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Rock's instability after mining activity

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Abstract: The abandoned or insufficiently clean-upped mining activities give a high risk to disasters because:
- After many years from the stopping of the mining activities, those areas are more difficult access and is very hard to identified;
- The geography and the topography of the underground works can not be founded and the specialists leaved the affected working areas;
- For the surface mining works were forgot the underground mining works;
- The underground waters are uncontrolled, dissolve the ore and the natural pillars in the galleries, oxidizing the metals from the pillars supporting the galleries, and finally pollute many areas;
Keywords: mining activities, underground works, risk, disasters

1. INTRODUCTION

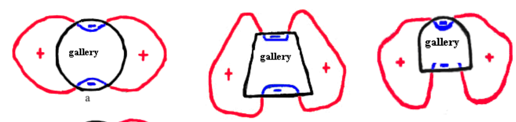
A mining activity is stabile when are respected some conditions:

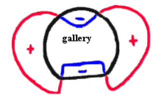
- The mining excavations are maintained open;
- Are protected to some accidentally crash down of rocks;
- The waters are controlled when get into the excavated areas and the underground waters are quickly evacuated.

Now a day, no elementary condition is no accomplished; the openings were not closed through backfilling of the external material. In this way, to the pressures created around the underground excavations is nothing which can resist. The main principle is that through the sustaining works which must make a common body with the stones and must accept all deformations will result a good stabilization of rocks. The supporting structure is determinate by the nature of rocks and the deep of works. Many mining activities were made in rocks subdued already to the perturbations resulted from old mining works.

2. CALCULATION OF THE COMPRESSION ROKS

Normally, in the massive which contain no others mining works, the tensions are equilibrated. The tensions are produces by the heavy of the superior rocks. Theoretic, the tension for a deep H can be model for an isotope rock massif homogeny through a gallery element on cube form in the mono axial solicitation conditions "Fig.1". The tensions resulted from the vertical compression on the rock cube have the tendency to be deformed in the axe "x" and "y". The geologic system is in equilibrium, the efforts δ_x





and δ_y are opponent for the deformations made by γH.

"Fig.1 Tensions in different profiles gallery"

The cube deformation after "x" and "y" (the deformation on Z axe is considerate= Ø), will be:

$$\frac{\delta_z}{E} - \frac{\mu}{E}\left(\delta_x + \delta_y\right) = 0 \qquad (1)$$

if $\delta_x = \delta_y$ si $\delta_z = \gamma H$

$$\Rightarrow \delta_x = \delta_y = \frac{\mu}{1-\mu} \times \delta_z = \frac{\mu}{1-\mu} \times \gamma H = \lambda \gamma H \qquad (2)$$

where $\lambda = \frac{\mu}{1-\mu}$ which is the coefficient of lateral pushing; The coefficient Poisson se can be evaluated to $\mu = 0{,}2$; $\Rightarrow$ $\delta_x = \delta_y = 0{,}25\ \delta_z$. The conclusion is that the horizontal tensions are smaller of four times then the vertical tensions. Increasing the excavation deep, the rocks become plastic and

$$\delta_1 - \delta_3 = \frac{1-2\mu}{1-\mu} \times \gamma H = \delta_t,$$ δ_t= the running limit

Where $\delta_1 = \delta_2$; $\delta_2 = \delta_3 = \delta_y$; for the feeble rocks μ=0,25 γ=2,5 t/m^3 and γ_t=15 daN/cm^2. The plastic rock can be found at 80 – 90 m deep. During the underground mining works the existing equilibrium will modifier, the static tensions will distribute and the dynamic tensions will appear. The most of tensions are concentrating around the digging works. Theoretic, the tension can be calculated for a circular gallery digging into an elastic rock with the equation

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for stability:

$$\delta_r = \frac{p}{2} \times \frac{m}{m-1} \times \frac{r^2 - a^2}{r^2} + \frac{p}{2} \times \frac{m-2}{m-1}\left(1 - \frac{4a^2}{r^2} + \frac{3a^2}{r^4}\right)\cos 2Ø$$

$$\delta_Ø = \frac{p}{2} \times \frac{m}{m-1} \times \frac{r^2 - a^2}{r^2} - \frac{p}{2} \times \frac{m-2}{m-1}\left(1 + \frac{3a^4}{r^4}\right)\cos Ø$$

$$\tau = \frac{p}{2} \times \frac{m-2}{m-1}\left(-1 - \frac{2a^2}{r^2} + \frac{3a^4}{r^4}\right)\sin 2Ø$$

Usually, in the galleries walls occur concentration of compressing tensions with values of 2.75 p and on ceiling and flour are present traction tensions with values of 0,25p. Those traction tensions are the most dangerous. A theoretic calculation exactly is hard to do because many times between the theoretic model and the reality in the gallery there are big difference. Some reasons: the massif is not a continually body; it is crossed by small or big fissures; the existing of some mobile rocks without cohesion coefficient; the distribution of the tensions during the excavation is not constant or uniform. For the rocks stability must study the report: $K_c\gamma H \leq R_2$ and $K_t\gamma H \leq R_1$, where:
K_c= the concentration coefficient of the compression tensions;
γ= cubical heavy, the rocks density
H= the distance to the surface
R_2= the resistance to the breakings to compression
K_t= the coefficient of concentration of the breakings tensions
R_1= the resistance of breakings to traction
δ_x, δ_y, δ_z= the tensions on the faces in the excavated gallery
μ= the coefficient of Poisson
E= the modulus of elasticity daN/ cm^2
δ_t= the running limit
r and Ø= the coordinates of the tension point
a= the work area
m= the constant of Poisson
p= the pressure in the intact rocks
The deformation in the stabile rocks exceeds not the limit of elasticity and the movement of the rocks on the edge of gallery is not more den 10 mm.
if $K_c\gamma H > R_2$ and $K_t\gamma H > R_1$
The instability phenomena will appear followed by the crossing down of the rocks because the tensions value exceeded the rocks resistance; in the ceiling and the flour of the gallery will appear deformation bigger den 200 mm. The stability can be evaluated after the report $\frac{\gamma H}{R}$ (where R= the rocks resistance):
$\frac{\gamma H}{R} < 0,25$ have place a minim deformation;
$\frac{H}{R} > 0,25$ have place an instable deformation;
The deformation appear initially in the rocks on the ceiling which will bend, the fissure will spread uncontrolled on bigger and bigger distances. The deep limits where the resistance to compression is 1000 m for granite, 300 m for schist and less den 100 for new rocks. The mine galleries are supported during the digging and exploitation with different characteristic systems. The old mine galleries (from the Dacian's or Roman's time) or the galleries which cross the massif

at exploration the metal fittings was inexistent.

"Fig.2 Gallery collapse. Boul Peak exploitation- Ruschita Locality"

Once with the increasing of the exploitation capacity was necessary to create a complex structure to support and stabilize the mining galleries. First, the wood was utilized and now a day is used too for the provisional works because of the small space and slight process. The noise and breakings forecast that the rocks pressure is very big. The frame was build by two columns and a beam. The metallic supports are now elements irreplaceable. The first supports were by rigid frames with profiles I. The simple frames with gliding elements (to permit small deformations) were modernized in 1934 in Germany, with profiles type TH (Tonissaint- Heintzmann). Those profiles had more technical characteristics: a big longitudinal moment of inertia to resist to bending; a transversal moment of inertia little superior of the longitudinal moment because the frame deformation must make only in the own geometric plan; a big polar moment of inertia to resist to rotary bending; a neutral line on the crooked side to reduce the collapsing. In underground, the steel is under the requests which rise above the limit of 50 – 58 daN/mm^2 and the breaking forecast was of 65- 75 daN/mm^2 with elastic extension of 20%. The frames were building with gliding bridles which take over small forces through controlled deformations. For the difficult conditions were utilized circular and elliptical frames. Between the frames were posted bandages with reinforced concrete or metallic bandages, nets or dash panels. The empty places behind the bandages were completed with others materials. The walls were building with reinforced or simple concrete. The workings by simple concrete used trapezoidal arch bricks. The brick lying from arch bricks allowed: rectangular supports, with right pillars and with the supporting beam on ceiling. This system was used only in the small workings for stabilization; the convex supports made by rings circular or elliptical arch bricks. Was working with arch bricks rings with distance between, the empty places were completed with steel nets or dash panels. The concrete supports increased the speed and the stability of workings. The concrete protect the rocks and against the flood of underground waters. Was used also the system of injection of concrete in alternative seals. This injection concrete system protected the walls of galleries against the mine climate. The mixed supports had combined more reinforcing technologies: anchors, nets, plates, and flat bars, nets reinforcing, and supporting frames by concrete or injection of concrete or reinforced concrete. More utilized was the supporting with anchors. Those supporting oppose to the deformations and reinforce the rocks in the affected area.

Others types of the mining workings are the vertical mine shafts which are enough complexes because cross different kind of rocks in short distances. The mine shafts collect the underground waters and must made isolation with a good liability to the stones.

Once a day with the increasing of deep, the rocks modified the resistance and need a better and bigger support of the mine shafts. For a supplementary protection on use the safety wall pillars. To understand the importance of the corrected execution of the digging workings, exploitation and maintenance, have to try a simple model. Start with the initial tensions into the massif with homogeny rocks: $\delta_z = \gamma h$; $\delta_x = \delta_y = \gamma h \dfrac{\mu}{1-\mu}$

This equilibrium is modified through the digging workings which disturb the existent equilibrium.

$$\delta_r = \delta_x\left(1-\frac{a^2}{t^2}\right);\ \delta_t = \delta_x\left(1+\frac{a^2}{r^2}\right) \qquad (3)$$

Those are Lame equations, where:

δ_r= radical tension

δ_y= δ_x lateral tensions

δ_t= length tangent

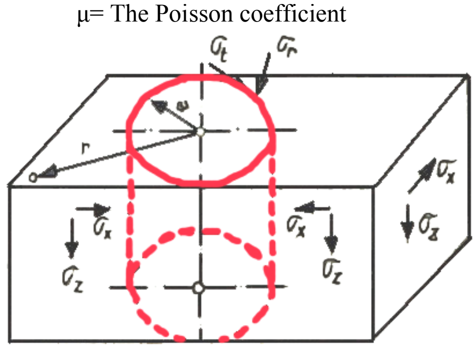
a= the mine shafts ray

If will calculate the distance from the centre of the mine shaft to the studied point „r” δ_r= 0, the tensions which will appear on the contour of the mine shafts will be “Fig.3”:

$$\delta_t = 2\delta_x = \frac{2\mu}{1-\mu}\times\gamma\times h = \delta_{\max}$$

where γ= specifically weight of rocks

μ= The Poisson coefficient



“Fig.3 Tensions in wells”

The equilibrium condition for the forces to create o bigger stability is $2\dfrac{\mu}{1-\mu}\gamma h = \delta_c$; δ_c= the resistance „r” of the rocks compression.

The conclusion is that the tangent tension on the mine shafts contour is constant and has a double value compare with horizontal tension. Also, the radial and tangent tension on the mine shafts contour is not influenced by the mine shafts diameter. $\delta_t = 2\delta_x = 2\dfrac{\mu}{1-\mu}\gamma h$ because $\delta_x = \delta_z\dfrac{\mu}{1-\mu}$, it follow that the tangent tension is a deep function and a Poisson coefficient function also. On the mine shaft contour - the radical tension δ_r=0 (reduced value)

- the tangent tension δ_t= $2\delta_x$ (double value)

The value of the radial tension δ_r increase with the deep and the distance „d” to the contour of the mine shaft, and the tangent tension decrease to the value of the existing tension in the monolith block. The refines or aquifers rocks are instable and can not ensure a good stability for the mine shafts working. For that are necessary supplementary supporting workings to ensure reaction forces bigger den the existing pressures. The shear tensions are enough equal on all mine shafts contour and express the resistance limit of the rocks and is give by the difference between the smaller and bigger values of the tensions.

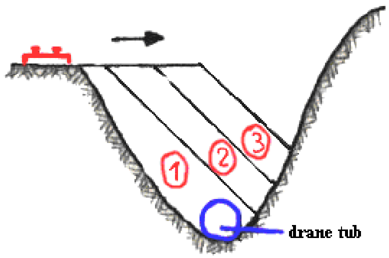
$$\tau_{\max} = \frac{\delta_1 - \delta_2}{\delta_c},$$

δ_1- δ_2 maximal and minimal values of tensions

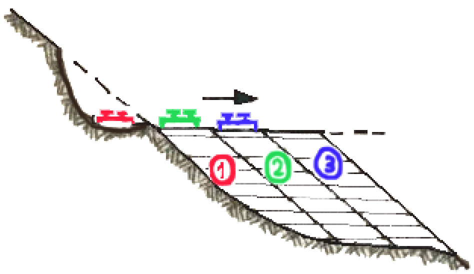
τ the value maxima of the shear tension

δ_c the resistance limit of the mono axial compressing

The rocks which build the mine shafts can be instable, aquifers, sands, refines or stables with medium hardness or big hardness. The supporting resistance is conditioned by the maximal tensions around witch can not excel the tensions of the supporting materials. No always the increasing of the materials thickness for reinforcing the mine shafts agrees with a increasing proportionally of the upward force. The supporting was made by different kind of materials: wood supporting in the areas with short time of running because are quickly damaged by water and fire and have no resistance to pressure; metal supporting is used in small sections more for reparations of the wood supporting or in provisional workings, between the frames were posted wood panels which were quickly damaged; the supporting with anchor was rare used; can be found in combination with metallic frames; the supporting with concrete prefabs small pieces were utilized in small or provisionally working; the supporting with big concrete elements was used in workings where were big flood from underground waters where were used also cast iron pipe; between those pipes was mounted rubber sealing garnitures; The supporting by concrete with continuous placement with help of gliding encasing which ensure a better stability to the external forces and which will appear because of powered into the mine shafts. The mining sloped workings are in more kinds: accessing galleries from surface to underground; connecting galleries used to transport miners, equipment, supporting materials; transporting galleries for the ore type chute or winze. At all those workings, a big importance has the joint in the connecting galleries.

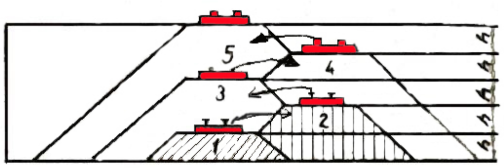


"Fig.4 Multilayer stockpiles in ungle plane"



"Fig.5 Multilayer stockpiles on horizontal in ungle plane "

The mining dumps represent an accumulation of sterile rocks resulted by uncovering or new opening are working which joint any mining activity and have different dimensions from a few cubic meters to a few hundreds cubic meters-Fig7. The studied dumps had a high content of ore because the existing technology in that time no permitted recovers all useful ore from the excavated material or by different motifs hundreds trucks with useful ore were loaded on the dumps. We had identified dumps with useful ore near by the flotation or processing station, but also dumps which follow to by transported but rested in the initial place.

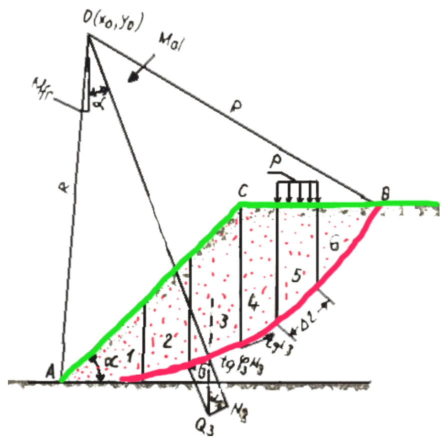


"Fig.6 Multilayer stockpiles on horizontal plane"

The dumps are located in the slopes of valleys or between the mountain slopes and can be overlapped. Usually, the external dumps have a cone or fan form. For the multistage dumps, the height of stages vary function of physic or chemic proprieties of rocks contained into the dump, method of loading of equipment and the physic or chemic proprieties of the terrain "Fig.4, Fig. 5 and Fig.6".

"Fig.7 Multilayer dump"

The verification of the dumps stability is made after the losing of stability and sliding in a cylindrical surface. Ideally for any model of calculation to be characterized by: the facility to obtain the basic information; the access to the basic information during the exploitation. Because all researches were made after 15 – 20 years after the abandonment of mine activities or after 3 – 5 years after the clean-upping, applying different methods is very complex. A facile method for study is to share the instable portion from the mountain slope in elementary sectors, the condition for stability is expressed by the report between the forces opposite to sliding and the forces moments which tends to sliding, and all moments are reference to the centre of sliding surface "Fig.8".



"Fig.8 Dump slope"

tg φ_i is the coefficient of interior friction;

C_i is the coefficient of rocks cohesion;

Δl_i is the length of sliding surface;

$N_i = O_i cos\alpha_i$ is the result Q_i of the own weight of the studied sector;

T_i = $Qisin\alpha_i$ is the tangent part of the result Q_i of the own weight of the studied sector;

α_i is the angle formed with the horizontal by Tg at the sliding arc.

Because the vegetation is small and the influence of the rain or underground waters is big, the stability coefficient will be:

$$s = \frac{\sum_{i=1}^{i=n} C_i \Delta l_i + \sum_{i=1}^{i=n} (N_i - S_i) tg\Phi_i}{\sum_{i=l}^{i=n} T_i}$$

where: S_i is the hydrostatic pressure which is present in the sliding surfaces. To ensure the dump stability this coefficient has to be unitary.

A quickly verification of the correct execution of the dump or identifying of some area which slide can be made through measuring the height of the damp and the canting of the slope and than plotting the information in the Demin curves.

H = the height limit of the damp;

h = size deduced from the graphic;

C = the cohesion of the material from the dump (tf/m^2);

γ = the density of material from the dump.

$H = h\frac{C}{\gamma}$ if will calculate the angle limit for bank sloping or the angle limit for the existing slope, will apply the relation: $h = \frac{H\gamma}{C}$

3. RESULTS AND DISCUSSIONS

To control the stability of damps is not enough only the mathematic calculation but some necessary activities also: the permanent control of the damps humidity through vertical drilling; limiting the heavy transport on the dump surface and especially on the lateral headpiece– forestry tractors, agricultural equipments; follow in time all modifications of dump through topographic measurements; the interception and guiding the underground and surface waters to avoid the waters pilling in the dump surface. Is very important relating the disaster potential to stabilize the decantation lakes or the decanters from the floatation. Usually those lakes were made under the preparation station level but there are cases when were builder over the preparation station level. Most of them were building between two slopes and were necessary the construction of a seeding dam with one decantation basin. The culvert pipe can be posted in many kinds: precipitate of slurry through a pipe with one opening; precipitate of slurry through a pipe with many opening; precipitate of slurry through a ring of pipes; precipitate of slurry through combination of all systems. To the precipitation of the slurry the drain is made on a slope which value depend of the transported material granulation, of the slurry consistence and of the loading opening. Where the slurry come out from the pipe is formed a bed in cone form which will be transformed in a bed beach. When this come to the dam level, the end of the transport pipe will be extended to the middle of lake or will increase the dam level.

The lake volume is calculated different for the water volume and for the material loaded:

$V = V_1 \times a + V_2$

where: V1 is the volume of loaded slurry;

V_2 is the volume of the water in decanter;

A is the coefficient of the rock volume increasing.

To ensure the stability of the dump steps, the sloping have to be right determinate concerning the nature of sloped material or the seeding dam. This angle can be:

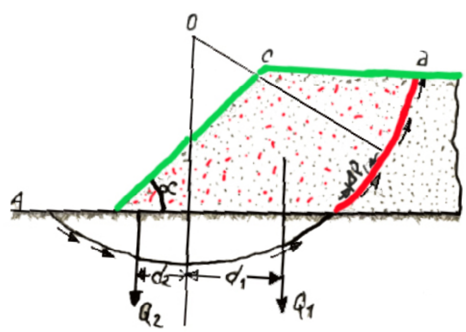
Table no. 1

slurry	dry	wet	fluid
sand	28-35	30-40	25-27
clay	40-50	35-40	25-30
vegetal soil	40	35	25
strong rocks	32-45	36-48	30-40

The slope stability is conditioned by the adapting of a convenient stability angle because of some uncontrolled infiltration forces which lead to the flushing out and loosening of the sloped material. Because of that each decanter lake has to be individually analyzed. The sloping angle is conditioned by the viscosity angle of slurry. If the curve of the infiltration water had a small height there are big chances for dislocation of dam.

"Fig.9 Sliding pond- Teliuc Locality"

The decantation lakes have to be building over a regulating pipe for the pluvial waters. For stability study of those kinds of workings have to evaluate: the maximal admitted pressure which can be taken by the terrain under the lake; the maximal task which cap be taken by the levels from the dump base to stop the pressing phenomena. The stability of the lakes can be verified in the supposition of the stability losing through the sliding after a cylindrical surface. The force moment which oppose to the backlash and the forces moments which release the phenomena are taken related to the centre of the sliding surface "Fig.10".



"Fig.10 Stability coefficient of settlig pit"

To identify the phenomena can be applied a simple method: $\frac{Q_2 d_2 + R\sum T\Delta l}{Q_1 \times d_1} \triangleright 1$

where: Q_1= the massif weight which slide; represented the active force developed on the sliding surface orientated in the sliding

Q_2= the massif weight of propping which a positive force developed in inverse sense of sliding

$\sum T\Delta l$ = the resistance to cutting which the rock has to the sliding surface which is developed in inverse sense of sliding

T = the resistance force and result from the Coulomb– Terzoghi equation: T = c + (T – n) tgφ

Where, T= the shearing resistance

c= is the rock cohesion

δ= is the pressure

n = is the water pressure

φ = is the interior friction angle

4. CONCLUSIONS

All the accidents resulted from the slurry scouring after the breaking down of the dam seeding give the conclusion that the decanters were wrong exploitation after 1990 and were not maintain. The cleaning-upping were made summary and the result is a big risk degree for the inhabitants from the downstream area.

5. ACKNOWLEDGMENT

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