Journal of Engineering and Applied Sciences 13 (7): 1713-1717, 2018
ISSN: 1816-949X


# Investigations of the Deposit Geological Structure Impact on the Technogenic Accident Risk at the Mining Plant

Kaerbek Rafkatovich Argimbaev
Saint-Petersburg Mining University, 21 Line V.O., 2, 199106 St. Petersburg, Russian Federation

**Abstract:** The influence of the deposit geological structure on the risk of technogenic accidents in the mineral deposits development have been studied in the study. The technogenic accidents take place due to the growing stress-strain state of rock mass. The rock mass is a very complex environment in which deformation processes of different nature simultaneously occur with the mining operations. The monitoring of the slope deformation is very difficult task, due to the fact that the full-scale field studies are considered to be complicated, expensive and require intensive labor consumption and a long period of time. Therefore, modeling (modeling of equivalent materials, computer modeling) gives the possibility of studying such geomechanical processes that do not provide any analytical methods or observations and measurements in natural conditions. Such factors as geological and hydrogeological structure of the deposit, pit wall parameters-depth, the angle of pit wall will influence on the stress-strain state. Therefore, it is necessary to consider all of these factors to obtain a complete and accurate picture of the deformation slop in computational modeling.

**Key words:** Risks, numerical simulation, stress-strain state, glide surface, deformation, force of gravity

## INTRODUCTION

Production of mining operations causes the faulting of the initial stress state of the rock mass. Mining operations are always associated with the formation of stripped area in the rock mass or mine workings. The surrounding rocks are moved to the stripped area, the amount of the displacement is greater the closer the rocks are located to outcropping, e.g., the rock mass contained the excavation is becoming deformed. Extension strain in the direction of the workings (in the "radial" direction) is usually together with the compression deformation in mutually perpendicular directions (in the "circumferential" and "longitudinal" direction) which are usually coincide with the boundary of the workings. The tensile and compression deformation of rocks causes the additional stresses that distort or break the initial stress state of rock mass (Kholodnyakov and Argimbayev, 2014).

Around the workings a new stress-strain state is occurring, it is substantially different from the initial excavations near the workings outline and does not differ far from the outline (Anonymous, 1975). Another characteristic feature of the new stress-strain state around the excavation is a relative increase or concentration of the county normal stresses and the relative decrease or deconcentration of the "radial" normal stresses (Yakubovskiy *et al*., 2015). The stress concentration area forms so-called bearing pressure zone and the deconcentration forms the unloading area in the rock mass.

If a new stress-strain state exceeds a certain limit of the rock mass, rock breaking begins. Stress concentration or bearing pressure zone moves deeper into the mass and unload the outline area. Breaking on the area of the working outline may be in the static or dynamic manner in the form of rock bursts and blow out. But even under the static breaking of rocks they can be dangerous for the normal operation of mines because they can become unstable and cave in the workings (ASTM D5878-08, 2008). For the analysis of the geomechanical processes in the quarry the mathematical modeling taken account the deposit geological structure is used.

## MATERIALS AND METHODS

Evaluation of the stress-strain state of the one-dimensional rock mass, excluding the effect of water have been carried out on the example of diamond deposit named after V. Grib. Rock mass is under stressed condition which is formed by the weight of the overlying rocks, i.e., gravitational forces. In many cases the tectonic forces of geological processes enlarge the gravitational forces in the inside of rock mass (Yakubovskiy *et al*., 2014).

Stress state of undisturbed rock mass, i.e., before mining operation is presented with the element of standard stress at the H depth from the surface in the point of rock mass: $\sigma_X$, $\sigma_Y$, $\sigma_Z$, $\sigma_Z$ are vertical stress. $\sigma_X$, $\sigma_Y$ are horizontal stress). Vertical stress $\sigma_Z$ is caused by the weight of overlying rocks:

$$\sigma_Z = \gamma H \quad (1)$$

There $\gamma$ is specific weight of rocks. Under the vertical stresses the rock mass is shortened vertically and extends horizontally.

However, horizontal expansion of rock is obstructed with horizontal stress as the result the horizontal stress is produced. For an isotropic rock mass deformation its characteristics are the same in all directions:

$$\varepsilon_{1,y} = -\varepsilon_{2,y} \quad (2)$$

where, $\varepsilon_Y$ is horizontal deformation under the vertical load $\sigma_Z\gamma H$. Thus, under the gravity load (weight of overlying rocks) at random point of the isotropic rock mass there is no any horizontal deformation (Pikalov *et al*., 2016).

Mining operation at the pit takes place together with the initial stress breakdown of rock mass. The surrounding rock production is transported to stripped area. Deformation of tension and compression of rocks in the rock mass causes the additional stresses that break down the initial stress of the rock mass (Kholodjakov and Argimbaev, 2014).

To determine the stability of rocks around the workings it is necessary to know how to determine the stress at any area of rock mass (Argimbaev and Kholodjakov, 2013; Anonymous, 1986; EN 1992-3. 1997). The triaxial stress state is described in the elastic model of the generalized Hooke's law:

$$\begin{aligned} E\varepsilon_1 &= \sigma_1 - v(\sigma_2 + \sigma_3) \\ E\varepsilon_2 &= \sigma_2 - v(\sigma_3 + \sigma_1) \\ E\varepsilon_3 &= \sigma_3 - v(\sigma_1 + \sigma_2) \end{aligned} \quad (3)$$

Where:

E = The general deformation module (module of deformation)

v = Coefficient of lateral deformation

The aim of the numerical simulation was to study the parameters of the stress-strain state in a homogeneous side welt mass, along the lines of pit wall. Such two-dimensional problems have no not exact analytical solutions, data of on slope deformation in natural conditions are limited, so, the most effective methods of stress-strain study of the rock mass in such conditions is a numerical simulation. The mathematical modeling based on the Finite Element Method (FEM) has been applied in this case, despite some improvements of the natural conditions it has significant advantages for research opportunities (Sarma, 2012; Robertson, 2012; Kaerbek and Kholodjakov, 2016).

## RESULTS AND DISCUSSION

During the simulation a homogeneous isotropic side welt mass in which rock properties do not depend on the direction and the coordinates of sampling have been considered (Kaerbek and Maya, 2016; Kaerbek and Ivanova, 2016; Argimbaev and Alexandrovich, 2016). The model parameters are shown in Fig. 1.

The angle slope is 39,8°, the quarry depth is 450 m. Average weighted rock mass competence which added all thickness of side rock mass has been considered as physical and mechanical properties. Given the coefficient of structural weakening of the calculated physical and mechanical characteristics make up: angle of internal friction $\varphi$ 37.7°, coupling coefficient C = 0.158 MPa, relative density $\rho$ = 2251 kg/m$^3$, stress-strain modulus E = 1232 MPa, Poisson ratio v = 0.21.

As the initial data for the pit wall stability calculations based on numerical finite element method must be taken the following parameters: the characteristic of developed source schema: the size of the rock mass area investigated, the shape and dimensions of the pit walls, the original physical and mechanical properties of rocks, specific boundary conditions in the form of compressive stresses: vertically $\sigma_Y = \gamma H$; horizontally $\sigma_X = \lambda\gamma H$ where $\lambda$ is coefficient horizontal stress.

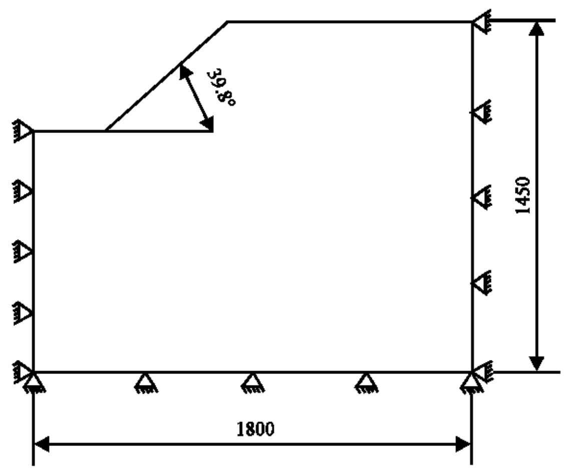


Fig. 1: Calculated scheme of the finite element model

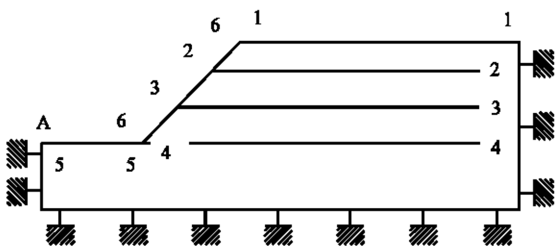


Fig. 2: Lines on which the estimation of SSS of the side rock mass

In contrast to engineering methods of pit walls stability calculation the computational modeling with the finite element method permits to evaluate the Stress-Strain State (SSS) rock mass: stress distribution, distribution of relative strain, displacement.

Among the obtained results of modeling the distribution of the relative horizontal deformation in the mass, the nature deformation identification of the wall as well as the safety factor calculation have been paid specific attention.

Safety wall factor in the calculation is n = 1.34 which is 4% above the rate calculated by the of engineering method. Safety wall factor in the calculation is which is 4% above the rate calculated by the engineering method.

The maximum critical value of the mass the SSS is determined when the pit wall was in the equilibrium state (with safety factor n = 1.05).

The particular interest in the evaluation of the stress-strain state of the distribution accounts for graphs of relative deformations and displacements along the lines shown in Fig. 2. For the lines 1-1, 2-2, 3-3, 4-4, as the zero point the point in which the vertical lines across the bow bottom of the quarry edge with the line 1-1, 2-2, 3-3, 4-4 correspondingly is accepted. For the line 6-6 as the zero point lower edge of the pit brow is accepted; for the line 5-5 is the point A. Lines 1-1, 2-2, 3-3, 4-4 are located 150 m apart each other.

Let us consider the distribution of the relative horizontal deformation along the lines 1-1, 2-2, 3-3, 4-4 (Fig. 3) when the edge of the pit position is in the limit state (with safety factor n = 1.05). Figure 6 shows that extension of side welt mass at the distance of 300 m from the top edge of pit wall take place. The local maximum, which is 100 m from the upper edge, indicates the location of the sliding wedge cleavage.

At the depth of 150 m below the surface (line 2-2) the local maximum of relative horizontal deformation is located at the distance of 150 m from the slope line is 0.016 mm/mm. At the depth of 300 m below the surface (line 3-3) the local maximum of the relative horizontal deformation is located at a distance of 100 m from the slope line and is 0.09 mm/mm.

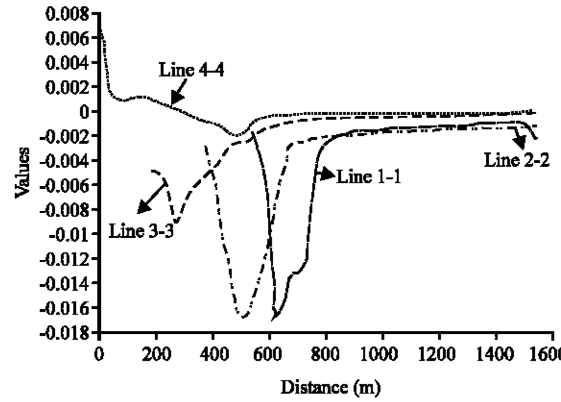


Fig. 3: Distribution of relative horizontal deformation along the lines (n = 1.05) 1-1, 2-2, 3-3, 4-4 lines on which the estimation of SSS of the side rock mass

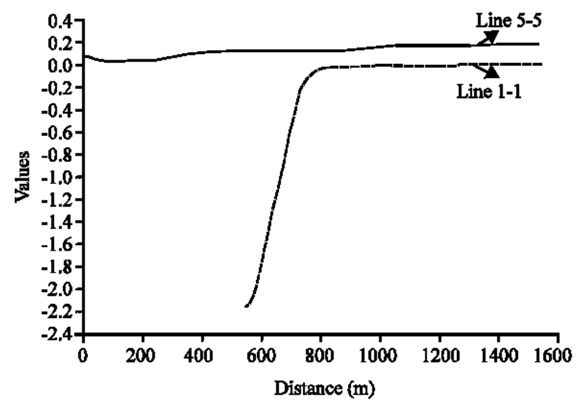


Fig. 4: Distribution of vertical displacement along the lines (n = 1.05) 1-1 and 5-5 lines on which the estimation of SSS of the side rock mass

The line 4-4 shows that near the lower edge, at a distance of 300 m there is a mass contraction due to strength and resistance from the quarry bottom. Let us analyze the dependence of the distribution shifts in the side welt mass (Fig. 4 and 5). Side welt line (line 1-1) is undergone the considerable strain and as a rule the well-defined cracks and cleavages tended to appear on it shift maximum within sliding wedge is 2.1 m the total area covered by the strains is 250 m.

Along pit bottom line (line 5-5) the maximum value near the lower edge of the side upraised to 0.1 m, the length of the upraised bottom is 150 m. Along the line of the slope (Fig. 5) there is a protrusion mass toward the working out area contrariwise the nucleus of the maximum relative horizontal deformation. Relative distribution of the horizontal deformations in board career should be noted that they are focused at a depth of 200 m from the line surface and are located on a proposed prism slip line collapse as depicted in Fig. 6.

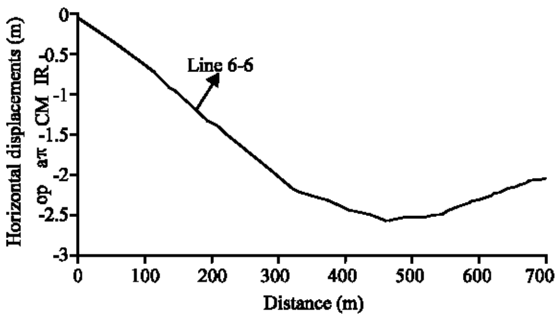


Fig. 5: Distribution of horizontal displacements along the line 6-6 (n = 1.05)

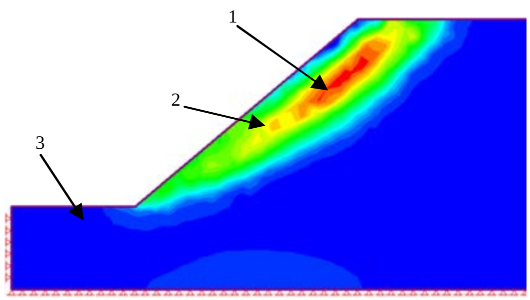


Fig. 6: The slide surface of homogeneous side welt mass 1-zone of high rock pressure; 2-zone of average rock pressure; 3-zone of low rock pressure

Summing up the results of long-term monitoring of the side welt deformations, the following signs of mass deformation, depending on their degree of stability have been found: at the stability factor $n \geq 1.3$ the welt mass undergoes mainly elastic deformation and herewith the value of the horizontal tensile strain $\varepsilon$ does not exceed $1.10^{-3}$ (1 mm/m).

At n = 1.2-1.25 it was observed the attenuation during the deformation with the value $\varepsilon = (2\text{-}5)\cdot 10^{-3}$ and total shift is 50-100 mm and more. At n = 1.10-1.15 side welt of the ground surface undergoes significant deformation with average value $\varepsilon = (10\text{-}30)\cdot 10^{-3}$ and occurring of well-defined cracks and cleavages, the total shifts are vertical and can be 1-2 m, deformation during the time is tapering out. By reducing the safety factor up to 1.05 and less over the time the side welt mass is breaking.

## CONCLUSION

In the process of mining operations in the area of the rock mass "weakened zone" is formed and characterized by the radial length of the mass contour. The degree stability of side welt mass depends on the "weak zone" parameters but so far this approach has not found the use in determining of the parameters of benches and pit stability. FEM allows us to monitor the origin and development of a weakened area.

The glide surface in the mass in the extreme position where it is observed clearly that the line glide surface of the possible sliding wedge does not extend to the lower edge of the brow and is located along the line of the slope at a distance of 0.2 H from the quarry bottom where H is the height of the pit wall. The calculation of the pit walls stability of for this rock mass glide surface are constructed in various ways, therefore there is no consensus on the construction of the most probable glide surface of sliding wedge in a uniform mass and it leads to inaccurate determination of the stability safety factor.

Data about the deformation nature of the side welt mass in mining operation and in its completion is extremely limited and does not give total information, due to the fact that the full-scale investigation requires labor coefficient, long period of time as well as they are considerable complexity. Computational modeling based on the finite element method permits us to make such an assessment.

## REFERENCES

ASTM D5878-08, 2008. Standard guides for using rock-mass classification systems for engineering purposes. ASTM International, West Conshohocken, Pennsylvania. https://www.astm.org/Standards/D5878.htm

Anonymous, 1975. Indian standard code of practice for stability analysis of earth dams. Indian Standards Institution (ISI), India. http://webcache.googleusercontent.com/search?q=cache:RSM8AklPrt4J:164.100.105.199:8071/php/BIS/IndStndrdLocatr/ShowScope.php%3Frow%3D14938+&cd=2&hl=en&ct=clnk&gl=pk&client=firefox-b.

Anonymous, 1986. Porous inorganic fillers for construction work. Russian State Standard 9758, Russia.

Argimbaev, K.R. and H.A. Kholodjakov, 2013. Determining the safe working height of an power shovel for the development of iron-bearing tailing dumps. World Appl. Sci. J., 27: 1087-1090.

Argimbaev, K.R. and I.V. Alexandrovich, 2016. Measuring the optimal focal distance to make Initial cavity in the mass of solid natural resources. Res. J. Appl. Sci., 11: 240-244.

EN 1992-3. 1997. Eurocode 2: Design of concrete structures-Part 3; Liquid retaining and containment structures. European Union Per Regulation, Europe.

Kaerbek, R.A. and B.O. Maya, 2016. The experience of the introduction of mobile crushing and screening complexes on a deposit of building materials. Res. J. Appl. Sci., 11: 300-303.

Kaerbek, R.A. and H.A. Kholodjakov, 2016. Development of numerical models for rock mass stress-strain behavior forecasting during ore deposit open-pit mining. Res. J. Appl. Sci., 11: 281-286.

Kaerbek, R.A. and M.A. Ivanova, 2016. Calculation coefficient of strength decrease of the rock mass fragments in the shotpile. Res. J. Appl. Sci., 11: 245-250.

Kholodjakov, H.A. and K.R. Argimbaev, 2014. Waste storage feasibility of concentrating mill in overburden dumps. World Appl. Sci. J., 30: 738-740.

Kholodnyakov, G.A. and K.R. Argimbayev, 2014. The choice and substantiation of the technological parameters of tailing formation in an overburden dump body. Scientific Bull. National Mining Univ., 2: 50-57.

Pikalov, V.A., A.V. Sokolovsky, V.N. Vasilets and K.V. Burmistrov, 2016. Substantiation of efficient parameters for hybrid open pitunderground mining of coal. Gornyi Zh., 1: 67-72.

Robertson, K., 2012. Regulation impact on Alaska mineral development-Tailings management. United States Department of the Interior, United States Bureau of Mines, Washington, D.C., USA.

Sarma, S.K., 2012. Stability analysis of embankments and slopes. J. Geotech. Eng., 1: 1511-1524.

Yakubovskiy, M.M., K.R. Argimbaev and M.A. Ivanova, 2014. Investigation of the quarry transfer points influence on reduction of mining operations. World Appl. Sci. J., 30: 1401-1403.

Yakubovskiy, M.M., K.R. Argimbaev and M.A. Ivanova, 2015. Constructions investigation of ore transfer points within mining limits while developing the deep pit. Intl. J. Ecol. Dev., 30: 83-87.