the current bearing capacity based on instrumental measurements of elastic modulus profiles along the thickness of the pillar.

Acknowledgment

The work was carried out as part of a major scientific project supported by the Ministry of Science and Higher Education of the Russian Federation, Agreement No. 075-15-2024-535 dated 23 April 2024.

References

1. Golasowski J., Vochta R., Hudeček V., Mikoláš M., Dvořák P. Room and pillar mining at OKD, as in the Czech Republic. *14th International Multidisciplinary Scientific Geoconference & Expo SGEM 2014*. Albena, Bulgaria, 2014. pp. 529–536.
2. Jong-Gwan Kim, Mahrous A. M. Ali, Hyung-Sik Yang. Robust design of pillar arrangement for safe room-and-pillar mining method. *Geotechnical and Geological Engineering*. 2019. Vol. 37. pp. 1931–1942.
3. Permyakov R. S., Romanov V. S., Beldy M. P. Salt Mining Technology. Moscow : Nedra, 1981. 271 p.
4. Makarov A. B., Rasskazov I. Y., Saksin B. G., Livinsky I. S., Potapchuk M. I. Geomechanical evaluation of roof-and-pillar parameters in transition to underground mining. *Journal of Mining Science*. 2016. Vol. 52, No. 3. pp. 438–447.
5. Instructions (Measures) for the Protection of Uralkali Mines from Flooding and Protection of Objects on the Earth's Surface from Harmful Effects of Underground Mining at Verkhnekamskoe Salt Deposit (Appendix 1). Perm : Berezniki, 2022. 117 p.
6. Instructions (Measures) for the Protection of EuroChem-UKK LLC's Mine from Flooding and Protection of Objects on the Earth's Surface from Harmful Effects of Underground Mining at Verkhnekamskoe Salt Deposit During Development of Sylvinite Formations. Perm : Berezniki, 2025. 114 p.
7. Tournaire M. Dimensions to be given to the pillars of the carriers and the Pressures to which the terrains are subjected in the depths. *Annales des mine*. 1884. Vol. 5. pp. 415–429.
8. Shevyakov L. D. Calculation of strength and strains of safety pillars. *Izvestiya AN SSSR. Otdelenie tekhnicheskikh nauk*. 1941. No. 7-9.
9. Baryakh A. A., Samodelkina N. A. To the calculation of pillar stability under condition of chamber mining. *Journal of Mining Science*. 2007. Vol. 43, No. 1. pp. 8–16.
10. Baryakh A. A., Lobanov S. Y., Shumihina A. Y., Lomakin I. S. Analysis of changes in load-carrying ability of rib pillars during the failure of technological parting. *MIAB*. 2013. No. 1. pp. 27–33.
11. Baryakh A. A., Lobanov S. Y., Lomakin I. S. Analysis of time-to-time variation of load on rib pillars in mines of the Upper Kama Potash Salt Deposit. *Journal of Mining Science*. 2015. Vol. 51, No. 4. pp. 696–706.
12. Baryakh A. A., Lomakin I. S., Samodelkina N. A., Tenison L. O. Loading of rib pillars in multiple seam mining at the Upper Kama salt deposit. *MIAB*. 2023. No. 1. pp. 5–19.
13. Pospelov D. A., Pankov I. L., Toksarov V. N., Naparina D. V. Studying deformation properties of salt rocks in laboratory and in-situ environments *Izvestija Tulskogo Gosudarstvennogo Universiteta. Nauki o Zemle*. 2024. No. 4. pp. 700–713.
14. Pospelov D. A., Toksarov V. N., Beltyukov N. L. Methodology for assessing the deformation modulus of rocks in a marginal massif using a borehole hydraulic jack. *Gornoye ekho*. 2022. No. 1(86). pp. 51–57.
15. Matveev B. V., Mikheeva M. M., Kartashov Yu. M. et al. Catalogue of Strength And Deformability Indices of Rock Samples arom Coal Deposits. Leningrad : VNIMI Publishing House, 1973. 40 p.
16. Tokhtuev G. V., Borisenko V. G., Titlyanov A. A. Physical and Mechanical Properties of Rocks of Krivbass. Kyiv : Gosgortekhizdat of the Ukrainian SSR, 1962. 102 p.
17. Altayev Sh. A., Smirnov L. N. On some physical and mechanical properties of rocks of the Karaganda coal basin (Saransk section). *Nauchnie Trudy Karagandinskogo nauchno-issledovatelskogo ugolnogo instituta*. 1966. Vol. 21. pp. 280–289.
18. Chang C., Zoback M. D., Khaksar A. Empirical relations between rock strength and physical properties in sedimentary rocks. *Journal of Petroleum Science and Engineering*. 2006. Vol. 51, Iss. 3-4. pp. 223–237
19. Moradian Z. A., Behnia M. Predicting the uniaxial compressive strength and static Young's modulus of intact sedimentary rocks using the ultrasonic test. *International Journal of Geomechanics*. 2009. Vol. 9, No. 1. pp. 14–19.
20. Malik M. H., RashID S. Correlation of some engineering geological properties of the Murree formation at lower Topa (Murree district), Pakistan. *Geological Bulletin University OF Peshawar*. 1997. Vol. 30. pp. 69–81.
21. Baryakh A. A., Asanov V. A., Pankov I. L. Physical and Mechanical Properties of Salt Rocks of the Verkhnekamskoye Potash Deposit: Textbook. Manual. Perm: Publishing house of Perm State Technical University, 2008. 199 p.
22. Stavrogin A. N., Protosenya A. G. Mechanics of Deformation and Destruction of Rocks. Moscow : Nedra, 1992. 224 p.
23. Martin C. D., Chandler N. A. The progressive fracture of Lac du Bonnet granite. *International Journal of Rock Mechanics and Mining Sciences & Geomechanics Abstracts*. 1994. Vol. 31, Iss. 6. pp. 643–659.
24. Eberhardt E., Stead D., Stimpson B. Quantifying progressive pre-peak brittle fracture damage in rock during uniaxial compression. *International Journal of Rock Mechanics and Mining Sciences*. 1999. Vol. 36. pp. 361–380. EM

M. RADOVANOVIC[1]*, Teaching Assistant, MSc in Mining Engineering, PhD Candidate, mlradovanovic@tfbor.bg.ac.rs*
R. PANTOVIC[1]*, Head of the Rock mechanics laboratory, Professor*
D. PETROVIC[1]*, Head of the Underground mining department, Professor*
M. STAJIC[1]*, Teaching Assistant, MSc in Mining Engineering, PhD Candidate*

[1]*University of Belgrade, Technical faculty in Bor, Bor, Republic of Serbia*

DETERMINATION OF THE EXCAVATION BLOCK BOTTOM STRUCTURE PARAMETERS USING FINITE ELEMENT NUMERICAL METHOD STABILITY ANALYSIS

The future of underground exploitation is reflected in the increase of the depth at which it takes place, considering that deposits at lower depths are mostly exploited. The increase in depth represents a challenge in mine design, because the increase in depth is accompanied by more difficult mining conditions. Also, one of the characteristics of underground exploitation in past years is the constant reduction of ore grades. That is the reason why caving methods take an important place in present mining. To provide safety mining environment using caving methods at greater depths it is necessary to ensure the stability of the excavation blocks. This can be achieved by selecting the optimal parameters of the excavation blocks and the proper construction of the bottom structure of the excavation blocks. In this paper, an analysis of the stability of the facilities at the bottom structure of excavation block was performed. The results of the numerical finite element analysis of the stress-strain state show that with proper bottom structure construction of the excavation block and with an appropriate layout of the loading chambers and drifts, satisfying stability can be achieved which ensures safety working conditions.

Keywords: *underground mining, mining methods, stability analysis, numerical methods, excavations, bottom structure, stresses*
DOI: *10.17580/em.2025.05.08*

Introduction

In the most of ore deposits at shallower depths, with high grade ores, the exploitation has been completed or is in the final phase, so the main characteristic of underground exploitation all over the world is the increasing of excavation depth with lower grades ore. Increasing of exploitation depth is accompanied by deterioration in the operating conditions: a change in the physical and mechanical properties of rocks, an increase in the temperature of geological environment, steady rock pressure rise in workings and its manifestation [1]. Even in such conditions, achieving a positive economic result in order to enable the profitability exploitation, are being strived. Making a profit with the lowest possible production costs is one of the primary goals of exploitation [2]. This is the reason why there is an inevitable need to reduce the costs of exploitation and obtaining ore by investing in the development of new technologies, modernization of infrastructure, optimization and better organization of production

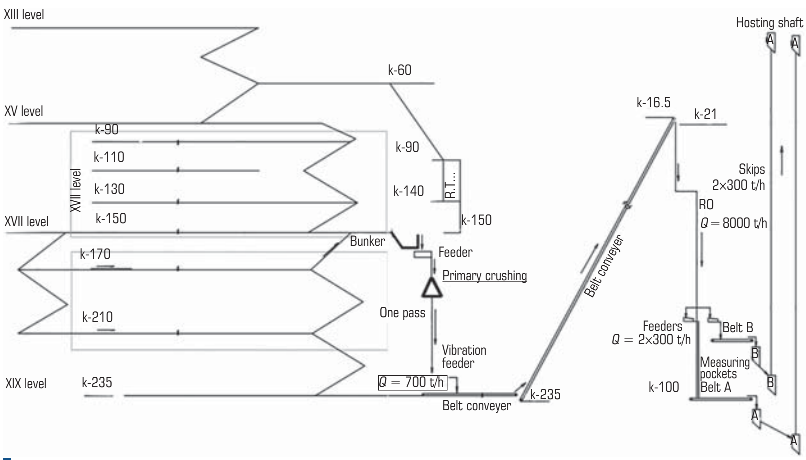

Fig. 1. Schematic view of exploitation in underground mine Jama

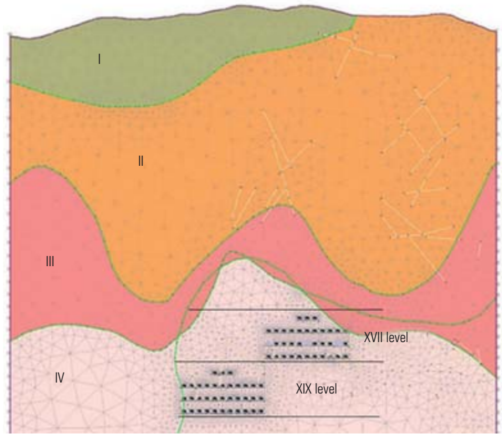

Fig. 2. Vertical crosscut of mining area with geotechnical environment

processes [3]. The increase in costs caused by the complex conditions of exploitation, as well as the conditions dictated by the competitiveness of the mining industry, require the management of mineral resources with the greatest possible efficiency. These conditions lead to the need of expanding production capacity [4].

Production increase, optimization and automation of the exploitation process can be achieved by applying the appropriate mining method. The selection of mining method has the highest role for mine planning and design. Selected mining method, in most of cases, determines most of the other solutions, which are related to the haulage and hoisting of mineral raw materials, ventilation system, and often the selection of the opening and development of the deposit. Mining method should provide some technical and economic conditions like safety and healthy working conditions, low ore losses (high ore recovery, with minimal ore dilution), required production capacity and low production costs [5].

The mining practice experience shows that the application of high-productivity caving methods has the best results for exploitation of low grade ore deposits at greater depths. The high production capacities achieved by the application of caving methods are conditioned primarily by the possibility of complete mechanization of all production processes of exploitation. Higher ore recovery is followed by higher ore dilution, which entails higher costs of haulage, hoisting and processing. This indicates that ore dilution can be increased to a certain limit, when the total costs of excavation and processing exceed the economic value of the ore [6].

Process of ore drawing is essential for ore recovery and ore dilution and thus for success applying of mining method. The bottom structure construction of the excavation block including undercut and development drifts and other facilities as well as their mutual position have the key role for succession of ore drawing process and stability of excavation. Ore drawing process is essential for ore recovery and ore dilution but at the first place must be safety working conditions which can be achieved through proper stability of excavations. Sometimes the most optimal excavation design parameters for ore recovery and dilution don't match with best stability parameters. The reasonable balance between ore recovery and drift stability has to be achieved [7]. Experience worldwide with block and sublevel cave mines shows that optimal cave design must be established as a result of together ore drawing and stability analysis research in the aim of achieving the best safety and economic results [8–10].

Modern approach in application of caving methods considers two most common bottom structure construction of excavation blocks — drawbells and trenchs [11]. During the research of the mining method design, beside determination of ore drawing optimal parameters, special attention must be dedicated to stability analysis of excavation blocks especially the bottom structure of excavation blocks in order to ensure safe surrounding for excavation works at great depths. In this paper, the results of the stability analysis using the numerical method of finite elements are presented. Stability analysis was performed for bottom structure design with trench undercutting and lateral loading in case of ore body Borska Reka in underground copper mine Jama (the Republic of Serbia).

Stability parameters research

The applicability of the mining method implies the determination of its basic structural parameters. In addition to studying and knowing the natural conditions of the deposit, it is necessary to realize a few technical and economic conditions like: health and safety work, low ore losses, required production capacity, low production costs.

The basic goal of calculating the stability of underground facilities is to define proper dimensions and shape. Adopting proper dimensions and shape will ensure the functionality of the facilities designed for different purposes during the exploitation of the deposits for the planned period.

The stability analysis was done on the example of the ore body Borska Reka. Area of Borska reka ore body is a continuance of hydrothermally altered zone of Bor ore deposit, heading towards northwest. This 1500 m long and 800 m deep area is inclined towards west with a 45° to 55° dip angle. The ore body Borska Reka, in the underground copper mine Jama Bor, lies between 500 and 1.200 meters below ground surface, near the city of Bor.

Currently the exploitation in underground copper mine Jama is carried out in ore bodies Borska Reka and T4. Ore body T4 is small ore body with limited reserves so perspective in underground mining in Jama mine is exploitation of ore body Borska Reka. Borska Reka ore body is developed in the Main Level XIX, at elevation K-235 m and in the Collection – Transport level XVII, at elevation K-155 m. For future exploitation in deeper parts of ore body, developing of level K-450 m is planned by applying some of caving methods (**Fig. 1**).

Bor ore field is a part of the Timok Magmatic Complex (TMC). TMC belongs to the East Serbian Carpatho-Balkanides which is part of the Tisza-Dacia Unit [12]. This complex is part of the Tethyan Eurasian Metallogenic zone which is known in world for its ore deposits of cooper and gold. The ore body Borska Reka is located in intensively altered andesites and their pyroclasts. From the surface, the degree of intensity of hydrothermal changes increases with depth. From chloritized and calcited rocks of andesitic composition, it gradually changes to kaolinized, and then to highly silicified, sulfated and pyritic rocks, that is, to the ore body itself. In the vicinity of the ore body Borska Reka, there is the Bor' fault, which represents the boundary between andesites and Bor's conglomerates.

An area was taken for modeling from surface to the level XVII at K-155 m elevation. In that area a total of 4 quasi-homogeneous zones were separated, representing geotechnical environments (**Fig. 2**): Bor's pelite (I), andesites (II), kaolinized andesites (III) and silicifed andesites (IV), in which actually was formed part of the ore body that was researched.

Determination of the stress around the underground object for specified size and shape, and for defined contour conditions, can be performed using three principal types of model: physical, analytic and numerical models [13]. Conventional physical model does not provide enough information on stress and displacements in the interior of the medium. Nowadays, physical model is rarely used in prediction stress practice because it is a laborious and long process [14]. State-of-the-art underground stability analysis methods include four main approaches for underground excavation design: analytical, empirical, observational, and numerical methods [15]. Which procedure will be adopted for stability analysis depends on the level of research and the accuracy of data needed to determine the failure criteria and secondary stresses of the rock mass.

Most underground structures have an irregular shape and they are located close to other underground facilities. This structures can consist excavations with attendant facilities, shafts, drifts, etc. All of them form a complex of three-dimensional shapes. Further, the rock mass is intersected by discontinuities which may or may not be coated with other materials. The analysis of stress-strain states in the rock mass, with a complex excavation construction, which cannot be done with exact mathematical methods, is performed using numerical methods [16–18]. Numerical methods in geomechanics have been known for many years, but their rapID development and practical application is closely related to the advance of computer technology and the progress of geomechanical computer softwares development.

The purpose of numerical modelling is to analyze and evaluate the behaviour of rock and rock structures under complex loading conditions, to compare and evaluate different rock engineering designs to select the most appropriate design for specific mining situations [19]. Generally, from the point of view of stress-strain analysis, all difficulties with underground mining structures stability parameters determining, are three-dimensional. With the progress of computer science, many packages are developed for three-dimensional stability analysis of rock mass and excavations. That opened the possibility to perform stability analysis using modern 3D software packages.

The examination of the stability conditions for the different sizes of excavation blocks and the development drifts was carried out using the Midas GTS NX geotechnical software. This package enables stability analysis by creating 3D numerical models using the finite element method. The main task of research was stability analysis of bottom structure of excavation block especially stability in the lateral loading chambers.

The finite element method (FEM) is the most used continuum method for the analysis of continuous or quasicontinuous media [20]. FEM can be applied, except for homogeneous materials, also for inhomogeneous, non-linear, time dependent and anisotropic behaviour of materials [21]. Making 3D finite element analysis model needs a long preparation time, respectively it requires the most accurate input data as possible in order to minimize the risk of errors. Therefore, in order to obtain the most precisely values of the stress, as well as the physical and mechanical characteristics of the rock mass, the in situ stress was measured in the Borska Reka ore body.

Besides to the design of the excavation block bottom structure, the depth of exploitation, physical-mechanical characteristics and the primary stress state of the rock mass, characteristics that are not a reflection of the natural state of the rock mass should be taken into account. During the underground drifts drivage, as well as due to blasting during the exploitation of ore, zones with worse geomechanical characteristics compared to the intact rock are formed in the areas around the excavation. The generalized Hoek-Brown failure criterion includes the degree of damage to the rock mass due to blasting to a certain extent. However, the constants that take into account the damage to the rock mass in this failure criterion are only adopted based on assumptions about the quality of blasting. That constant does not provide precise data on the degree of damage to the rock mass and the size of the disturbed zone, which can have a great impact on the quality of the rock mass. So the most effective solution would be if the degree of damage to the rock mass was determined in-situ and used as such in the stability analysis. In this way, when creating the model, it would be possible to define a zone with weak geomechanical characteristics around the facilities at the bottom of the excavation block and thus obtain more precise results that would have a much greater application from an engineering point of view.

The physical-mechanical characteristics of the rock mass obtained by laboratory testing on samples from parts of the drifts that were weakened by the impact of mining works, the physical-mechanical characteristics of the undisturbed rock mass and the stress-strain state obtained by field measurements were used as input data for creating the model.

Mining on greater depths requires knowledge on rock mass behavior, as well as on physical, technological and strength characteristics of rocks [22]. Scientific and accurate in-situ stress measurement can be used to determine the mechanical properties of engineering rock mass and analyze the stability of surrounding rock, which is a necessary prerequisite for scientific geotechnical engineering design and excavation. The measuring process of in situ stress was conducted on four measuring points in underground copper mine Jama. The measuring points was located at the level –60 m in Main haulage drift, level –150 m in the Loading drift no. 16, level –150 m in the Loading drift no. 51, and at level –235 m in the Development drift. Two of these facilities (loading drifts) are in bottom structure of the excavation blocks (stopes).

The in-situ stress test uses borehole stress relief method with CSIRO HID Cell as measuring probe, which can obtain the three-dimensional stress magnitude and direction of the measuring point in a single hole. The in-situ stress measurement starts with construction of 130 mm diameter borehole and 8–12 m depth. After that the bottom of the hole has to be grinded in order to make a hole for guiding. The center of the borehole bottom is point where small (measuring) hole of 38 mm diameter and depth of 600 mm is drilled. Finally, a measuring probe is installed in the measuring hole and it is connected with the measuring instrument outside the hole. To accomplished stress relief it is necessary to continue core drilling of 130 mm diameter around measurement hole with measuring probe inside it. During the drilling process, the wire passes through the center of the drill pipe and is led out to connect with the measuring instrument to monitor the changes in the relief process. With the deepening of the stress relief groove, the core is gradually isolated from the external stress field, the core recovers elastically, and the measured value of the instrument changes accordingly. When the instrument reading no longer changes, drilling can be finished and core is took out. Readings of the instrument before and after the stress relief is the relief reading value. **Figure 3** shows drilling and device installation steps.

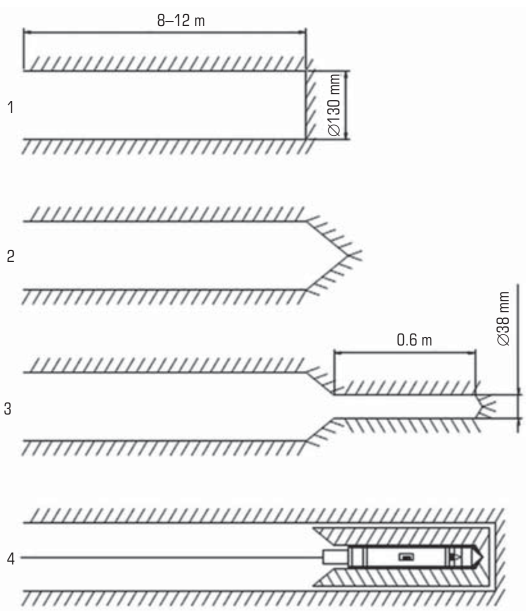

Fig. 3. Drilling and installation steps

Table. Parameters used for numerical modeling

Numerical model zones	Elastic modulus (MPa)	Poisson's ratio	Unit weight (kN/m³)	Cohesion (kN/m²)	Friction angle (°)	UCS (MPa)	Tensile strength (MPa)	Hoek's constants		
								m_b	s	a
Ore	40 423	0.205	29	5167	34	81.279	0.608	2.734	0.0206	0.502
Disturbed zone	10 219	0.207	29	1226	28	26.417	0.077	1.326	0.0039	0.506
Overlying rock mass	26 729	0.23	27	4912	34	78.200	0.621	2.579	0.0205	0.502
Caved ore	10 000	0.25	22	100	30	/	0	/	/	/

The magnitude, direction and dip of the maximum, middle and minimum principal stresses can be calculated by using the calculation program on the obtained test data.

In order to determine width and quality of damaged zone around drifts and stopes which would be used in stability analysis model, 100 mm diameter core drills are made on level –130 m (seven locations) and level –150 m (three locations). Boreholes of 0.5–1 m depth were drilled in side of loading drifts where blasting effect from surrounding stopes has great influence on rock mass quality. Obtained core samples are used for determining the geomechanical parameters which will be used in further analysis.

Results and discussion

For research of the stress-strain state of the excavation design the model shown in **Fig. 4** was established. The model was constructed in the Midas GTS NX software. It is finite element analysis software for geotechnical analysis of soil and rock deformation and stability in 3D. For excavation design trench undercutting bottom structure was adopted because that kind of undercutting, besides drawbells, is most common used and already was used in underground mine Jama.

The analyzed model represents excavation block which has a variable width (10–20 m), a variable distance between the lateral loading chambers (10–20 m) and constant height of 80 m which is same as horizon height in mine. Also undercut height has been changed depending on the slope of the sidewalls of the trench. The excavation block and upper part of ore body represents caved ore and roof. The left and right side related to the excavation block represent the ore in an undisturbed state. Also around excavation a disturbed zone was established.

After creating the block model, the data required for the analysis is entered. Four quasi-homogeneous zones are present on the model: ore in an undisturbed state, caved ore, disturbed zone and overlying rock mass, described with parameters shown in **Table**.

The parameters of the ore in an undisturbed state were taken as the result of laboratory testing on samples from ore body Borska Reka, while the parameters of the caved ore were taken for Tilva Ros ore body. Tilva Ros ore body was also part of underground mine Jama where excavation was carried out by Sublevel caving method. This ore body is no longer in exploitation but recorded parameters from production is used for this analysis, because the geotechnical environment has similar physical and mechanical characteristic as an analyzed. The parameters for disturbed zone (physical-mechanical characteristics and zone width) were obtained from results of laboratory testing on samples from loading drifts in bottom of excavations in ore body Borska Reka. The geotechnical environment shown in Fig. 2 represents overlying rock mass (Bor's pelite, andesites, kaolinized andesites) which isn't shown in this model because the emphasis is on bottom structure stability, but it is included in numerical stability analysis. To simplify the model, these three different environments (in the fourth one from the Fig 2. the ore body has formed) are represented as one quasi-homogeneous zone in the model because all of them have similar physical-mechanical properties (only kaolinized andesites have weaker properties but they are the part of a caved material which is represented in the model as a caved ore). The values of the principal stresses were taken from conducted test described above.

The influence of Bor's fault was assessed by measuring in situ stresses in the ore body (values and orientations). Local faults and other discontinuities are not directly taken into account in the model, which is certainly the main limitation of the model in addition to homogeneity and isotropy, but it can be saID that these limitations are partially taken into account because the input parameters for the analysis are mostly taken from laboratory samples taken from ore body itself.

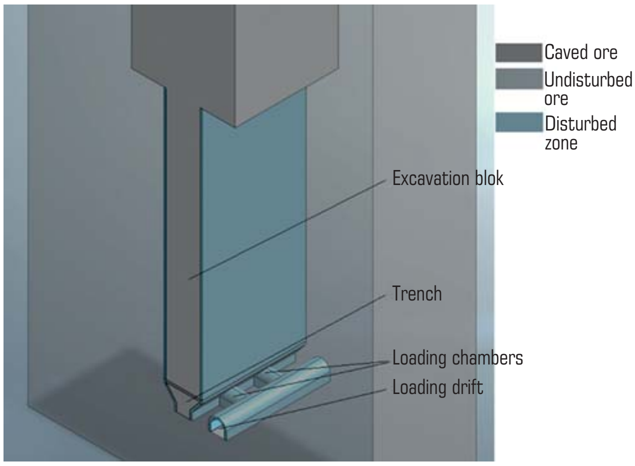

Fig. 4. Excavation block design

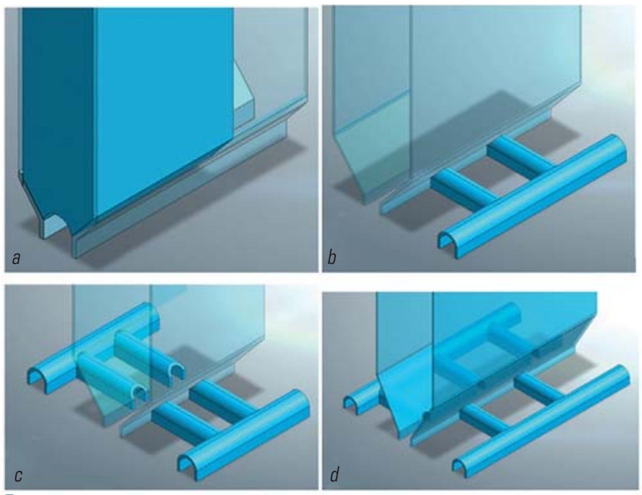

Fig. 5. Bottom structure design:
a – Frontal loading; *b* – One-sided lateral loading; *c* – Two-sided lateral loading; *d* –Two-sided lateral loading with loading chambers in checkerboard pattern

For the analysis, the generalized Hoek-Brown failure criterion was used in the case of ore and disturbed zone while the Mohr-Coulomb non-tension model was used for caved ore. The applied software uses elastic modulus, poisson's ratio and uniaxial compressive strength of intact rock as input parameters. For chosen failure criterion there is a need to get some more input parameters which can be determined in applied software, by laboratory tests, empirical formulas or in some other softwares. Also, besides estimated GSI, elastic modulus and uniaxial compressive strength of intact rock was used to obtained values of the Hoek constants m_b, s and a were obtained using RocLab software in case for generalized Hoek-Brown failure criterion.

The in-situ stress measurement results are obtained by combining the stress relief test data with the rock property test data. The results show that one principal stress is located in a nearly vertical plane and two principal stresses are located in a nearly horizontal plane. Also vertical principal stress is similar to pressure of overlaying rock mass for observed measurement points.

The following parameters were used for stability analysis:

– Vertical stress $s_V = 17000$ Pa;

– Horizontal stress $s_h = 10000$ Pa.

Based on the input data, a numerical finite element model, consisting of a hybrID mesh set is formed. The research was carried out for the case of frontal loading and one-sided and two-sided lateral loading which is shown in **Fig. 5**. During the research position of loading chambers in case of two-sided lateral loading was changed in order to find the best position of loading chambers in aspect of stability. Also stability analysis was carried out in case of lower position of loading chambers and loading drift related to undercutting level.

Stability analysis results are obtained using the software simulation. The special attention was paID to the stability of the facilities at the bottom structure of excavation block, because all the technological operations for the exploitation of ore are performed there. The results are presented through main effective stress, displacements and safety factor, whereby the parts that are unstable have safety factor less than 1, and the parts that are stable have safety factor greater than 1.

The results of the stability analysis are shown on the example of one-sided lateral loading with two loading chambers, considering that in all other cases shown in Fig. 5, changes in mean effective stress and displacements occur in a similar way. The analysis refers to the very bottom of the excavation block, given that all operations on obtaining of ore take place in this part of the excavation. **Fig. 6** shows the change in main effective stress in relation to different distances between the lateral loading chambers and for different block widths and cutting heights.

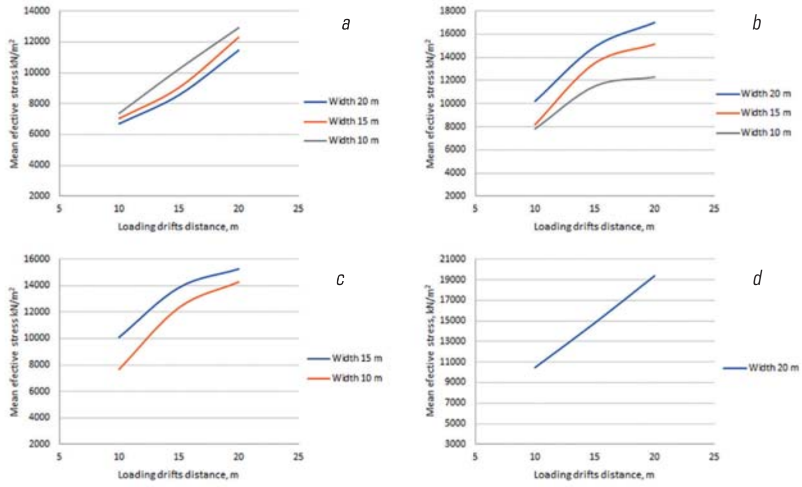

Fig 6. Relation between main effective stress and loading chambers distance:
a – B/h ratio 1.67 (h = 12 m, 9 m, 6 m); *b* – B/h ratio 1.43 (h = 14 m, 10.5 m, 7 m); *c* – B/h ratio 1.25 (h = 12 m, 8 m; *d* – B/h ratio 1.21 (h = 16.5 m)

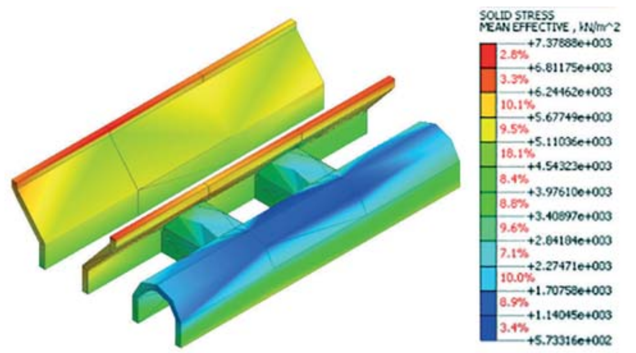

Fig. 7. Main effective stress at the bottom structure with one-sided lateral loading

The analysis showed that a certain regularity in the change of the main effective stress and displacement occurs with a change in the distance between the lateral loading chambers (drifts) for a certain width/height ratio of the bottom of the block. Detail *a*) from Fig. 6 shows the change, i.e. the increase in main effective stress with increasing distance between the side loading chambers for a trench width-to-height ratio of 1.67, detail *b*) from the Fig. 6 shows the main effective stress change in relation to the distance between the lateral loading chambers for a trench width-to-height ratio of 1.43, in detail c the ratio between the lateral loading chambers for a trench width-to-height ratio of 1.25, and in detail *d*) width-to-height ratio of 1.21. The highest main effective stress concentrations occur on the side of the trench at the junction of the trench and the excavation block with maximum values between the loading chambers as shown in **Fig. 7**.

Similar to the results shown on the previous graphs, changes in displacements for same cases shows increasing of displacements value related to the distances between lateral loading chambers for a certain width to height ratio of bottom structure of excavation block (**Fig. 8**).

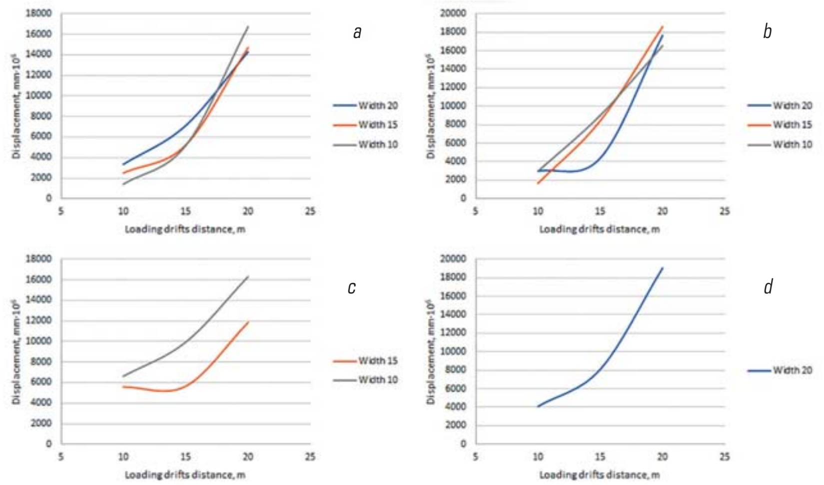

Fig 8. Relation between displacements and loading chambers distance:
a – B/h ratio 1.67 (h=12 m, 9 m, 6 m); *b* – B/h ratio 1.43 (h= 14 m, 10.5 m, 7 m); *c* – B/h ratio 1.25 (h= 12 m, 8 m; *d* – B/h ratio 1.21 (h = 16.5 m)

As already have been mentioned safety factor is one more indicator that shows if some part of excavation block (in this case bottom structure) is stable or unstable. The following **Fig. 9** shows relation between safety factor and distances between lateral loading chambers for a certain trench width to height ratio. Opposite to previous two indicators, this time safety factor decreases with increasing of lateral loading chambers distance (increasing of lateral loading chambers distance is followed with increasing of stress and displacement, so it is logical that safety factor decreases).

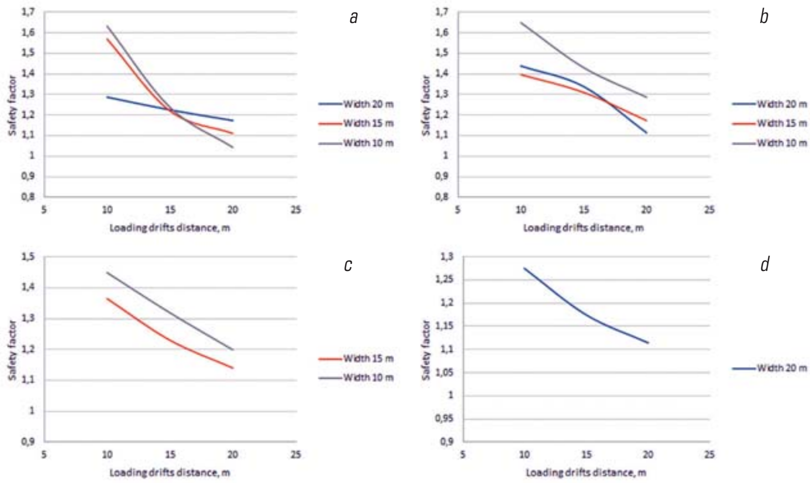

Fig. 9. Relation between safety factor and loading chambers distance:
a – B/h ratio 1.67 (*h* = 12 m, 9 m, 6 m); *b* – B/h ratio 1.43 (*h* = 14 m, 10.5 m, 7 m); *c* – B/h ratio 1.25 (*h* = 12 m, 8 m); *d* – B/h ratio 1.21 (h = 16.5 m)

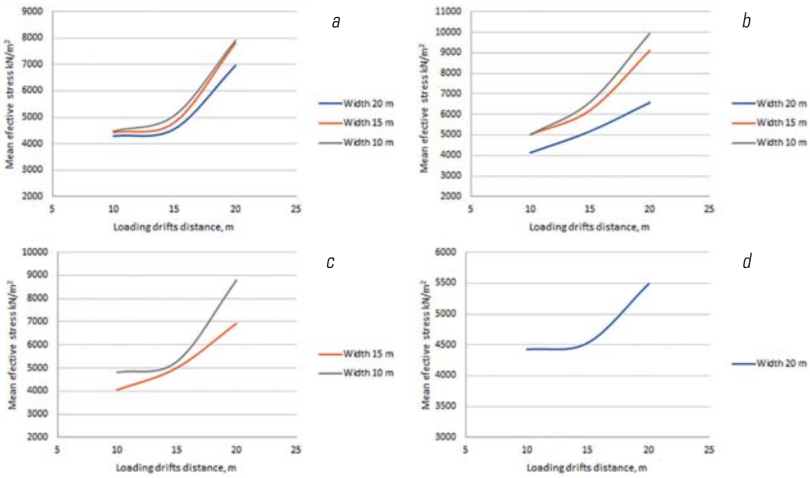

Fig. 10. Relation between main effective stress and loading chambers distance in case of stability of loading and haulage drifts:
a – B/h ratio 1.67 (*h* = 12 m, 9 m, 6 m); *b* – B/h ratio 1.43 (*h* = 14 m, 10.5 m, 7 m); *c* – B/h ratio 1.25 (*h* = 12 m, 8 m); *d* – B/h ratio 1.21 (*h* = 16.5 m)

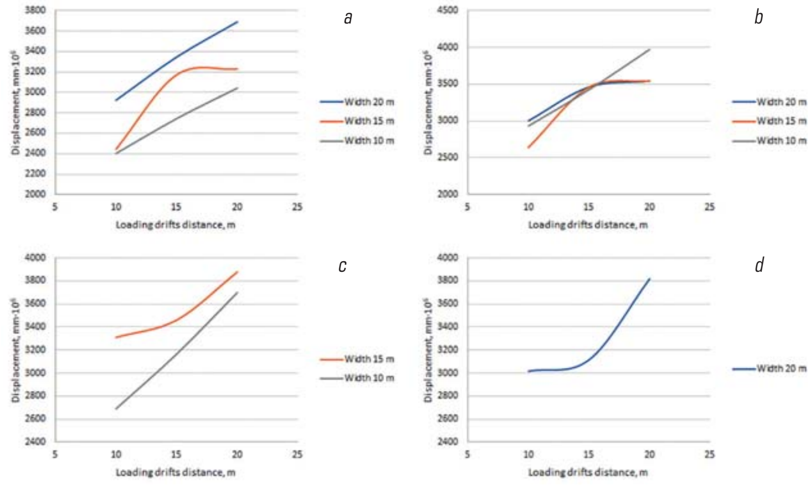

Fig 11. Relation between displacements and loading chambers distance in case of stability of loading and haulage drifts:
a – B/h ratio 1.67 (*h* = 12 m, 9 m, 6 m); *b* – B/h ratio 1.43 (*h* = 14 m, 10.5 m, 7 m); *c* – B/h ratio 1.25 (*h* = 12 m, 8 m); *d* – B/h ratio 1.21 (*h* = 16,5 m)

Given that after undercutting and caving of the ore, which is carried out from the undercutting drift at the bottom of the trench, the loading of the caved ore is carried out from the loading chambers (drifts) and later transported through the haulage drift, the stability of these structures (isolated from the trench) was also separately analyzed based on main effective stress and displacement, which is shown on **Fig. 10** and **Fig. 11** respectively.

It can be seen from the graph that the stresses and displacements change according to the same principle as for the entire bottom of the block. Given that the lateral loading chambers are located under the slope of the trench, they are partially protected in this way and the stress redistribution is such that the stresses and displacements are lower than in the trench itself. In all analyzed cases haulage drift is at proper distance so that it is not endangered with excavation works.

The contact zones between the blasted (caved) ore and the undisturbed ore in upper edges of excavation block show that the stress concentration in that part is increased. In case of frontal loading stresses is as we expected, higher than in case of lateral loading because junction between loading chambers and trench is protected with reef of ore (slope of the trench). In the junctions of the loading chambers with the trenches at the bottom structures of the excavation blocks in both cases (one-sided and two-sided) of lateral loading, the stress concentration is also increased, but that the safety factor for observed parameters is greater than 1. Higher safety factor values are achieved for bottom structure with lower position of loading chambers and drifts related to undercutting level. The greatest concentration of stress is at the sides of the trench in part where trench turns into an excavation block.

Figure 12 shows results of stability analysis based on relationship between mean effective stress and displacements and safety factor. In theory a higher trench height favors safer loading because the height of the reef of ore that protects the loading chamber is higher in that case. But analysis showed that not only the height but the width also has great influence on loading chamber stability. Actually the width to height ratio has the great influence on stability with conclusion that B/h ratio with higher undercutting height gives better safety factor. One-sided lateral loading is shown as better in case of smaller block width, while with larger block width two-sided lateral loading with opposite loading chambers in checkerboard pattern also shown good stability results. At the end of the research, the input parameters were changed, primarily uniaxial compressive strength and main stresses in order to obtain a model based on which, for certain characteristics of the rock mass, the geometric parameters of the bottom structure would be obtained, which ensure safe work conditions.

Conclusion

The exploitation of ore deposits takes place in more complex conditions and at greater depths, so the observation of the geomechanical parameters of excavation blocks and rock masses is of increasing importance.

In caving methods, except the parameters of excavations, the bottom structure construction plays a very important role. Given that at the bottom of the block, i.e. in the facilities at the bottom of the block, all the main technological operations during exploitation take

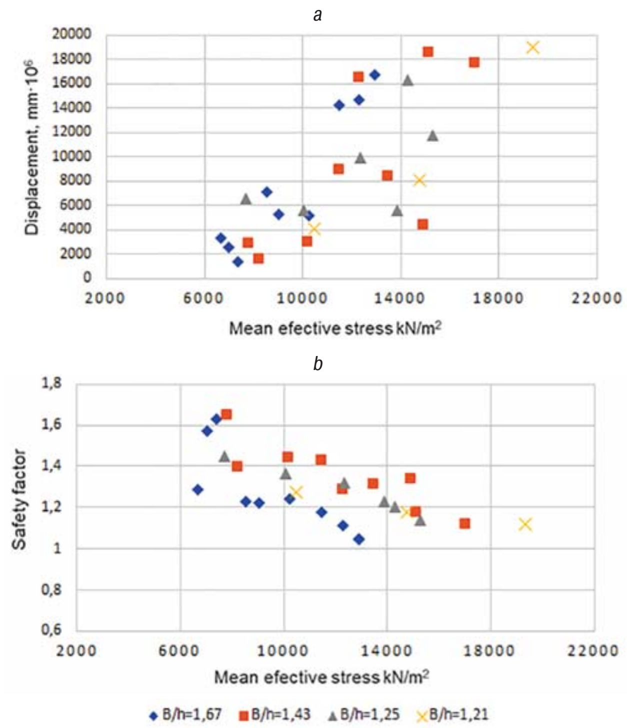

Fig. 12. Relation between mean effective stress and Displacements (*a*); Safety factor (*b*)

place, which means that the stability of these structure is of the greatest importance for the safety of employees during excavation, as well as the successful application of the chosen method.

In order to successfully perform a stability analysis, it is necessary to know the stress-strain state of the rock mass, as well as the physical-mechanical properties of the ore and rocks. In general, all problems of mining structures, are three-dimensional. That is why the research was carried out on a 3D model created in software for geotechnical analysis which is use finite element method for stability analysis.

3D analysis requires as precise input data as possible to minimize the risk of errors. Therefore, in order to obtain the most realistic input data, field measurements were made. Research is based on data collected in copper underground mine Jama Bor.

Research is carried out on numerical model of excavation block based on the defined input parameters, which can be used to determine how and in what way a certain parameter affects the stability of the bottom structure of the excavation block. Research was conducted for various geometrical parameters of excavation block and bottom structure and for different position and relation of facilities in bottom of the block.

Obtained results represented with safety factor show in which case and with which geometrical construction bottom structure of excavation block has the best stability parameters.

The final goal of the research is construction of a model which can be used for determination of the most favorable geometric parameters of the bottom structure of the excavation block for the appropriate input parameters, which can provide the required stability to the observed mining method and therefore enables safe conditions for work on exploitation of ore.

References

1. Kurlenya M. V. Deep-Level Mineral Mining in Siberia and Russian Far East: Actual Objectives and Trends of Research. *Journal of Mining Science*. 2021. Vol. 57. pp. 539–545.
2. Puzder M., Pavlik T., Molokač M. et al. Cost-ratio model proposal and consequential evaluation of model solutions of manufacturing process in mining company. *Acta Montanistica Slovaca*. 2017. Vol. 22(3). pp. 270–277.
3. Wiecek D., Burduk A., Kuric I. The use of ANN in improving efficiency and ensuring the stability of the copper ore mining process. *Acta Montanistica Slovaca*. 2019. Vol. 24(1). pp. 1–14.
4. Malli T., Pamukcu C., Kose H. Determination of optimum production capacity and mine life considering net present value in open pit mining at different overall slope angles. *Acta Montanistica Slovaca*. 2015. Vol. 20(1). pp. 62–70.
5. Milicevic Z., Milic V. Underground mining technology of mineral deposits. Bor, Serbia : RDS Group, 2013. 332 p.
6. Milic V., Radovanovic M. Determination of the main parameters of semi-level induced caving method with lateral loading. *Journal of Mining Science*. 2021. Vol. 57. pp. 76–85.
7. Guo M., Tan Y., Chen D., Song W., Cao S. Optimization and Stability of the Bottom Structure Parameters of the Deep Sublevel Stope with Delayed Backfilling. *Minerals*. 2022. Vol. 12, Iss. 6. ID 709.
8. Campbell R., Banda H., Fajar J., Brannon C. Optimising for success at the Grasberg Block Cave. *Proceedings of the Fourth International Symposium on Block and Sublevel Caving*. 2018. pp. 3–16.
9. Cuello D., Newcombe G. Key geotechnical knowledge and practical mine planning guidelines in deep, high-stress, hard rock conditions for block and panel cave mining. *Proceedings of the Fourth International Symposium on Block and Sublevel Caving, Australian Centre for Geomechanics, Perth*. 2018. pp. 17–36.
10. Shelswell K. J., Labrecque P. O., Morrison D. M. Increasing productive capacity in block caving mines. *Proceedings of the Fourth International Symposium on Block and Sublevel Caving, Australian Centre for Geomechanics, Perth*. 2018. pp. 107–118.
11. Milicevic Z. Sublevel and Block Caving Methods. Bor (Serbia) : University of Belgrade Technical Faculty in Bor, 2008. 459 p.
12. Bajic S., Bajic D., Gluscevic B., Ristic Vakanjac V. Applaying the Vikor method to select the optimal underground mining technology. *Comptes Rendus de L'Academie Bulgare des Sciences*. 2023. Vol. 76, No. 1. pp. 96–104.
13. Jager A. J., Ryer J. A. Hanbook on Rock Engineering Practice for Tabular Hard Rock Mines. Braamfontein (RSA) : SIMRAC, 2001. 371 p.
14. Brady B. H. G., Brown E. T. Rock mechanics for underground mining 3rd Ed. Dordrecht : Springer, 2004. 628 p.
15. Sepehri M. Finite Element Analysis Model for Determination of In-situ and Mining Induced Stresses as a Function of Two Different Mining Methods Used at Diavik Diamond: thesis of inauguration of Dissertation … of Doctor of Engineering Sciences. Edmonton : University of Alberta, Department of Civil and Environmental Engineering, 2016. 305 p.
16. Galachenko Yu. P., Eremenko V. A., Kosyreva M. A., Vysotin N. G. Features of secondary stress field formation under anthropogenic change in subsoil during underground mineral mining. *Eurasian Mining*. 2020. No. 1. pp. 9–13.
17. Galachenko Yu. P., Eremenko V. A. Evolution of secondary stress field during underground mining of thick ore bodies. *Eurasian Mining*. 2021. No. 1. pp. 21–24.
18. Kosyreva M. A., Eremenko V. A. Numerical stress-strain modeling of honeycomb mine structures with vertical stopes of cylindrical form. *Eurasian Mining*. 2024. No. 1. pp. 36–39.
19. Wagner H. Deep Mining: A Rock Engineering Challenge. *Rock Mechanics and Rock Engineering*. 2019. Vol. 52. pp. 1417–1446.
20. Bobet A. Numerical methods in geomechanics. *Arabian Journal for Science and Engineering*. 2010. Vol. 35(1). pp. 27–48.
21. Nikolic M., Roje-Bonacci T., Ibrahimbegovic A. Overview of the numerical methods for the modelling of rock mechanics problems. *Technical Gazette*. 2016. Vol. 23, No. 2. pp. 627–637.
22. Mirenkov V. E. Deformation of Rock Mass in the Vicinity of Underground Opening at Great Depth. *Journal of Mining Science*. 2021. Vol. 57. pp. 380–385. EM