NUMERICAL SIMULATION OF THE OPEN PIT STABILITY BASED ON PROBABILISTIC APPROACH

Purpose. To identify development regularities related to a stress-strain state and stability of the open pit walls while mining the steeply inclined iron-ore body at various stages of mining considering deterministic and stochastic components of the rock mass structural heterogeneity.

Methodology. Numerical 3D simulation of the rock stress-strain state; application of the “strength reduction” procedure to determine a safety factor, taking into account the rock mass heterogeneity based on a stochastic model.

Findings. The distribution of maximum shear deformations and displacements within the rock mass, making up the pit wall, has been obtained. Potential slide surfaces and safety factors have been determined at various stages of the inclined ore body mining. The effect of the rock mass structure on the pit wall stability has been estimated. A comparison of calculations based on the 3D and 2D models has been carried out. The correction coefficient has been obtained, which allows using the 2D model for multivariate calculations. The relationship of safety factor versus the overall slope angle has been developed.

Originality. It has been proved that pit walls retain their stability (the stability factor (SF) is not less than 1.27) while mining the steeply inclined ore body with the transverse panels from top to bottom within each newly cut layer, despite the increasing depth of mining. It is shown that modeling of the real geological structure of a three-dimensional rock mass area factors into the decrease in (by 7%) the safety factor in comparison with the results of the homogeneous model. A probabilistic-statistical approach has been proposed to consider the heterogeneity of the rock mass and avoid unreasonably optimistic forecasts of the pit wall stability. It is shown that 3D modeling gives SF, which differs by 8% from the values obtained in the 2D model. This allows substantiating the correction coefficient to improve the 2D modeling results.

Practical value. The regularity of changes in the pit wall stability has been determined depending on the overall pit slope angle in terms of mining-geological and mining-technical conditions of the Kachar deposit, taking into account the real rock heterogeneity.

Keywords: ore mining, open pit, wall stability, stochastic environment, numerical simulation, rock stress-strain state, safety factor

Introduction. Ukraine and Kazakhstan are leaders in iron-ore deposits being developed intensively by open-pit mining. Modern iron-ore open pits have numerous problems concerning the peculiarities of mining steeply inclined seams, increasing mining depth as well as growing risks of pit wall instability and consequent landslide processes [1]. Slope control in open pits is a complex problem, whose solution includes first of all mining and technological issues on rational procedure and intensity of mining operations owing to the use of certain equipment types. It is shown that modeling of the real geological structure of a three-dimensional rock mass area factors into the decrease in (by 7%) the safety factor in comparison with the results of the homogeneous model. A probabilistic-statistical approach has been proposed to consider the heterogeneity of the rock mass and avoid unreasonably optimistic forecasts of the pit wall stability. It is shown that 3D modeling gives SF, which differs by 8% from the values obtained in the 2D model. This allows substantiating the correction coefficient to improve the 2D modeling results.

In terms of open-pit mining the steeply-dipping deposits by the round-shape open-pit fields, development of the rock overburden slopes and ore by transverse panels from top to bottom within each newly cut steeply-dipping layer is one of the most efficient solutions [2]. In this context, high-productive use of the earthmoving and motor complexes as well as the required intensity of mining operations can be reached by elimination of dead-end dump turn on wide sites, more qualitative rock crushing in terms of multi-row delay-action blasting, and reduced rock movement after blasting into the mined-out space. High technical and technological indices are obtained by reducing premature widening of the walls of deep and ultradeep open pits. However, before the approach of mining operations to the limiting surface contour of deep open pits, the height of steeply inclined layers increases; that will intensify the risk of rock mass instability in terms of the greater thickness of the overburden loose rocks (up to 160 m).

Due to that fact, a problem arises concerning both the prognosis of a stress-strain state of the rock mass at different mining stages and estimation of the pit walls stability taking into consideration mining and geological features of the deposit as well as mining and technical conditions of the ore deposit development.

Literature review. The open pit depth is constantly growing; thus, stability of the slopes under development is of great importance to optimize the resulting slope angle by its maximum increase with observance of the safe mining standards. The stability of the open pit slopes in terms of large angles is evaluated by a combined analysis involving the methods of limit equilibrium and numerical simulation [3], i.e. by using such systems as MSARMA (“Evaluation and analytical system for slope stability analysis”) [4], and so on.

Numerous cases of slope collapse recorded in copper, gold, iron, diamond, lead, zinc, and platinum open pits all over the world make the researchers develop their hazard assessment systems where main mining components are analyzed and risks are determined based on such algorithms as discriminatory analysis, “certainty ellipses” construction, and others [5].

A complex analysis of the slope stability with the use of radars (the SAR system) was proposed in [6] to monitor iron-ore pit walls in the province of Carajás (the Amazon region).

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Paper [7] describes positive practices of applying a combination of methods including Google Earth and field surveys to study a process of slope destruction in the iron-ore open pit.

Preliminary evaluation of the pit wall stability may be performed based on physical modeling. A physical model helps to visualize a process of slope collapse and assess the main geometrical parameters of the landslide process with the following prognosis of possible geohazard in a real open pit [8, 9].

A large body of research focuses on the high probability of dramatic deformation processes within the open pit walls, which is stipulated by the composition of several unfavorable factors. Paper [10] shows that unpredictably massive deformations within the pit wall occur due to the effects of physical weathering that has stipulated reduced soil stability and the development of fissures along with the weak layer. The authors of the paper [11] indicate the complicated conditions of lignite development in Greece due to the fact that within the recent decades the open pits have become larger and deeper (down to 200 m); as a result, currently, they are more difficult to control.

Modeling the critical states of the pit walls is similar in many ways to the analysis of a landslide development [12]. The authors of the paper [13] reveal a problem of landslide modeling in soft clays. Reduction of clay stability in terms of plastic deformation results in the fact that breaking is irreversible so that the soil shears within a small slope can cause a series of regressive failures and result in a sudden catastrophic landslide.

Authors of all the aforementioned papers are unanimous in the idea that mathematical simulation is an important constituent part of geotechnical studies. Modern numerical methods make it possible to consider the enumerated factors determining slope stability. The results of numerical simulation by both a method of finite and discrete elements and a method of finite differences are discussed. The authors of the early mentioned paper [12] point out that successful forecast of landslides in clays requires reliable numerical simulation, being capable of reflecting extreme material deformations. The positive experience of finite element modeling of the deformation processes (using RS3 program) was demonstrated while reconstructing the landslide in Saint-Jude, Quebec, Canada [13]. Paper [14] shows that transportation of overburden and mineral raw materials in terms of open pits is one of the most complicated and costly processes. Consequently, the design of traffic arteries also involves numerical modeling to minimize costs for long-term construction and maintenance. The FEM analysis of a stress-strain state of the natural or artificial slope allows not only forecasting the potential slip surface but also developing certain methods to ensure the stability of a geotechnical object [15].

Most researchers agree that the accuracy of the modeling is guaranteed by the initial data reliability, first of all, relative to mechanical characteristics of lithological differences.

Thus, the authors of the paper [16] note that the assessment of shear strength is of critical importance for stability analysis. However, the limited availability of direct measurements in engineering practice makes the evaluation of real soil properties quite difficult. That very circumstance is highlighted by the authors of the paper [17]. It is stressed that due to the difficulty of identifying mechanical properties of weak seams within the pit walls, it is necessary to carry out regressive analysis in combination with the data on rock displacement monitoring.

Unsolved aspects of the problem. The represented example demonstrates that stabilization of slopes and open pit walls is determined to a large extent by the adequate prognosis of geological- mechanical processes under certain geological conditions. In this context, mathematical, in particular, numerical simulating the stress-strain state (SSS) of the rock mass is quite an efficient tool. Nevertheless, even in the case of using powerful and multi-functional software complexes for creating 3D models, the reliability of the initial data is the compulsory condition.

The mechanical properties of rocks are such obligatory data apart from geometrical parameters of the areas and loading conditions. In this context, there is always some uncertainty in the determination of those characteristics stipulated by the considerable rock mass heterogeneity. To date, there is no unambiguous approach to consider the structural heterogeneity of the rock mass, in particular, the systems of joints, while numerical simulating the rock stress-strain state and assessing the stability of the pit walls.

The purpose is to identify regularities in changes of pit wall stability at different stages of technological operations while mining steeply-inclined layers based on 3D modeling of the rock wall considering the rock mass structural heterogeneity. The research was carried out in terms of the Kachar magnetite-ore deposit (Kazakhstan) where the problem of pit stability is quite acute due to the intense deepening of mining.

Description of the research object. The Kachar magnetite-ore deposit belongs to the major iron-ore belt of Turgay trough. Engineering and geological conditions of the deposit development are complicated due to the considerable thickness of unconsolidated sediments as well as weak surfaces in hard rock directed towards the mined-out space. The thickness of the unconsolidated sediments varies from 130 to 200 m, being 160 m on average. Meso-Cenozoic deposits are represented by sedimentary rocks of different ages: clay loams, clays, sands, argillaceous opokas and sandstones, and conglomerates. Below, clays of the bedrock weathering layer occur; their thickness reaches up to 30 m, being 5—10 m on average. Within certain areas, there is no clay of the weathering layer at all. A complex of hard and medium-hard Paleozoic rocks is represented by sedichronal, effusive, sedimentary-igneous, and metasomatic rocks (porphyrites, aphyrites, tuffs, tuffites, tuff sandstones, limestones, and ores of magnetite composition). There are also joint systems of different orientations.

Mining of rock overburden and steeply inclined ore layer from the top to bottom means that the mining depth increases at each stage of operations; thus, the resulting slope angle increases as well. Consequently, safer and more efficient mining under the specified conditions requires adequate estimation of the pit wall stability.

Methods. To identify the rock stress-strain state, a well-proven numerical method of finite elements is used implemented in the licensed software RS3 (Rocscience). The computational scheme (Fig. 1) includes a wall part made up of hard rock as this is the area which stability is of special interest for the development of technical solutions. The main part of the rocks involved in the deformation process is represented by ore, limestones, and tuffs. The layers of those rocks are modeled according to the data on their thickness and parameters of bedding. Digital 3D representation of the ore body at each mining stage is performed in the software SURPAC, which files are imported in the RS3 code.

While simulating a three-dimensional area of the rock mass, boundary conditions are determined by the fixation of the model displacements: plane XY (bottom of the model) is fixed along the axes X, Y, and Z; plane YZ is fixed along the axes X and Y, and plane AX is fixed along the axis Y.

A finite-element mesh of the model (Fig. 1, b) is obtained by the uniform division of the area into 10-apex tetrahedrons. The number of elements for the model represented below is 267 595.

The pit wall stability is assessed based on the elastoplastic model of the rock mass and the Coulomb-Mohr failure criterion

$$\tau = c + \sigma_\tau \tan \varphi,$$  

(1)

here $\tau$ is tangential stress (shear stress) and $\sigma_\tau$ is normal stress. Internal friction angle $\varphi$ and cohesion $c$ are the strength characteristics included in the strength conditions.

The safety factor is the most significant result of the numerical simulation indicating the possibility of wall operation in terms of the present geometrical parameters

$$F = \frac{\text{shear strength}}{\text{shear force}}.$$
It has been noted before that according to the data by the geological service of an open pit, there is certain jointness of different degrees within the rock mass. Paper [20] indicates the prevailing role of jointness in reducing rock strength and the difference of the heterogeneous mass strength from one of the rock samples tested in a laboratory. Since the strength properties of rocks determine directly the pit wall stability and are the most important initial data to simulate the rock mass state, special attention is paid to the consideration of the aforementioned difference (scale effect) in the mentioned studies. There are different methods to consider rock mass heterogeneity and its effect on mechanical properties. Most of them are based on the generalization of laboratory tests of samples and in-situ observations, which were further implemented in different rating classifications. The rating systems help to identify the rock mass quality depending on the availability of various features of heterogeneity, i.e. the ones of jointness.

We propose a probabilistic approach, based on the statistic theory of strength to consider structural features of rocks. Consideration of rock heterogeneity based on a probabilistic approach to the strength property assessment. The sample with \( n \) values of the studied mechanical characteristic \( X \) is the result of rock testing. The average value \( \bar{X} \) is the statistic estimation of the theoretical mean \( x_m \). Assume a priori that the real mean \( x_m \) of the characteristic \( X \) differs from the average sampling \( \bar{X} \) due to heterogeneity of a real rock or soil mass, for which the samples under testing are structural elements. This dissimilarity is characterized by the structural coefficient \( k_c \) being equal to

\[
k_c = \frac{x_m}{\bar{X}}. \tag{2}
\]

We will also consider that the strength of the heterogeneous rock mass is less than the one of the rock samples (structural elements of the rock mass). Indeed the probability to identify some defects within the large volume is higher; thus, the strength characteristic of a larger body is lower, i.e. one can assume that \( x_m \) is the least value of all the values of the given mechanical characteristic, and inequality \( X \geq x_m \) takes place. The probability of such inequality \( P \) is determined by the distribution function \( F(x) \) of the sign under analysis \( X \)

\[
P(X \geq x_m) = 1 - F(x_m) = P_0. \tag{3}
\]

Probability \( P_0 \) characterizes a level of the object significance; it is preset a priori. Then, a true value of the mechanical characteristic \( x_m \) will be determined from equation (3), whose concrete form depends on the hypothesis about the type of function distribution of structural elements strength \( F(x) \). In [18], the authors propose to take normal or logarithmically normal distribution of probabilities as a stochastic model of strength of structural elements of the rock mass. In terms of the current research, based on the studies on different soil types, we propose to consider the strength characteristic, i.e. cohesion and internal friction, to be distributed according to the uniform law. Contrary to the normal one, this law assumes a considerable (up to 60 %) spread of the characteristic values relative to the mean, which is peculiar for the jointed soil or rock.

In case of the uniform law the distribution function looks like

\[
F(x) = \frac{x-a}{b-a};
\]

Taking into consideration (3), we obtain

\[
P_0 = 1 - \frac{x_m-a}{b-a},
\]

then

\[
x_m = (b-a)(1-P_0)+a. \tag{4}
\]

It is known that in case of uniform distribution, the moments of distribution – the mean \( M(x) \), standard deviation \( \sigma(x) \), and relative variation \( \eta \) are equal to

\[
M(x) = \frac{b+a}{2}; \quad \sigma(x) = \frac{b-a}{2\sqrt{3}}; \quad \eta = \frac{\sigma}{x} = \frac{b-a}{\sqrt{3}(b+a)};
\]

Since the theoretical mean \( \mu \) is the smallest value of all the values of the considered object sign, then

\[
x_m = x_m^{\prime} - \mu.
\]

The sample with \( x_m^{\prime} \) values of the studied mechanical characteristic \( X \) is the result of rock testing. The average value \( \bar{X} \) is the statistic estimation of the theoretical mean \( x_m \). Assume a priori that the real mean \( x_m \) of the characteristic \( X \) differs from the average sampling \( \bar{X} \) due to heterogeneity of a real rock or soil mass, for which the samples under testing are structural elements. This dissimilarity is characterized by the structural coefficient \( k_c \) being equal to

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Since the theoretical mean \( \mu \) is the smallest value of all the values of the considered object sign, then

\[
x_m = x_m^{\prime} - \mu.
\]
Having divided both parts of equation (4) by $M(x)$ and considering (2), we get
\[ k_c = \sqrt{3\eta(1 - 2\eta)} + 1. \] (5)

It is clear that the structural coefficient $k_c$ depends first on relative variation $\eta$ and characterizes a level of environment heterogeneity; secondly, it depends on probability $P_{\text{in}}$, which characterizes a level of object significance.

Thus, while using strength characteristic $X$ (cohesion $C$ and angle of internal friction $\phi$) in geomechanical calculations, their natural range should be considered by introducing structural $k_x$ into the corresponding formula
\[ x_m = k_x X. \] (6)

At this stage, since there are no data on the parameters of joint systems, relative variation is taken as the one equal to value $\eta = 0.15$; probability of determination of the structural coefficient was taken as the one equal to $P_{\text{in}} = 0.9$. Then, according to (5), the structural coefficient is $k_c = 0.8$. The initial strength characteristics (cohesion and internal friction angle) are reduced according to (6) and represented in Table.

**Results.** Finite element analysis of the rock stress-strain state with the use of the elastoplastic constitutive model and the Coulomb-Mohr failure criterion allows singling out zones of the greatest rock displacements (Fig. 2). Practically, a zone of maximum displacements covers an ore layer, which has the least strength characteristics. The calculations are performed for 11 planned stages of ore body mining within the considered open pit field (western slope). Each stage involves changes in the area geometry (changes in the resulting slope angle) and rock mass structure (changes in the thickness of ore and rock layers). However, the features of displacements remain the same.

The safety factor of the open pit slope is the most significant result indicating the possibility of safe mining in terms of the specified geometrical parameters of walls. As it has been mentioned before, a “strength reduction” method is used to identify that value. A procedure of step-by-step stability reduction while using numerical methods, in particular FEM, is of special efficiency. Its essence is in the following: the solution is taken in a stage-by-stage way; at each solution stage the strength characteristics are reduced with a certain pitch and the rock strain-stress state is determined. The process continues up to the moment when a finite-element model experiences its collapse. Mathematically, it means the divergence of the iteration process in a numerical solution. When the algorithm cannot converge within the maximum number of operations preset by a user, one cannot obtain the distribution of stresses, which satisfy both the Coulomb-Mohr failure criterion (1) and global equilibrium conditions. If the algorithm cannot meet these conditions, it is considered that the model “failure” takes place. Thus, the divergence of the iteration process in the numerical solution means physically the slope collapse. A value of a strength reduction factor, in terms of which collapse in the solution occurs, is taken as the slope safety factor $F_c$.

The final result of a “strength reduction” procedure for the area represented in Fig. 1 ($1^{\text{st}}$ stage of mining) and the rocks with properties given in Table 1 is the factor of slope stability $F_c$ (strength reduction factor) equal to 1.27. A region of maximum shear strains is shown for the same stage (Fig. 3). This region determines a potential slip surface, which can take place hypothetically in terms of the strength characteristics reduction by $F$ times.

It should be noted that the model-based representation of real contours of the ore body and rock layers, experiencing their changes at each mining stage, is quite a labor-intense process. That is why initially a homogeneous rock mass was simulated; moreover, the ore properties, having the least strength properties, were assigned to the material under modeling. The safety factor for such a homogeneous model at the $1^{\text{st}}$ stage of mining is 1.35. It means that even though there are more stable layers (limestone, tuff) in the layered model (Fig. 1), the safety factor obtained on its basis and being equal to 1.27 is by 7% less. The pattern of maximum shear strains (Fig. 3) shows that the potential slip surface goes exactly along the boundary of ore and rock layers. Thus, consideration of the real rock mass structure represents less optimistic and more reliable stability evaluation, confirming additionally the efficiency of a finite element analysis, whose advantage is in the possibility to model all the features of rock mass structure and texture.

Above, it has been highlighted that the resulting slope angle, changing with the advance of operations, is the main geometrical parameter of the open pit wall. Along with the deepening of mining, the resulting angle increases, and correspondingly, the safety factor ($SF$) of a slope experiences its reduction as well (Fig. 4).

![Fig. 2. Zone of maximum total displacements (in red) at the 11th stage of mining](image2)

![Fig. 3. Area of maximum shear strains (potential slip surface at the 11th stage of mining)](image3)

| Rock          | Cohesion, KPa | Average gravity, t/m³ | Internal friction angle, degree |
|---------------|---------------|-----------------------|--------------------------------|
| Limestone     | 568           | 450                   | 25.0                           | 34               | 27               |
| Tuff          | 416           | 330                   | 26.8                           | 41               | 33               |
| Ore deposit   | 366           | 290                   | 33.0                           | 39               | 31               |

Table
The general tendency of decreasing the pit wall stability in terms of increasing slope angle is known quite well as it is based on the idea of limit equilibrium of particles on the inclined surface. However, the quantitative interpretation obtained in this research is the practical result that provides assessing certain changes in the stable state of the pit wall at different stages of mining under real geological and technical conditions. Each mining stage involves the measurement of the mined-out space and the same properties of the rock layers. The calculations are carried out for all 11 stages of ore body mining. In particular, in case of the complete three-dimension model. The calculations did not record any monotonous decrease in SF as well.

The loop-like nature of the graph in Fig. 4 stresses the expediency of finite element simulation in each specific case as the numerical method helps to consider all the geometry features of the area under analysis and structures of the environment being modeled. The indicated factors determine directly the slope stability, which is demonstrated by the relationship. The SF (equal to 1.27) obtained at the final (11th) stage is the admissible one in terms of open-pit mining; however, it shows that along with the deepening of operations, the slope state approaches the limit one. If there is more intense jointness, the bearing capacity of rocks will experience certain changes. The proposed approach allows considering this circumstance by recalculation of the structural coefficient at the expense of increasing relative variation of strength characteristics. The latter is determined depending on the distance between joints according to the methods proposed in [18].

Thus, the proposed totality of the methods is an important component of designing safe technological systems for ore mining, which is of special significance under complicated geological conditions, i.e., while developing steeply inclined seams.

It should be noted that 3D-based calculations are rather costly from the viewpoint of calculation time and necessary parameters of the computer operating system. The alternative here is to use a two-dimensional computational system to implement FEM for the analysis of one of the slope plane sections. In this context, one should remember that in this case, a plane deformation hypothesis is not true. Nevertheless, reduction of the problem dimension has its advantages while performing considerable calculation series.

For comparison, the calculations are performed for the plane section of the area in terms of specification of the same rock mass structure and the same properties of the rock layers. In general, a pattern of stress distribution, strains, and displacements (Fig. 5) coincide with the one obtained in the numerical model. The calculations did not record any monotonous decrease in SF as well. The loop-like nature of the graph in Fig. 4 stresses the expediency of finite element simulation in each specific case as the numerical method helps to consider all the geometry features of the area under analysis and structures of the environment being modeled. The indicated factors determine directly the slope stability, which is demonstrated by the relationship. The SF (equal to 1.27) obtained at the final (11th) stage is the admissible one in terms of open-pit mining; however, it shows that along with the deepening of operations, the slope state approaches the limit one. If there is more intense jointness, the bearing capacity of rocks will experience certain changes. The proposed approach allows considering this circumstance by recalculation of the structural coefficient at the expense of increasing relative variation of strength characteristics. The latter is determined depending on the distance between joints according to the methods proposed in [18].

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For comparison, the calculations are performed for the plane section of the area in terms of specification of the same rock mass structure and the same properties of the rock layers. In general, a pattern of stress distribution, strains, and displacements (Fig. 5) coincide with the one obtained in the complete three-dimension model. The calculations are carried out for all 11 stages of ore body mining. In particular, in case of the 11th (final) mining stage, the SF based on the plane model is 1.17 (as opposite to 1.27 in a three-dimensional model). The difference is caused by the previously mentioned model of plane deformation, i.e., with the neglect of one of the components of complete stress and strains tensors in the 2D model.
Чисельное моделирование стойкости борту кар'єра на основе ймовірнісного підходу

С. К. Молдаваській, О. О. Свіжукова, Д. В. Бабець, О. С. Ковров, Т. К. Аділь

Мета. Встановити закономірності зміни напружено-деформованого стану та стійкості борту кар’єру при вимірі залізної руди на різних етапах розвитку гірничих робіт при відпрацюванні крутоопадаючих пластів з урахуванням закономірних й стохастичних складових структурної неоднорідності породного масиву.

Методика. Чисельне 3D моделювання напружено-деформованого стану породного масиву, застосування процедури «зниження міцності» для визначення коефіцієнта запасу стійкості, облік неоднорідності середовища на основі стохастичної моделі. Отримані результати відображають зміну значень коефіцієнта стійкості борту кар’єра від результатуючого кута укосу.

Наукова новизна. Встановлено, що при розробці крутоопадаючих залізних рудних пластів поперечними панелями зверху вниз борту кар’єру зберігає інтенсивнішу стійкість (коефіцієнт запасу стійкості (КЗС) становить не менше 1,27) не залежачи на збільшенні кутів висоти гірничих робіт. Показано, що моделювання реальних геологічних будов і вивчають за допомогою чисельних моделей. За-пропоновано імовірнісно-статистичний підхід до врахування стійкості кар’єра від результатуючого кута укосу.

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